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COMALCO LIMITED

EXPLORATION DEPARTMENT

E.L. 7/74 MOINA.

AREAS COVERED BY MOINA SHEETS 1,2,3.

REPORT ON ALL INVESTIGATIONS TO SEPTEMBER 1978.

P.W. ASKINS

September, 1978.

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1. INTRODUCTION.

E.L. 7/74 was first granted on 18th July 1974. On 29th October 1974 an additional 9 km^2 in the Moina area previously held as reserve by the Mines Department, was incorporated in the licence. On 1st November 1976 an additional 80 km^2 was added to the E.L., to cover further limestone areas in the Mayberry area.

Also on 1st November 1976 an additional 18 km^2 in the Oliver's Hill area was released from Mines Department Reserve (SR 1976 No.39) and incorporated in the E.L.

The total area of the E.L. is now 488 km^2 as shown on Fig.1.

This report covers all investigations carried out at Moina by Comalco since the E.L. was granted. The specific areas covered in this report lie on 5 sheets: Moina sheet 1 - Shepherd and Murphy area, sheet 1 extension, sheet 2 - Iris River area, sheet 2 extension and sheet 3 - Lea River area. Two additional separate reports will cover the Black Bluff - Smith Plains area and the remainder of the Exploration Licence (i.e. regional work).

2. EXPLORATION AIMS AND METHODS.

As a result of our exploration in North Queensland where we had located fluorite rich magnetite-skarns, it became apparent that similar rocks occurred at Moina. C.J. Gove inspected the Moina area in February 1974 (Gove, 1974) and reported that fluorite-magnetite skarn was indeed widespread at Moina. The sample results from Gove's report are incorporated in this report.

Our exploration in the Moina area has been directed at locating a fluorite orebody for Comalco's captive aluminium smelting needs. The Moina area is an old tin-tungsten mining district, so the possibility of by-product tin, tungsten, molybdenum bismuth and gold has been also assessed.

The investigations have included literature search, gridding, topographic survey, geological mapping, costeaning and rock sampling, geochemical soil sampling, ground magnetic surveys, electrical and magnetic induced polarization surveys, gravity survey, seismic trials, and diamond drilling. Each of these is covered in detail below.

3. LITERATURE SEARCH.

A thorough search has been made of all literature covering the Moina area and the E.L. as a whole. The references at the end of this report are believed to be fairly comprehensive.

4. GRIDDING AND TOPOGRAPHIC SURVEY.

Large areas have been gridded as outlined on Fig.2. The grid base line was aligned magnetic E-W and all lines were cut magnetic north south.

A topographic survey of Sheet 1 - Shepherd and Murphy area, was commissioned by Urban Planners and Surveyors. The results of their survey-data sheet and contours are plans TAS-76-53 and 53A.

Slopes read during the gridding for sheets 2 and 3 are shown on plans TAS-78-162, 163.

5. GEOLOGY.

A. REGIONAL GEOLOGY:

Regional geology of the area is covered in detail by Jennings (1963), and will be dealt with in the report on regional exploration. Two 1 inch to 1 mile geological map sheets cover the area: Middlesex, (Jennings and Burns, 1958) and Sheffield, (Jennings et al, 1959).

B. GENERAL GEOLOGY OF AREAS IN THIS REPORT:

The oldest rocks are a sequence of complex Cambrian acid volcanics, pyroclastics and clastic rocks which outcrop just east of the sheet 1 area. They will be dealt with in detail in later reports.

Unconformably overlying the Cambrian succession are Ordovician Roland Conglomerate, Moina Sandstone and Gordon Limestone.

The Roland Conglomerate is a continental or shallow marine rock composed chiefly of cobbles of quartzite with a sandy-silty matrix. It is often silicified.

Moina Sandstone is a shallow marine or near-shore deposited quartz sandstone and siltstone, which passes upwards through about 5 to 10 metres of interbedded limestone, calcareous sandstone and calcareous or dolomitic siltstone into Gordon Limestone.

The Gordon Limestone is a fairly pure bluish grey limestone with some silty layers especially near its base. In the Moina area it shows bedding disturbed by burrowing organisms, and lacks macro fossils. The fluorite skarns of the area are metasomatic replacements of the Gordon Limestone.

The Ordovician rocks have been folded probably during the Silurian into broad open folds with E-W and NW - axes, with an associated series of NW trending fractures.

During the Devonian the Dolcoath Granite was passively emplaced. It outcrops in the Forth River Valley but a gravity survey (Sheehan, 1969) indicates that it extends subsurface in an elliptical shape with Stormont at its north west edge and Campbell's Reward Mine at the south-east edge. The granite is spatially and genetically associated with various types of tin-tungsten-bismuth-fluorite and possibly silver-lead and gold mineralization in the area. It has been dated with K-Ar methods at 345 million years by McDougall and Leggo. (Webb, 1974).

Deep leads are numerous in the area, beneath Tertiary basalt. The pre basalt topography was rugged, and basalt flows plus the effects of Tertiary and Quaternary glaciation have been major influences in the formation of the present topography.

Greybilly occurs locally beneath the basalt. It resembles the Roland Conglomerate in composition and texture, but lacks joint systems, which helps to distinguish it from the conglomerate.

The sheet areas effectively cover a synclinorium containing all the known occurrences of Gordon Limestone or its metasomatized equivalents.

Prominent structures trend E-W or NW; these include both faults which are nearly vertical and gently undulating open folds with near vertical fold axes.

One major fault, the Bismuth Creek Fault, occurs on Sheet 1. This will be covered in more detail below and in other reports. A parallel fracture system exists throughout the area; there is a prominent system trending through sheets 2 and 3, which I have named the Iris River Fault Zone.

The basal parts of the Gordon Limestone throughout the area are metasomatized to some extent, but the best developments of skarns occur adjacent to faulted areas The faults and fractures have acted as the major plumbing system to enable mineralizing fluids to reach the limestone from the granite below, and the contact of Moina sandstone and Gordon limestone is a relatively minor part of the plumbing system.

The major skarn areas are at the Shepherd and Murphy Mine area, which will be dealt with in detail below, the Tea Tree Creek skarn, Stormont Bismuth Mine skarn, and possibly a poorly exposed skarn yet to be fully investigated north of Fletcher's Adit on the Lea River.

The level of erosion on the Lea River sheet area and the western part of the Iris River area is at the base of the limestone, so there are only patchy outcrops of limestone, calc-silicate rock or skarn, which correspond generally to gentle undulating folds or to downfaulting.

The thickest section of limestone is on the Iris River sheet, probably in the vicinity of the Department of Mines drill hole DOM 1, where the limestone is 178 m thick. This corresponds to the centre of the basin shaped synclinorium. Lake Gairdner covers much of this central section.

The Bismuth Creek Fault defines the known eastern edge

of the Gordon Limestone in this synclinorium, except for a complexly faulted wedge to the east of the Shepherd and Murphy Mine, which will be dealt with below.

C. MINERAL DEPOSITS:

No Cambrian or Precambrian mineralization is known in the area. All mineralization seems to be Devonian, or insignificant Tertiary placer deposits derived from the Devonian mineralization.

Vein type deposits ('lodes') were the most common type worked in the past. These typically have a quartz gangue and were worked for tin and tungsten, galena and gold. All veins are small; the largest are at the Shepherd and Murphy Mine where they average 20 to 50 cm in width and are up to 400 m long.

The granite and adjacent rocks are greisenized locally and carry uneconomic wolframite and cassiterite values in known areas.

Contact metamorphic and metasomatic deposits are widespread—minor bismuthinite-gold bearing skarn has been worked in the past (at Stormont) but the most significant newly recognized skarns are those containing fluorite.

The regional distribution of mineral deposits will be dealt with in the regional report on E.L. 7/74.

The Shepherd and Murphy Mine, where the fluorite bearing skarns occur, is described in detail below.

D. GEOLOGICAL MAPPING:

Each of the 5 sheets has been geologically mapped at a scale of 1:2500. See plans TAS-76-46, 47, 48; and TAS-77-65,66.

E. THE SHEPHERD AND MURPHY MINE AREA:

a) General

Detailed geology is shown on plan TAS-76-46; a simplified plan and section are figs 3, 4. The Shepherd and Murphy Mine, located near Bismuth Creek at Moina, has been described by various authors, the most comprehensive being Twelvetrees (1913), Blake (1956), Williams (1957) and Robinson (1957).

The main orebodies worked were four almost vertical east-west trending sub-parallel quartz veins up to 50 cm wide, 400 m long and about 100 m deep. The

veins (Nos. 2, 4, 5 and 6 lodes) carry wolframite, cassiterite and bismuthinite in a gangue of quartz with minor fluorite, mica, topaz, beryl, calcite and laumontite. Traces of scheelite, pyrite, sphalerite, chalcopyrite, magnetite, molybdenite and galena also occurred.

The veins occur partly in Moina sandstone in the south and east of the mine, and partly in overlying calc-silicate rocks and skarn to the west. The veins pass from sandstone to calc-silicates to skarn without any significant change in width or composition. The calc-silicates and skarn represent metasomatically altered calcareous or dolomitic siltstones and limestone conformably overlying the sandstone. Unaltered limestone outcrops to the west of the mine area, and is intersected in diamond drill holes put down by Comalco and the Mt. Lyell Mining and Railway Co. Ltd. (McKibben,1971).

Tertiary deep leads occur beneath basalt which covers much of the mine area. The basalt is an agglomeratic, and in part vesicular, alkali olivine type.

b) Structure

The structure of the rocks in the mine area is difficult to ascertain in detail because of lack of outcrop. Overall the contact between sandstone and calc-silicate rocks/skarns dips north-west at about 30° but there is a great deal of fracturing both parallel to the EW lode system and parallel to the Bismuth Creek Fault (Fig.5). The Bismuth Creek Fault persists regionally for several miles, and seems to have existed prior to mineralization, but to have had a small amount of strike slip movement (east block south) after the mineralization. During this movement the skarn body to the east of Bismuth Creek was separated from the main body (Fig.3.) This movement of east block south may be apparent only, because slight vertical displacement of the shallowly dipping beds could have produced the present outcrop distribution. Movement along the compressional Bismuth Creek Fault produced tension fractures trending E-W. These open E-W fractures were subsequently filled by the lodes. These fault-fracture systems doubtless were the necessary plumbing system to allow access of mineralizing fluids from the granite below.

A reverse fault dipping at about 25° NE ('Hugo's Fault') has brought Moina sandstone above the skarn body to the east of Bismuth Creek, (Fig.6).

There is an uncertain amount of folding of the rocks in the mine area. At only one point in the underground openings has a fold been mapped in the past (no mapping has been carried out underground by Comalco). At this point the sandstone - calc-silicate rock contact is an open anticlinal fold about 100 m across plunging about 50° to the west. On the Cradle Mountain road there is irregular "drag" folding adjacent to the Bismuth Creek Fault - a similar fold pattern of both E-W plunging folds and "drag" folds with axes parallel to the Bismuth Creek fault possibly persists through the whole mine area.

c) Dolcoath Granite

This granite occurs at a depth of about 200 m in diamond drill holes ML 1 and ML 2. It is extremely greisenized at the top, becoming less greisenized with depth. Where it is only partly greisenized it has a granitic texture with grainsize about 4-5mm, and now consists of 60% quartz, 20% mica (mainly phlogopite), orthoclase which is extensively sericitized and kaolinized, and oligoclase which is also highly altered. It has accessory fluorite, topaz and magnetite (Webb, 1974). A study of the greisen-granite relations from core samples is now being undertaken by Dr. T. Kwak at Latrobe University.

Unaltered granite outcropping in the Forth River Valley, 4 km east of the mine, has an average composition of potash felspar 40%, quartz 35%, plagioclase 20% and biotite 5% (Gee, 1966). The granite contains accessory fluorite and zircon (Jennings, 1963). An analysis of the Dolcoath granite is shown on Table 1.

d) Moina Sandstone

The Moina Sandstone is a well bedded sequence of pale coloured quartz sandstones and quartz siltstones with rare pebbly beds and rare shale. At the top of the sequence finer silty beds predominate over sandy beds.

In many places, particularly close to the granite, it has been indurated and partially recrystallised to a dense off white quartzite. A network of fine micro-fractures and veinlets of fine grained dark minerals (presumably biotite + chlorite + sericite + magnetite) occurs throughout, but is particularly dense at the top of the sequence. Coarser grained greisen veinlets consisting of quartz, pale micas, fluorite, traces of cassiterite, wolframite, bismuthinite, pyrite occur scattered throughout. The top 10 m of the sequence, below calc-silicate rocks, contains scattered veinlets up to 1 cm wide of pink and white felspar (adularia

and andesine, (Fander, 1976)) which carry a little scheelite.

Analyses of Moina sandstone are shown on Table 2.

e) Gordon Limestone

Unaltered Gordon Limestone is a fine grained compact pale to dark grey rock with well defined bedding which is slightly disturbed apparently by burrowing organisms. Fluctuations in grain size of the limestone define most of the bedding, but particularly towards the base there are thin beds of silty or dolomitic limestone. The bottom transition to the Moina sandstone consists of about 10 metres of limestone containing 0.5 to 10 cm interbeds of siltstone, calcareous dolomitic siltstone and limestone. Unaltered exposures of this transition zone occur in Bell Creek, near Bell Mt.

The bulk of the limestone is non-dolomitic containing less than 3% MgO, (Table 2). No sections of the limestone transition zone are particularly pyritic.

f) Calc-silicate rocks

Calc-silicate rocks of various sorts occur mainly just above the top of the Moina sandstone. They are a metasomatic alteration of the transition zone rocks between Moina Sandstone and Gordon Limestone.

The predominant calc-silicate rock is pale cream to pink, coarse grained (up to 1 cm), and in most places crudely layered - the layers being 2 to 10 cms apart and representing original bedding of the replaced calcareous dolomitic siltstones and limestone. It is composed chiefly of pinkish to brown garnet and buff to pale greenish pyroxene. Webb (1974) found the garnets to consist of about 50% - 60% of the andradite and 50% - 40% of the grossularite molecule with minor pyrope and spessartine components; and pyroxene belongs to the diopside - hedenbergite series and has a composition corresponding to slightly manganiferous salite. Also present in varying amounts are quartz, fluorite, epidote, actinolite, K-felspar, magnetite, biotite, ferrohastingsite, wollastonite, idocrase, talc and chlorite. Table 2 shows an analysis of typical calc-silicate rock. They contain scattered veinlets of adularia and andesine, similar to those in the upper part of the sandstone. In some places these veinlets are so numerous that the calc-silicate rock has a sheeted appearance.

Table 3 gives qualitative x-ray mineral determinations of calc-silicate rocks (and other skarns) from diamond drill hole SMD 6. (For locality see Fig.3.)

g) Skarns

The main skarn type in the Shepherd and Murphy area is a fluorite-magnetite skarn with a characteristic fine grained rhythmically finely layered contorted texture. It has been named by Comalco as "wrigglite" as a convenient field name. Details of the wrigglite follow in Section 6 of this report.

Sphalerite rich skarn and chlorite rich skarn occur east of the Bismuth Creek Fault in drill core. Pyrrhotite rich skarn occurs in drill hole SMD 9 on the western side of sheet 1. Details of these are in Section 7 of this report.

6. WRIGGLITE.

A. DISTRIBUTION AND SIZE:

In the vicinity of the Shepherd and Murphy Mine, wrigglite replaces Gordon Limestone for up to 70 m from its base. Wrigglite crops out close to the mine, and it is covered by basalt to the west and overlain by unreplaced Gordon Limestone in the north-western part of the area. The areal distribution can be estimated from the extent of the magnetic anomaly in Fig.5.

Contacts with calc-silicate rock are somewhat gradational; alternate layers of calc-silicate and wrigglite from 1 cm up to 3 m wide can occur near the base. Contacts with limestone are exposed in a few drill holes. They are fairly sharp; massive wrigglite gives way to massive limestone over only a metre or so; in this interval a few veinlets of wrigglite occur in the limestone. The limestone directly above the wrigglite has a tendency to be bleached and is probably recrystallized. The contact zone gives clues to the mode of formation of wrigglite; this will be dealt with below.

Only relatively minor quantities of wrigglite have been found outside the area of Fig.3.

The indicated and inferred tonnages of wrigglite based on drilling are given in Section 17.

B. TEXTURE:

Wrigglite has a distinctive rhythmically finely layered contorted texture. The textures are illustrated in Fig.7. and Appendix 3. Grain size of the common constituents varies from about 3mm down to micron sizes, and the average grain size is such that the constituents cannot be easily discerned with the naked eye or even with a hand lens. On freshly broken surfaces, especially if they are wet, the fine layered texture is nearly invisible and the rock superficially resembles basalt.

The fine layers cannot be traced for many centimetres before they either close back on themselves or are cut off by other layers. The layered sequences seem to be "unconformable" with one another. The overall impression in an outcrop is of a rather chaotic finely banded contorted structure. Some of the shapes are pipe-like with individual layers concentric but contorted; some layered sequences can be seen to have a central

fine fracture; other layered sequences commence at irregular siltstone beds. The layers are interpreted to be analogous to Liesegang Rings, formed by fluids diffusing out from a pipe-like conduit, or from a fracture, or through permeable siltstone, and replacing adjacent limestone. The origin of the layering is discussed in more detail below.

C. MINERALOGY:

Some thin sections have been examined by the author, and some by Fander (1976). The wrigglite is fine grained and mineralogically complex, and no rigorous mineralogical study has yet been completed. An honours student, T. Waite, of Latrobe University is currently carrying out more detailed studies.

The main constituents are magnetite and fluorite and one or more different silicate minerals. The most abundant silicate minerals are felspar, sericite, ferrohastingsite, green biotite, and idocrase. The sericite appears to be a replacement of the felspar.

Haphazard particles of coarser grained material include intergrowths of ferrohastingsite-fluorite-garnet-magnetite-diopside. In places pyrite is abundant partly as ragged particles replacing prior minerals.

Minor minerals include fine cassiterite (5 - 50 μm), very fine scheelite which fluoresces lime yellow under UV light, fine bismuthinite (20 μm), and patches of sphalerite.

Veinlets of quartz-fluorite, adularia-fluorite, andesine-fluorite and muscovite-quartz-fluorite-oligoclase are common in places. Scheelite is present in these veinlets as grains up to 2 mm - this scheelite fluoresces bluish white under UV light and probably has a lower Mo content than the finer disseminated scheelite. Molybdenite sometimes occurs as thin veneers at the edge of these veinlets.

No magnesium bearing fluoride (sellaite) nor fluoro-silicates like humite, clinohumite, chondrodite, norbergite have been identified. The implication is that the replaced limestone is poor in magnesium and that magnesium in significant quantities has not been introduced during metasomatism.

Fluorite is the only identified mineral species which has essential fluorine. No sellaite (MgF_2) or cuspidine

$Ca_4(F,OH)_2 Si_2O_7$ (Appendix 1) for example, has been found, but they are reported from other world occurrences of similar skarns. Tourmaline is virtually absent.

Table 4 shows the chemical composition of a bulk sample of wrigglite.

Table 3 gives qualitative x-ray mineral determinations of wrigglite in Hole SMD 6. Presence or absence of idocrase was not determined. This work was carried out by Dr. Ian Grey at the C.S.I.R.O. Division of Mineral Chemistry, but no further work by C.S.I.R.O. has been done or is planned. Appendix 2 is a mineralogical study carried out by Amdel of two pieces of wrigglite from core of ML 2. Amdel have not identified idocrase but it is probably present (microprobe work now in progress at Latrobe University confirms its presence).

Appendix 3 is a polished and thin section study by W. Fander of 4 samples of wrigglite from surface outcrops and tailings at the Shepherd and Murphy Mine.

Other information about the mineralogy of the wrigglite is contained in the beneficiation data in Section 18 of this report.

Fig.7. is a colour photograph of wrigglite and calc-silicate layers from core.

7. OTHER SKARN TYPES.

A. SHEET 1 - SHEPHERD AND MURPHY AREA:

a) Chlorite rich skarn

This occurs in core in the skarn body on the eastern side of the Bismuth Creek Fault; it occurs close to the fault in very fractured ground, particularly in ML3A. A description by W. Fander is in appendix 4.

It is interpreted to be a late stage lower temperature alteration of pre-existing wrigglite and calc-silicate rock.

Some sections of this skarn contain up to 8200 ppm Sn, much higher than the average for skarns in the area.

b) Sphalerite rich skarn

Coarse black sphalerite is disseminated through chlorite? - actinolite? rich skarn, occurring again to the east of the Bismuth Creek Fault, particularly in holes SMD 13 and 16. No thin section or polished section study has yet been carried out.

The skarn tends to occur in broken leached ground and again appears to be a lower temperature alteration of pre-existing wrigglite and calc-silicate rocks.

c) Pyrrhotite rich skarn

Significant quantities of this skarn are known to occur only in one drill hole, SMD 9, on the western side of the sheet area.

It consists of medium to coarse grained pyrrhotite, magnetite and fine grained actinolite? and chlorite? and minor very fine grained fluorite, and has an irregular wispy banded texture. The banding is nowhere near as regular as that in wrigglite. Dr. Kwak and T. Waite are presently carrying out petrological studies of this skarn, in conjunction with their work on wrigglite.

When this skarn was intersected we thought we may have found sulphide rich facies skarns similar to those at Mt. Bischoff, Cleveland and Renison. However it contains relatively low Sn values, generally about 100-900 ppm and up to 1700 ppm. Nevertheless the grade of cassiterite bearing

pyrrhotite bodies such as Cleveland is known to be patchy, so the low Sn values were not particularly daunting. Hence we embarked upon an M.I.P. survey in search of more pyrrhotite rich bodies. The results of this survey are in Section 13.

B. SHEET 2 - IRIS RIVER AREA:

Isolated patches of wrigglite occur on this sheet. They have been intersected in drill holes SMD 17, SMD 23 and are outlined by the ground magnetics. (See Section 11).

The other skarn occurrence is at Tea-Tree Creek, where some "normal" wrigglite occurs in Mines Department drill core (DOM1, DOM2, see later descriptions). Much of the material at the surface is extremely rich in raggedly distributed medium to coarse pyrite. This is interpreted to be a supergene replacement of original pyrrhotite, so that the original rock here could have been very similar to that in SMD 9, discussed above.

Tin values in this skarn are all low, of the order of 0.3% and scheelite occurs in only trace amounts (Collins, 1975).

The skarn occurs here as only a thin veneer above calc-silicate rock and sandstone; most has presumably been removed by erosion.

C. SHEET 3 - LEA RIVER AREA:

a) Stormont Bismuth Mine

The geology of this mine is dealt with in some detail in Burns (1959). This gold bismuthinite bearing garnet rich skarn occurs at 3150W/1500S. It is bounded by vertical faults trending NW, is less than 50 m wide and is probably a replacement of the transition beds at the base of the limestone. It therefore probably has no appreciable depth extent. See later section for sampling carried out here.

Another patch of similar but nearly barren skarn occurs about 50 m to the north.

About 100 m south of the Stormont Bismuth Mine there are patchy outcrops of fine grained magnetite-actinolite-fluorite rocks. These are small remnant patches of replaced limestone situated on Moina sandstone or transition calc-silicate rocks.

b) Fletchers Adit Area

Fletchers adit lies in the Lea River valley at 2750W/1050S.

The general area to the north and west of here contains poor outcrops of variable calc-silicate rocks and skarns, largely covered by what is probably only a thin veneer of basalt or basalt soil. See later I.P., magnetics, soil geochemical results. This area may have thicker mineralized skarns and warrants further investigations by costeaning/drilling.

8. RELATIONSHIPS BETWEEN SKARNS AND SURROUNDING ROCKS:

In no place does granite occur in contact with the skarns. The skarns are separated from the granite by about 150 m of Moina sandstone.

No paragenetic study of the various alteration phases in the rocks close to the mine area has been attempted, but it is suggested that because of the close spatial relationships between these altered rocks and the quartz veins (lodes), that each was formed penecontemporaneously in response to the same mineralizing fluid. i.e.

granite	altered to	greisen
limestone	→	wrigglite
calcareous siltstones	→	calc-silicate rocks
siltstones	→	metasiltstones (hornfels)
sandstones	→	quartzites and veinlet alteration

whilst quartz-wolframite-cassiterite lodes deposited in fractures.

9. SURFACE ROCK SAMPLING, ANALYSES.

A. SHEET 1 - SHEPHERD AND MURPHY AREA:

(a) Existing outcrops and dumps

Samples were collected at the existing outcrops and dumps initially by Gove (1964)[74]. Results of analyses by the Comalco Laboratory and by Amdel are in Appendix 9.

A resumé of analytical techniques used by Comalco and Amdel is in Appendix 8.

Later samples were collected by G. Weste in 1975, Appendix 10. Terminology for rock weathering is as used in the Aus I.M.M. volume "Field Geologists Manual".

Samples collected by A.H. Bartlett in 1976 are in Appendix 11. The overall reliability of these analyses by the Comalco Laboratory can be gauged from the low totals of the analyses.

(b) Costeans

3 bulldozed costeans were put in the Shepherd and Murphy Mine area to help delineate the basal parts of the wrigglite, and expose wrigglite for sampling. One costean exposed only deep basalt, and was filled in. The other two costeans were mapped (see plan TAS-78-137). and sampled. The method of sampling and analytical results are in Appendix 12. Very few samples have been analysed; samples are stored in Devonport. Samples from the western-most costean are no longer representative of the total weathered rock since they were inadvertently sieved to -80 mesh.

(c) Mines Department Samples

Two samples collected by Peter Collins of the Mines Department were analysed. The results of both the Mines Department analyses and those by Comalco are in Appendix 13.

(d) U.V. Lamping

Traverses were made at night with a shortwave UV lamp of exposures in the Shepherd and Murphy area, particularly of sandstone along the track to ML 3. This sandstone was thought to possibly contain scheelite in stringers of quartz or felspar as is observed in some drill core. Only very minor specks of scheelite were found.

(e) Significant results from this sampling

Most useful analytical information from this area is from drill core so only a few comments are necessary. Gove's sampling, Appendix 9, shows that high F (3.6%) occurs in limestone. Such high values have never been repeated in drill core, so it is likely that samples M01 and perhaps M08 have been interchanged in error by the laboratory.

The highest Sn value in the sampled wrigglites occurs in the western costean; Sn reaches 0.97%, but values as high as this do not occur elsewhere except in ML1 in mostly altered skarns. Normally Sn content of wrigglite does not exceed 0.3%.

The continuous sample in the crusher cut shows the extent of leaching of fluorite out of completely and highly weathered wrigglite; it contains only 0.4 - 3% F compared to over 7% for fresher wrigglite.

B. SHEETS 2 AND 3, IRIS RIVER AND LEA RIVER AREAS:

(a) Sampling

Some samples collected by G. Weste from the Stormont area are in Appendix 10.

Specimens of skarns and calc-silicate rocks collected during geological mapping by R. Poltock were scanned for scheelite by UV lamping and sent for analyses to Amdel. Sn, W, Bi, Be and Au were the main elements sought. Results are in Appendix 14; gold analyses are also shown on plans TAS-77-89 and 92.

Mines Department dozer cuts in the Tea-Tree Creek area (i.e. near DOM 1 and DOM 2), an old Mt. Lyell costean on the Tea-Tree Creek road, the Stormont mine open cut, and Fletcher's adit were sampled as shown on plan TAS-78-137. Selected samples were sent to Amdel for analyses. Results are in Appendix 14.

(b) Results

No significant amounts of scheelite were found by lamping the specimens.

Tin in quantities above 500 ppm occurs at the Tea-Tree Creek locality and in the area west of Fletcher's adit.

Tungsten values are all low except for one sample of 2000 ppm in skarn in the area west of Fletcher's adit.

Bismuth values over 500 ppm occur at the Tea Tree Creek locality, near Fletcher's adit, Stormont, and in the Iris River at 100W-1400S.

Beryllium values do not exceed 200 ppm in any samples.

Gold values exceeding 0.1 ppm occur only in the Tea-Tree Creek area, and in the Stormont Mine; only in the Stormont mine samples do gold values exceed 1 ppm.

The most mineralized areas are therefore Tea-Tree Creek, Stormont and the area west of Fletcher's adit. The area west of Fletcher's adit is the area with the best potential for volumetrically significant quantities of skarn, and is therefore an area where further investigations are recommended. The area near 100 W/1400S in the Iris River lies in a geochemically and magnetically active area within the Iris River Fault Zone (see later geochemical and magnetic survey results). This area requires further gridding to the east, geological mapping, sampling, etc.

10. GEOCHEMICAL SOIL SAMPLING.

A. SHEET 1, SHEPHERD AND MURPHY AREA.

Because outcropping wrigglite was obvious in the mine area and most of the prospective area is covered by basalt, no geochemical soil sampling was attempted in the eastern part of the sheet. The only area soil sampled was the far west of the sheet - this area is the Post Office anomaly, mostly on sheet 2. (see below).

B. SHEET 2, IRIS RIVER AREA.

General area:

Samples were collected over a wide area of this sheet. The aim was to detect mineralized skarns. Patchy outcrops of skarn are common on this sheet and basalt was suspected to be thin because of the presence of grey-billy in many places. Samples of C horizon were collected at 50 m spacings with the aid of a hand auger where necessary. The - 80 mesh fraction was analysed for Cu, Pb, Zn, F, Bi, Mo by the Comalco laboratory. For analytical methods used see Appendix 8. No attempt was made to analyse for Sn or W since it was thought that the other elements would be adequate pathfinders. Analytical results are plotted on drawing TAS-77-83.

All elements were found to reflect known outcropping skarns and calc-silicate rocks, but bismuth and lead are the most specific. Contoured bismuth and lead values are on drawings TAS-77-89 and 86.

Bismuth values of about 50 ppm or less are background; values range up to a peak of 1990 ppm at the Tea Tree Creek skarn locality. The bismuth content in the soil is possibly enriched compared to the underlying rock since sampled rocks in the area do not in general have such high values (see Appendix 14).

Lead values of about 50 ppm or less are background and values range up to 1580 ppm in calc-silicate rocks in the south-eastern part of the sheet.

There are no skarns detected in the geochemical survey which had not been previously found by mapping. It seems that a shallow cover of alluvium, basalt etc. is sufficient to geochemically mask skarn bodies.

The fluorite, tin and tungsten content of the skarns, and calc-silicates is not known except for sporadic rock sampling, but all the known skarn bodies are very small and the calc-silicate outcrops have no depth extent.

Therefore none are thought to be good exploration targets at this stage. A possible exception is at the far south east of the sheet, which is a complex area in the Iris River Fault zone requiring further evaluation.

Post Office anomaly:

This anomaly was originally an open ended I.P. chargeability anomaly in an area near the old Moina Post Office. It was thought that the anomaly could correspond to cassiterite-sulphide mineralization in a fracture system in limestone and was investigated by further I.P. (Section 12) and by geochemical soil sampling.

Soil samples were collected by hand auger. The soil was found to be very deep, of the order of 8 m, which was confirmed by later diamond drilling of SMD 14. Samples which were collected from shallow depths because the auger hit pebbles, cobbles etc. are therefore not regarded as reliable. The soil seems to be transported clays and some gravels lying beneath greybilly, which outcrops sporadically in this area.

The location of samples, the reliable samples, and the anomalous values, are on Fig.9. Tabulated analyses of the -80 mesh fraction and the depth of each sample are in Appendix 15.

Some of the samples are highly anomalous, e.g. 200W/250N which has 190 ppm Pb, 500 ppm Zn and 120 ppm Cu. Most of the higher Sn values occur in samples taken at unreliable depths; they are thus probably due to alluvial/eluvial cassiterite.

The geochemical data is not conclusive because of the trasported nature of the soil. It is possible that the anomalous values represent mineralization in the underlying limestone, but it is not likely in view of the results of drilling SMD 14 at one anomalous site at 100W/175N. These drilling results are discussed in detail in Section 12; it was found that the I.P. chargeability anomalies were due to lens like beds of the limestone which contain finely disseminated pyrrhotite/pyrite, and no signs of any other sulphides or mineralization were found. It is likely therefore that the anomalous soil values here are all in transported soil, possibly originally derived from weathering of skarn several hundreds of metres to the south.

C. SHEET 3, LEA RIVER AREA.

Samples were collected and analysed for the same reasons and with the same methods as for Sheet 2.

The data is on plan TAS-77-93 and contours for Bi, Pb, Cu, Zn are plans TAS-77-92, 85, 84, 94.

Again all elements were found to reflect known outcropping skarns and calc-silicate rocks. One patchy anomalous area was found which does not correspond to mapped skarn or calc-silicate rock at 2450W to 2550W/900S and 925S. Above background lead, zinc, fluorine and bismuth values occur here. Greybilly and basalt soil are mapped at this point with sandstone immediately to the north. There is no associated magnetic anomaly nearby (see Section 11) so the source is unlikely to be skarn or calc-silicate rock, and is possibly either (a) transported material associated with greybilly or (b) a leakage anomaly from vein type mineralization in the sandstone below. Vein type mineralization is a likely source since quartz veins with galena, sphalerite, chalcopyrite occur in the Lea River about 100 m north-west.

All other anomalous areas reflect known outcropping skarns and calc-silicate rocks, and as for the sheet 2 area, do not materially aid prospecting of the area.

11. MAGNETOMETER SURVEYS.

A. GENERAL.

Because wrigglite contains about 20% magnetite, magnetometer surveys are an obvious exploration tool. Our experience in North Queensland wrigglite investigations was that magnetics is the best aid to detect and delineate wrigglite occurrences.

Initially we were concerned that the overlying basalt might be significantly magnetic and either mask or confuse the signatures from wrigglite. Samples of basalt core from Mt. Lyell drill hole ML2 were sent to Geox Pty. Ltd. to determine magnetic susceptibility. This was found to be low (less than 200 c.g.s. units) in a 1 ft. 6 in. sample (see Appendix 16). Our subsequent surveys over sheets 1, 2, 3 showed that basalt throughout is relatively dead magnetically (in strong contrast to other areas in Tasmania e.g. the Highclere area).

A patchy magnetometer survey at widely spaced centres was previously carried out by Mt. Lyell. This was judged inadequate for our requirements so our surveys were carried out 'from scratch'.

All sheets were initially surveyed with fluxgate magnetometers which read the vertical component of the field; later, sheets 1 and 2 were additionally surveyed with proton precession magnetometers which read the total field with much greater precision and sensitivity.

B. SHEET 1, SHEPHERD AND MURPHY AREA.

Fluxgate survey:

This survey was carried out in 1975 using Scintrex MF2 fluxgate magnetometers. Most of the area was read at 25 m centres. Data is on drawing TAS-M-75-19A and contours are on TAS-76-51.

Extremely high readings over wrigglite were obtained in places, and the extent of wrigglite was found to be very large, the bulk occurring beneath basalt. As a rule of thumb later drilling indicated that all known wrigglite is outlined by about the 500 gamma contour. A simplified contour plan showing only 500 and 1000 gamma contours is drawing TAS-75-21.

No attempt will be made to cover the detailed interpretation of the magnetics here since it is discussed in other sections of this report (with drilling data, resource tonnage calculations, etc.). It is possible to interpret the magnetic high and low trends as pre mineralization and post mineralization fractures and folding. This has been explained briefly in Section 5 and illustrated on Fig.3 and 5.

The Bismuth Creek Fault is obvious on the magnetics - there is an apparent displacement of skarn bodies along the fault. Relative low magnetic trends are often parallel to the Bismuth Creek Fault - these are interpreted to be due to weathered or altered rocks, where magnetite is replaced by hematite-limonite, along fractures. This does not mean necessarily that the fracture is post mineralization;it may be pre mineralization and have been a favourable locus for weathering, or a favourable locus for late stage hydrothermal fluids which altered previously formed magnetite, or both.

Relative high magnetic trends often trend E-W - since these are parallel to the quartz lodes, it is assumed that they are zones close to E-W fractures where more intense metasomatism (more magnetite formation or more complete replacement of limestone by wrigglite) has taken place. They can alternatively or in addition be explained as thickening and thinning of the wrigglite by E-W trending folds which are known to occur elsewhere on the sheet.

Proton Precession survey:

This survey was carried out in 1976,77 chiefly with a Geometrics G 816 proton precession magnetometer mounted on a 3 metre high pole. Almost identical data was obtained with a less sensitive hand held model, the Geometrics G 836; none of the G 836 data is included in this report.

The area was read every 25 m along north south grid lines spaced 50 m apart. The data is on drawing TAS-78-139 and contours on TAS-77-78.

A broadly similar picture to that of the fluxgate survey was obtained; wrigglite is thought to be outlined more or less by the 62500 gamma contour. There are differences in detail however; in some places high trends are more definitive on the fluxgate, but the proton precession picture is more precise in areas of weaker total field. For example where the field is relatively weak in the vicinity of 500E/250S the proton precession contours outline a continuous high trend which is absent on the fluxgate plot - prior to the proton precession survey a fault was postulated through this point, but the continuous proton precession trend precludes its existence.

C. SHEET 1, EXTENSION.

Fluxgate survey:

Coverage of this sheet with fluxgate data is poor and no results differing from the proton precession results (below) were obtained.

Data is on plan TAS-78-138 and contours are not reproduced.

Proton Precession Survey:

This sheet area covers northern extensions of the Bismuth Creek Fault and areas where limestone is thought to be relatively thick. Magnetics were carried out to detect buried wrigglite and as an aid to interpretive mapping.

Data is on plan TAS-78-149 and contours on TAS-77-81.

No obvious magnetic high trends corresponding to wrigglite mineralization were detected. The trace of the Bismuth Creek Fault and parallel fractures can be assumed to pass through magnetic low areas.

D. SHEET 2, IRIS RIVER AREA.

Fluxgate survey:

The original survey here was at 25 m centres and extensions were carried out later at 25 intervals along lines either 50 m or 100 m apart. The data is shown on TAS-78-141 and contours on TAS-76-50.

Features east of the Iris River and north of 000N are best outlined by the proton precession survey and are discussed below.

There are small intense magnetic anomalies associated with known mapped skarns or calc-silicate rocks. Their small size is in agreement with the conclusion reached from mapping i.e. bodies are small because the erosion level is near the base of the original limestone.

There is a very active anomalous zone associated with the known relatively thin skarn at Tea Tree Creek.

North of this there are two magnetic high anomalies which trend NW. The larger is centred about 1350W/750S and the smaller is centred at about 1300W/600S. Greybilly and basalt soils are mapped here, with a sink-hole,which obviously corresponds to limestone,between the two anomalies. These two zones have not been tested by drilling. The intense nature of the anomalies and their clear NW trend suggests that they are wrigglite replacement of limestone along two NW trending fractures. Their depth extent is

unknown and they may carry high Sn, W, Bi or Au contents. It is therefore recommended that drilling be carried out here.

Proton Precession survey:

This survey was carried out on the eastern and northern part of the sheet as an aid to an M.I.P. survey conducted to locate massive pyrrhotite-cassiterite mineralization.

The area was read at 25 m intervals along 50 m spaced lines. Data is on drawing TAS-78-142 and contours on TAS-77-80. Hand drafted profiles are on file in the Devonport office.

The area east of the Iris River has several medium intensity magnetic anomalies which trend N-W or W-N-W. Two of these have been drilled (SMD 17 and SMD 23) and have been found to be due to wrigglite which presumably occurs within fractures which trend NW or W-N-W. More discussion on the magnetics here is in Section 13 which deals with the M.I.P. survey.

North of line 000N there is a broad high which continues onto Sheet 2 extension and is discussed below.

E. SHEET 2, EXTENSION.

Fluxgate survey:

This is merely an extension of work to the north of the Sheet 2 area. Data is on plan TAS-78-138 and contours are not reproduced since the proton precession data is more definitive.

Proton Precession survey:

Data is on plan TAS-78-155 and contours on TAS-77-82.

A very distinct N-E trending broad high just commences to close at the northern end of the survey and continues onto Sheet 2. The feature is very broad, with a half width of about 200 m, and its length is at least 2 km since it trends onto Sheet 3 where it is approximately delineated along tracks. It is interpreted to be due to deep rocks of the order of 200^{+} m ; these are likely to be Cambrian magnetite rich andesitic to acid pyroclastics similar to those which occur at Lorinna (see Regional report). The anomaly corresponds to aeromagnetic anomaly number 35, which was outlined by an early Mt. Lyell survey (Zarzavatjian, 1966 a).

F. SHEET 3, LEA RIVER AREA.

Fluxgate survey:

This sheet has not been surveyed with proton precession magnetometers. The fluxgate survey is a mixture of reading at 25 m intervals along 25 m, 50 m and 100 m north-south lines, as shown on data sheet TAS-78-140. Contours are on TAS-76-49.

An active magnetic zone occurs south-west of the Stormont mine and corresponds to small mapped magnetite skarns. The small high intensity anomalies here suggest that the skarns are small and shallow.

Slightly larger (areally) anomalies occur just south-east of the Stormont mine in areas mapped as alluvium, greybilly and basalt. Magnetite skarn must underlie these rocks at shallow depth. They are unlikely to be thick but the high Bi and Au content of the calc-silicate rocks at the Stormont mine could indicate that the overlying magnetite skarn at this point could be rich in Bi and Au. It is therefore recommended that the area be resurveyed with proton precession magnetics for confirmation of the anomaly, then two vertical drill holes at say 3125W/1575S and 3075W/1650S be drilled to test it.

A low order anomaly about 150 m x 75 m is centred at 2750W/1575S. Basalt soil is mapped here. It is recommended that the anomaly be checked with a proton precession magnetometer survey, and drilled if promising intersections are found at the previous two recommended drill sites.

Patchy low order anomalies in the south east corner of the surveyed area could be due to incorrect calibration of the instrument because +500 gamma readings are obtained over sandstone. The area should be resurveyed by proton precession if the surveys recommended above are carried out.

Anomalies west of Fletcher's adit are areally small which suggest that magnetite skarns are small. However non magnetitic skarns and calc-silicate rocks similar to those at the Stormont mine could be prospective here because of the relatively high Sn, W, Bi and Au values found in rock samples from this area.

Centred at 2000W/600S there is a low order anomaly, dimension about 150 m x 75 m, in an area mapped as basalt. This area lies within the Iris River fracture zone. The survey here is patchy and this area should be re-read if a proton precession magnetic survey is conducted in the area.

12. ELECTRICAL INDUCED POLARISATION SURVEYS.

A. SHEET 1, SHEPHERD AND MURPHY AREA

Preliminary:

Prior to any I.P. survey being conducted in the area, samples of core of basalt and wrigglite from Mt. Lyell's drill hole ML2 were tested for geophysical properties including density, magnetic susceptibility and metal factor. These were to be used in part as an aid to interpretation of I.P. surveys to follow. Results are in Appendix 16.

Scintrex survey:

A time domain electrical induced polarization gradient array survey was carried out in 1975 by Scintrex Pty. Ltd. The aim was to determine whether I.P. was a suitable technique for locating wrigglite, and if sulphide rich skarns occurred.

A full report on this survey by A. Howland-Rose is attached, Appendix 17, and results are on drawings TAS-78-145, 146, 147 and 148.

Electrical soundings were also carried out by Scintrex over the area of ML2, and an anomalous feature which is centred at about 500S for 200 m E and W of 200 E, and about 200 m wide. (Results also in report by Howland-Rose.)

The survey found that the skarn zones were both resistive and chargeable, but that substantial I.P. anomalies within Moina sandstone eclipsed the effects in skarn, and so to interpret whether Moina sandstone or skarn lay beneath basalt is extremely difficult.

Two possible chargeable skarn zones beneath basalt were identified, centred at 500S/200E (which was electrically sounded) and at 650E/250S. Also of considerable interest were highly chargeable zones within Moina sandstone to the east of Bismuth Creek.

Comalco investigations:

The chargeable zones east of Bismuth Creek were further investigated with electrical soundings by Comalco. Results and discussion are in Appendix 18.

The two chargeable zones beneath basalt were subsequently diamond drilled:

(a) SMD 8 was sited at 00/550S on the anomaly which was sounded. It intersected Tertiary

semi consolidated mudstone, which contains some fine pyrite, from 26 to 33 m. It is possible that a small E - W trending lens of mudstone and deep lead trending through here is the cause of the anomaly.

(b) SMD 4 was sited at 700E/240S, though not specifically to test this I.P. feature. This intersected wrigglite, calc-silicate rocks and sandstone apparently no different to intersections elsewhere where there is no associated I.P. anomaly. This I.P. feature is thus not obviously explained by the drilling. It is possible that hole SMD 4 was sited too far west and missed the main part of the chargeability trend (see profiles, not contour plans. The chargeability high on the contour plan should probably be redrawn to trend parallel to the resistivity high.) The chargeability high runs parallel to, but 25 m north of the resistivity high in this area. With the aid of proton precession magnetic contours these features are interpreted to be: the chargeability high could be Tertiary pyritic mudstones etc. in a deep lead in a basalt filled valley. All pre-basaltic rocks (skarns, calc-silicate rocks and sandstone) compared to basalt have similar resistivities so the resistivity high indicates only that here the basalt is thin, i.e. a pre basalt ridge on the southern side of the valley. There is a magnetic high at this point so it is likely that the ridge would be wrigglite at the (faulted?) contact with sandstone.

B. SHEET 2. IRIS RIVER AREA.

Post Office Anomaly:

In the 1975 Scintrex survey, open ended chargeability anomalies were found on the western edge of sheet 1. These trended E - W in limestone, and it was thought at the time that fractures, pipes etc, mineralized with cassiterite bearing sulphides as are found in Malaysia might be found here. It was also thought possible that the anomaly was due to disseminated pyrrhotite/pyrite surrounding Renison style massive pyrrhotite.

A Comalco follow up I.P. survey in 1975/76 is reported in Appendix 19, and on drawings TAS-78-150, 151. Soil sampling of this area has been reported in Section 10. It was concluded from the survey and later drilling that the anomaly is due to fine disseminated syngenetic pyrrhotite/pyrite in a lens shaped set of beds near the top of the limestone sequence.

C. SHEET 3. LEA RIVER AREA (by G. Weste).

Introduction:

A number of skarn and calc-silicate rock outcrops occur on the "Moina" sheet 3 - Lea River Area Geology sheet (drg. No.76-48). To aid exploration of the area containing these rocks a gradient array electrical induced polarisation survey was carried out. The main aims of the survey were to:

(1) detect any identifiable electrical characteristics of the skarns and/or calc-silicate rocks;

(2) locate massive sulphide (particularly pyrrhotite) and associated abundant disseminated sulphide;

(3) aid geological mapping and to detect trends under basalt.

The survey:

The survey was carried out in April 1976 by Comalco personnel using a Scintrex 2.5 kw time domain transmitter and IPR7 receiver. A 25 metre dipole was used and the distance between stations was 25 metres. Coverage of the gridded area was patchy, some lines being only partially surveyed. Coverage of skarn outcrops was incomplete, limiting interpretation of their electrical characteristics. Some instrument problems occurred due to rain but only one survey line appears to have been affected.

Results:

The resistivity and chargeability stacked profiles are shown on drawings 78-142 and 143 and overlay the Lea River Area Geology sheet. Contour plans (drg. No.78-143 and 144) were prepared before trends were determined from the profiles and this somewhat restricts their value.* The following discussion of the results is mainly based on the stacked profiles. The area generally is characterised by coincidental resistivity and chargeability

highs. Areas of lower resistivity also tend to be less chargeable. Resistivity and chargeability trends are mostly parallel and often coincident. Trends in the east of the area covered are west-north-west turning west in the central portion and returning to west-north-west again at the western boundary. Trends are disrupted by line 2800 W which contains some erroneous values. The greatest electrical contrast occurs in areas of outcropping skarn and calc-silicate rocks with chargeabilities of up to 33 millisecond against a background of 18 milliseconds, and resistivity highs of up to 7000 ohm metre against a background of about 1000 ohm metre. Resistivity highs and corresponding chargeability highs are generally located in the uppermost siltstone and sandstone immediately alongside skarn and calc-silicate rock outcrops. This is particularly the case in the Stormont gold-bismuth mine area where north-north-west chargeability and resistivity highs flank two skarn bodies. (2900 W/1420 S to 3200 W/1300S, 3000 W/1525 S to 3200 W/1350 S, and from ? 3000 W/1625 S to 3200 W/1525 S)

The major broad chargeability high from 2900 W/1000 S to 3300 W/1000 S appears to cut through a calc-silicate outcrop on line 3300 W. A series of narrow westerly trending chargeability highs in the south west corner of the surveyed area are associated with patchy skarn. These highs and the one along 1000 S are probably chargeable siltstone and sandstone immediately underlying the skarn indicating that the skarn is fairly thin. Although all the pre basaltic rocks are relatively resistive and chargeable, the sandstones and siltstones appear to be more resistive and more chargeable near the skarns and calc-silicate rocks. (6000 ohm metre and 25-30 milliseconds for siltstone and sandstone versus 1100 - 1500 ohm metre and 12 - 18 milliseconds for skarn plus calc-silicate rock). The electrical characteristics of the skarns are not easily seen as the underlying more chargeable and resistive siltstone and sandstone tends to dominate the profiles.

Basalt tends to depress the characteristics of underlying rocks and between 1200 S and 1600 S along line 2500 W the basalts appear to be fairly thick (similar surveys elsewhere indicate a resistivity of around 180 ohm metre and low chargeability for basalts).

Relatively high chargeabilities and resistivities in the north east part of the area on lines 2500 W and 2600 W are due to either siltstone and sandstone at shallow depth or to greybilly. The low resistivity at 2800 W/118 S is due to instrument malfunction.

Conclusions:

1. The skarn and calc-silicate rocks appear to be of inter mediate resistivity and chargeability. The siltstones and sandstones immediately underlying them are highly resistive and strongly chargeable and these characteristics can be used as an exploration guide.

2. Within the area covered it is likely that the Fletcher's adit skarn could continue at shallow depth to the west and east. The data indicates that if skarn does exist to the east it is probably thin.

3. The open ended anomaly in the north-west corner indicates that skarns may occur to the west of the north-west corner of the grid area. High values in the southwest corner indicate possible skarns to the west and south-west of the Stormont Mine.

4. It is highly unlikely that massive pyrrhotite occurs along the grid lines covered. The only possible conductor is at 2500 W/1200 S.

5. Induced polarisation trends in the area are west-north-west and possibly correspond to fold axes.

Recommendations:

1. Exploration needs to be carried out to the west of the grid area to cover extensions of the chargeable zones on each end of line 3300 W with an aim of locating skarns immediately adjacent to chargeability and resistivity highs.

2. A short pole-dipole traverse be conducted over the Stormont Mine area to gain more information on its electrical properties.

* Trends are shown on drawing TAS-78-168.

13. MAGNETIC INDUCED POLARIZATION SURVEY.

A. PRELIMINARY.

During the drilling program to delineate wrigglite on Sheet 1, hole SMD 9 at 150E/100S intersected 27 m of pyrrhotite-fluorite-magnetite-actinolite skarn below limestone at a depth of 91 m. It was therefore thought that Renison or Cleveland style mineralization could occur in the area. The skarn contained only relatively low Sn values (generally about 100 - 900 ppm and up to 1700 ppm), but values at Cleveland are known to be very patchy so this was not particularly daunting.

To aid the detection of massive pyrrhotite an M.I.P. survey was commissioned by Scintrex Pty. Ltd. The survey was carried out in 1977 in areas peripheral to the main wrigglite mass, since it was thought that (a) the main wrigglite mass had been sufficiently investigated by drilling and (b) that pyrrhotite skarns if they existed could occur peripheral to the magnetite skarns; theoretically sulphide rocks could occur in a lower temperature zone surrounding a core of high temperature skarns.

Notes on the M.I.P. method, its theory and application are in Appendix 20.

B. SCINTREX REPORT.

A full report by A. Howland Rose on the survey details and results is Appendix 20. Plans to accompany report are a chargeability contour plan, horizontal magnetic field contour plan and interpretation plan, drawings TAS-78-134, 135, 136.

Spectacular chargeability and horizontal magnetic field (H_N) anomalies, generally trending E-W, were obtained. In simple terms the coincident low chargeability and high H_N anomalies ('D type' anomalies) are caused by material less chargeable than the surrounding rocks and more conductive, such as massive pyrrhotite.

It was disturbing that no distinct anomaly was obtained over SMD 9.

C. COMALCO FOLLOW UP INVESTIGATIONS.

(a) Chargeable zones:

The main area of chargeability anomalies is in the same area as the "Post Office anomaly" previously covered by Comalco with electrical induced polarization (E.I.P.) and tested by drilling SMD 14. (See

Section 12). The M.I.P. chargeability zones are essentially no different to the E.I.P. zones.

These zones, if all are similar to that of the Post Office anomaly, are due to lens (?) like beds of stylolitic slightly dolomitic limestone containing disseminated pyrrhotite/pyrite within relatively clean sulphide poor limestone. No further investigation of these zones was carried out.

(b) Conductive zones:

General

Conductive zones coincident with chargeability lows ('D type' anomalies) are distinct, narrow, and trend E-W. It was thought possible that they represented pyrrhotite replacement along E-W fractures. E-W fractures are known to occur in the area: the Shepherd and Murphy lodes trend E-W.

Drilling and Electrical hole logging

One of the most significant 'D type' anomalies was drilled with hole SMD 20, at 300W/275S, vertical. A summary log is -

0 - 23m	Soil, Tertiary mudstones
23 - 38	Limestone, very broken, weathered
38 - 65	Clay, sand, rubble
65 - 69	Calc-silicate rock, metasiltstone
69 - 95	Metasiltstone, sandstone.

End of Hole.

In essence the hole penetrated cavernous limestone and clay etc. filling sink holes or caverns from the surface to 65 m. No mineralized core was recovered.

The hole below about 70 m was geophysically logged (the hole was cased from surface to 70 m preventing full logging). No resistivity low was found.

Another 'D type' anomaly was tested with hole SMD 21 at 00/75S, vertical. The eastern end of this anomaly comes close to hole SMD 9, which intersected the pyrrhotite skarn. A summary log of SMD 21 is:-

0 - 106 m	Limestone
106 - 113	Calc-silicate rock
113 - 124	Calc-silicate rock and wrigglite
124 - 155	Metasiltstone, sandstone.

The hole was geophysically logged - for details see Appendix 21. A distinct resistivity low occurs down the hole and the low intensifies with longer electrode spacing. The data are consistent with an irregular water filled cave commencing about 5 to 10 m away from the hole. A sink hole occurs about 10 m away from the collar of the hole at the surface.

The 'D type' anomaly trend containing hole SMD 20 was then again tested with hole SMD 22, at 100W/ 275S, vertical, i.e. 200 m east along the trend of the anomaly. A summary log is:-

0	- 20 m	Soil
20	- 41	Limestone with small caves
41	- 45	Clay (cave-fill)
45	- 53	Limestone, very broken
53	- 69	Limestone
69	- 74	Calc-silicate rock
74	- 78	Limestone
78	- 87	Calc-silicate rock
87	- 91	Metasiltstone and calc-silicate rock
91	- 110	Metasiltstone, sandstone.

Note that to 53 m there are many caves and broken limestone.

Conclusions

The results of drilling holes SMD 20, 21, 22 clearly indicate that the 'D type' anomalies are caused by underground cavities filled with water and wet clay; these cavities contrast markedly with the adjacent resistive and relatively chargeable limestone. The underground cavities apparently follow fracture systems in the limestone which trend E-W. No mineralization associated with these fractures was found. Tony Howland-Rose concurs with this conclusion.

Final evaluations

About the same time as this conclusion was reached a proton precession magnetic survey of the area was completed. The previous survey had been by fluxgate and not sufficiently sensitive. The relatively low order proton precession mangetic anomalies which could reasonably have a pyrrhotite source do not correspond to the M.I.P. 'D type' anomalies; this agrees with the conclusion reached above.

Drilling of further M.I.P. targets was therefore

shelved and instead one of the lower order magnetic anomalies which could have a pyrrhotite source was drilled with SMD 23 at 200W/675S, vertical. The anomaly is long, narrow and is south of areas previously tested. The hole intersected wrigglite from 6 - 19 m below basalt, but no pyrrhotite skarns.

At this stage drilling for pyrrhotite targets was abandoned. Untested magnetic anomalies exist but it is considered that they most likely correspond to relatively small wrigglite bodies, and because of beneficiation problems no further wrigglite bodies are sought at this stage.

14. GRAVITY SURVEYS.

A. CRADLE MOUNTAIN ROAD TRAVERSE.

(a) Aim:

A gravity survey was attempted to:

(i) locate any buried granite cusps which might be greisenized and carry tin mineralization;

(ii) aid geological interpretation in the complex Bismuth Creek fault zone; and

(iii) explore areas below basalt south of the Shepherd and Murphy mine area.

(b) Method:

Pegs were placed every 25 m along the road and levelled with a dumpy level. Readings were taken beside each peg with a Soden gravity meter. Data was corrected for drift, bouguer, free-air and latitude, but not topography. The profile interpretations should therefore be kept to 250 m segments.

(c) Results:

Results are plotted on drawing TAS-78-123 A, B. Interpretations of the profiles are shown on these drawings. Data and computation sheets are on file in the Devonport office.

The profile shows a broad low order trough within the Bismuth Creek fault centred at station 2600. This is where a granite cusp can on geological grounds reasonably be expected to occur at depth.

A small one station 0.2 milligal high occurs at sta 2800; if real this feature might be related to Cambrian rock at shallow depth below sandstone.

Another small one station 0.3 milligal high occurs at sta 4475; it occurs in basalt and its cause is unknown.

There are two broad troughs in basalt, centred at sta 1800 and sta 3250. Both of these can reasonably be due to thicker basalt in pre-existing stream valleys. The southern trough is at about 500S/1450E and would correspond to the southern extension of the deep lead which cuts through the Shepherd and Murphy mine area.

B. MASSIVE PYRRHOTITE SEARCH.

(a) Aim:

Gravity traverses were attempted over drill hole SMD 9 (with pyrrhotite mineralization) and over Scintrex M.I.P. 'D type' anomalies as a trial to determine whether gravity was an effective exploration tool for massive pyrrhotite.

(b) Method:

Readings were taken with the Soden meter along grid lines 100W, 00, 100E, 150E, 200E. Data was corrected for drift, free-air and latitude.

(c) Results:

Results are plotted on drawing TAS-78-122. Some interpretive notes are on the drawing. Data and computation sheets are on file in the Devonport office.

The profiles indicate that no significant pyrrhotite bodies can occur at the M.I.P. anomalies; in fact weak negative anomalies occur even over hole SMD 9 and these are consistent with sink holes/caverns in the limestone. This confirms our previous interpretations, (Section 13).

Samples of granite, greisen, basalt, calc-silicate rock, limestone, pyrrhotite skarn, wrigglite hornfels and sandstone from core were sent to the Mines Department for density measurements in case densities were needed for gravity modelling. Data is in Appendix 22. No modelling has yet been carried out.

(d) Conclusion:

The gravity method is judged to be an effective method of detection of heavy bodies like pyrrhotite but magnetics are a quicker easier and cheaper method. Gravity surveying could be a useful tool to detect skarn bodies which have no associated magnetite,pyrrhotite, or other sulphides i.e. those skarn bodies which would not be easily "seen" by I.P. and magnetics.

15. SEISMIC AND RESISTIVITY TRIALS.

Trials of resistivity and reflection and refraction seismic were attempted over previously drilled basalt covered areas on sheet 1 and over Department Mines drill hole DOM 1 on sheet 2.

The results and conclusions are in Appendix 23.

16. DIAMOND DRILLING

A. PRELIMINARY

Initial drilling on the sheet areas was carried out by the Mt. Lyell Mining and Railway Co. Ltd. in 1970, 71. The Mines Department drilled 3 holes in 1972-73. Comalco commenced drilling of the Shepherd and Murphy Mine area early in 1976.

B. MT. LYELL DRILLING

Mt. Lyell's drilling consisted of 3 holes ML1A, ML2, ML3A, angled to test extensions of the known lodes at the Shepherd and Murphy Mine, (McKibben, 1971). The sites of the holes are plotted on the geology plan, TAS-76-46.

No significant intersections of mineralized quartz veins were found. Wrigglite was intersected in ML2 and ML3A but it was not recognized as a fluorite rich skarn and it was not split and analysed. Greisenized granite was intersected in ML1 and ML2 at about 200 m below the surface.

The Mt. Lyell core was kindly made available to Comalco and has been logged, split and analysed. Logs are in Appendix 24, analyses in Appendix 25. The core is now stored at our Devonport base.

C. MINES DEPARTMENT DRILLING

Three holes, with a total depth of 521 m, were drilled on sheet 2 area to test mineralized skarns. Full details of core logs, analyses etc. are in Collins, (1975).

Pulps of the skarn intersections were given to Comalco so that analyses for fluorine and other elements could be carried out by the Comalco laboratory. Permission was also granted to sample other sections of the core. Analyses by both the Mines Department and by Comalco are in Appendix 25.

D. COMALCO DRILLING

(a) Methods:

All drilling was carried out with Comalco personnel and equipment. The drill was a Joy 22HD mounted on a 6 x 6 GMC truck.

The first Comalco drill hole was labelled SMD 4, the number being consecutive to the 3 prior holes of Mt. Lyell.

All core was logged, marked for splitting, photographed, split, analysed and stored at the Devonport base.

Many holes were scanned with a McPhar TV1 scintillometer to check for radioactive minerals but none have been found.

All holes except SMD 14 were logged by P. Askins. P.W. Stainton logged SMD 14 (and logged several other holes in addition, with sedimentological emphasis),(Appendix 27).

(b) Aims:

Initial drilling was aimed at delineating the extent and grade of the wrigglite. Most holes were therefore sited as close as physically practicable to magnetic highs.

Later drilling was designed to test M.I.P. anomalies which were thought to possibly be due to massive pyrrhotite-cassiterite mineralization.

(c) Results:

Full geological logs and analyses are given in Appendices 24 and 25. Recovery logs are in Appendix 26. Comments on all Moina holes, including Mt. Lyell and Mines Department drilling, are listed below:-

DOM 1: 360W/100N, vertical, depth 325 m. Is probably sited where the sequence of Gordon Limestone is thickest. Calc-silicate rock with only very minor wrigglite is at the base of the limestone.

DOM 2: 1340W/860S, vertical, depth 101 m. Sited in outcropping skarn. Magnetite-fluorite skarn, some pyrite and pyrrhotite rich, plus calc-silicate rocks to only 15 m.

DOM 3: 1200W/940S, vertical, depth 95 m. Sited on basalt, to 10 m. Magnetite skarn virtually absent, only calc-silicate rock to 23 m. Magnetite skarns probably stripped off here by erosion.

ML 1A: 1170E/070N, 50° towards 180°M, depth 265 m. Commenced probably in the Bismuth Creek fault zone in altered sandstone. No skarns intersected. Greisenized granite intersected at 225 m. Extra analyses and petrological studies of greisenized granite are being carried out at present:- will be included in a later report.

ML 2: 910E/150S, 50° towards 180°M, depth 335 m. Intersects wrigglite limestone partly replaced by wrigglite at about 168 m; this is one of the few intersections showing partial replacement relationships. Greisenized granite intersected at 331 m. No significant mineralized quartz veins.

ML3, ML3A: 1450E/125S, 50° towards 000°M, depth 260 m. Hole collared in sandstone, drilled through Bismuth Creek fault into wrigglite. Wrigglite in places is chloritic and relatively poor in fluorite and magnetite; here fluorine analyses are low. A petrological description of this rock is in Appendix 4.

The highest Sn analyses in core are in this hole; some sections richest in Sn might be due to chloritic alteration of pre-existing wrigglite; higher Sn values tend to correlate with low F values, Fig.10. Sn values are up to 8200 ppm; analyses of separate intervals are not continuous, but complete bulked samples were analysed. Bulk analyses over the full intersection of wrigglite are:

342 - 377.5 ft. i.e. 35.5 ft. (10.8m), 2750 ppm Sn, 10.0% F
377.5 - 403.5 ft. i.e. 26.0 ft. (7.9m), 4200 ppm Sn, 2.3% F
403.5 - 478 ft. i.e. 74.5 ft. (22.7m), 2500 ppm Sn, 11.4% F

SMD 4: 700E/240S, vertical, depth 109 m. To test one of the outer limits of the main magnetic anomaly. Sited on basalt. Basalt and Tertiary sediments to 68 m. Patchy wrigglite interbedded with calc-silicate rock to 92 m.

SMD 5: 370E/130S. vertical, depth 81 m.
To test the area well to the west of the Shepherd and Murphy Mine, at a point where magnetic anomaly is relatively weak. 22 m of basalt and Tertiary sediments, then limestone to 46 m showing partial replacement to wrigglite near base, then interbedded wrigglite and calc-silicate rock to 61 m.

SMD 6: 860E/50S, vertical, depth 102 m. To test area covered by basalt close to Shepherd and Murphy Mine where magnetic anomaly is pronounced. 21 m of basalt, then wrigglite with some interbedded metasiltstone to 83 m.

SMD 7: 975E/90N, vertical, depth 71 m. Area of outcropping wrigglite in Shepherd and Murphy Mine area, pronounced magnetite anomaly. 45 m of wrigglite, with some interbedded wrigglite and calc-silicate rock to 65 m.

SMD 8: 00/550S, vertical, depth 61 m. To test I.P. chargeability high trending east. Basalt and much Tertiary sediments to 32 m, then wollastonite hornfels, calc-silicate rocks and only a few cm of wrigglite to 46 m. I.P. anomaly as explained in Section 12 possibly due to Tertiary deep lead sediments.

SMD 9: 150E/100S, vertical, depth 130m. Designed to further test the relatively weak magnetic anomaly on which SMD 5 is sited. Intersected limestone to 91 m, then pyrrhotite-magnetite-fluorite-actinolite skarn to 118 m; then fault zone with coarse fluorite to 125 m, sub parallel to core. Pyrrhotite skarns are crudely layered, wispy and contorted. Sn values disappointingly low:- Maximum 1700 ppm, generally about 100 - 900 ppm.

SMD 10: 700E/50N, vertical, depth 117 m. To test northern part of main magnetic anomaly. Basalt and Tertiary sediments to 39 m, then wrigglite variously interbedded with metasiltstone and calc-silicate rock to 83 m. Some mineralized quartz veins intersected.

SMD 11: 750E/100S, vertical, depth 120 m. To test central part of main magnetic anomaly. Basalt with mudstone interbeds to 31 m, then wrigglite interbedded with various metasiltstone and calc-silicate rock to 69 m.

SMD 12: 450E/50N, vertical, depth 123 m. To test an area with no magnetic high, to determine rock types, at NW perimeter of the main magnetic anomaly. Basalt with underlying deep lead sediments to 40 m, then cavernous limestone to 91 m, then relatively minor wrigglite interbedded with calc-silicate rock and metasiltstone to 109 m.

SMD 13: 1450E/50S, vertical, depth 182 m. To test the magnetic anomaly to the east of Bismuth Creek. Anomaly was puzzling since sandstone occurs at the surface.

Sandstone to 84 m, then through a fault into skarn to 113 m. Skarn is very rich in sphalerite in places; up to 27.5% Zn (89.10 to 90.00) and averaging 8.6% Zn over 16.65 m from 87.35 to 104.00 (see Table below). This skarn seems to be replacement of pre-existing calc-silicate rock and wrigglite. Below skarn there is calc-silicate rock to 124, then sandstone.

SMD 16 was drilled into the same anomaly and established that the fault is a flat lying reverse fault, where older sandstone has been brought over the top of limestone or skarn. It is not known whether this reverse fault predates or post dates mineralization.

From	To	Int.	% Zn	Int x Zn
87.35	88.50	1.15	6.30	7.25
88.50	89.10	0.60	13.80	8.28
89.10	90.00	0.90	27.50	24.75
90.00	91.00	1.00	14.50	14.50
91.00	92.00	1.00	5.25	5.25
92.00	93.00	1.00	12.50	12.50
93.00	94.00	1.00	11.50	11.50
94.00	95.00	1.00	8.50	8.50
95.00	96.00	1.00	13.00	13.00
96.00	97.00	1.00	5.40	5.40
97.00	98.25	1.25	0.43	0.54
98.25	99.25	1.00	1.31	1.31
99.25	100.25	1.00	7.75	7.75
100.25	101.25	1.00	10.40	10.40
101.25	102.25	1.00	7.00	7.00
102.25	103.25	1.00	4.65	4.65
103.25	104.00	0.75	1.63	1.22
Total 16.65 m			wt.av. 8.63%	143.76

SMD 14: 100W/175N, vertical, depth 194 m. Not logged by PWA, see graphical sedimentological log of P.W. Stainton. To test E.I.P. chargeability anomaly (Post Office anomaly).

Limestone from surface to 192 m, then wollastonite hornfels. Hole if continued would probably have penetrated calc-silicate rocks from this point. E.I.P. anomaly explained by fine pyrite, pyrrhotite associated with stylolitic limestone and thin dark dolomitic beds concentrated in top 60 m of hole (see Section 12 for details and some petrological descriptions).

SMD 15: 900E/018N, vertical, depth 116 m. Designed to test a relative magnetic low trending NW within the main magnetic anomaly. Thought to be due to faulting and late stage alteration of pre-existing wrigglite, with possibly high Sn values as for example in ML 3A.

Hole was probably sited too far south and missed fault zone (if it exists). It is recommended that this feature be drilled again with an angled hole.

Basalt to 20 m, then deep lead sediments to 33 m, then calc-silicate rock to 109 m, with metasiltstone below.

SMD 16: 1450E/125S, vertical, depth 171 m. To further test the area around SMD 13. Sandstone to 24 m, then through a fault into calc-silicate rocks, wrigglite and some patchy sphalerite bearing skarns to 129 m. Tin contents higher than SMD 13, but not as high as ML3A. Tungsten, occurring mainly as scheelite disseminated in the wrigglite or in the relatively common pink felspar veins, is relatively abundant in this hole.

The four holes which have penetrated the skarn zone to the east of Bismuth Creek i.e. ML3A, SMD 13, SMD 16, SMD 24, have shown that the body is complex, in part Sn rich, in part W rich, in part Zn rich. See Fig.6, which is a geological cross section.

SMD 17: 810W/060S, vertical, depth 74 m. To test magnetic anomaly well away from Shepherd and Murphy mine. No surface outcrop. Intersected interbedded sandstone and limestone to 26 m, then interbedded calc-silicate rock, normal wrigglite, metasiltstone to 42 m thence through into sandstone-siltstone. Sn and W contents quite low.

Holes 18 and 19: drilled at Mt. Jacob (see separate report).

SMD 20: 300W/275S, vertical, depth 95 m. To determine source of an M.I.P. 'D type' anomaly which could feasibly have been due to massive pyrrhotite-cassiterite mineralization similar to that at Renison. For details see Section 13. Intersected cavernous broken limestone to 53 m, thence calc-silicate rocks to 70 m, with siltstone sandstone below. No pyrrhotite rich skarns.

SMD 21: 000/075S, vertical, depth 155 m. To determine source of another M.I.P. 'D type' anomaly. Limestone to 106 m, thence calc-silicate rock with minor normal wrigglite

to 124 m, thence metasiltstone. No pyrrhotite rich skarns. Hole geophysically logged; shows proximity to caverns, for details see Section 13.

SMD 22: 100W/275S, vertical, depth 110 m. Final hole to test M.I.P. 'D type' anomalies. On same anomaly as SMD 20 but 200 m to the east. Cavernous limestone to 69 m, thence limestone and calc-silicate rock, with a few cm of wrigglite at the base, to 91 m; metasiltstone-sandstone below. No pyrrhotite rich skarns.

Holes SMD 20, 21, 22 demonstrated that the cause of the M.I.P. anomalies is not massive pyrrhotite mineralization but is due to non chargeable conductive wet clay etc. in sink holes and underground caverns within relatively resistive and chargeable limestone. Full details in Section 13.

SMD 23: 200W/675S, vertical, depth 37 m. To test long narrow relatively weak magnetic anomaly south of areas previously tested. Basalt to 6 m, thence interbedded wrigglite calc-silicate rock and metasiltstone to 27 m, thence metasiltstone. Anomaly therefore due to normal wrigglite, not pyrrhotite rich skarns.

SMD 24 SMD 25: 1380W/010S. These are holes sited east of Bismuth Creek angled towards the fault, designed to further test the complex skarn zone, then explore for vein stockworks associated with the fault and mineralized greisenized granite. Very difficult drilling conditions were encountered and the holes were abandoned before target depths.

The holes have not yet been logged in detail nor analysed; they will therefore be reported on in a later report.

17. WRIGGLITE TONNAGES AND GRADES.

A. GENERAL.

A preliminary calculation of the tonnage and grade of wrigglite at Moina was made in September, 1976. It was based on all available information to that date, and therefore does not include information gained from any drilling after SMD 12.

Revised calculations will be carried out and reported on soon.

The preliminary results should be an approximation of the revised results for the main wrigglite area, but will differ for the area east of Bismuth Creek, where holes SMD 13, 16 have necessitated major changes in the geological sections.

B. METHODS, ASSUMPTIONS.

(a) Plan and sections:

The area of wrigglite, which approximately is outlined by the +500 or +1000 gamma fluxgate magnetic field contours, but modified by the drilling information, is shown on drawing TAS-76-57. This area was divided into blocks outlined by north-south sections, as shown.

Geological sections were compiled by using the best fit of information from outcrop geology, drill hole exposures, underground mapping by the Mines Department, and interpretations of the fluxgate magnetic data.

(b) Tonnage calculations:

Tonnage calculations for each block are shown on the geological sections, drawings TAS-76-55 and 56. Tonnages are quoted as "indicated" in accordance with Aus. I.M.M. usage. The degree of faulting and folding in the prebasalt rocks and the rugged pre basalt relief make accurate "measured" tonnage calculations impossible. Closer spaced drill holes will be necessary to obtain "measured" tonnages. The "indicated" tonnage requires testing by further drill holes before it can be quoted with confidence.

The "inferred" tonnage is from the area east of Bismuth Creek. At the time of calculation geological information was very sketchy here and hence figures could only be quoted as "inferred". Drilling of holes SMD 13, 16 has since necessitated major revision of this "inferred" figure, which should therefore now be ignored.

Amongst the assumptions in the calculations are:-

(i) only open pittable wrigglite calculated. Where the overburden/wrigglite ratio exceeds about 2.5:1, the wrigglite is not included in the calculation.

(ii) density of wrigglite 3.3

(iii) density of overburden 2.5 (basalt, some limestone some sandstone.

(iv) open pit slope 60°.

(v) In many places there is a grade boundary between wrigglite and fluorite bearing calc-silicate rock. Only mineralized rock with an approximate bulked average analysis of greater than 10% CaF_2 has been included in the calculation.

(c) Grade:

The grade of wrigglite is approximate only, since only six drill holes have penetrated "indicated" wrigglite. Table 6 summarizes weighted average grades. Average grade for the whole wrigglite deposit is visually estimated from this table.

TABLE 6

WEIGHTED AVERAGE ANALYSES

DIAMOND DRILL HOLES, MOINA, WITHIN "INDICATED" WRIGGLITE

Hole No.	From metres	To metres	Interval metres	CaF_2 %	Sn ppm	W ppm	Remarks
ML 2	43.3	50.4	7.1	17.8	ca 800	ca 400	
	50.4	61.0	10.6	11.6	ca 1400	ca 150	
	61.0	82.9	21.9	19.2	ca 1500	ca 800	
	43.3	82.9	39.6		1350	540	bulked sample analysis
SMD 6	22.75	83.05	60.30	17.6	1400	910	
	22.75	67.70	44.95	19.5	1540	930	
SMD 7	1.00	55.00	54.00	20.9	1450	1030	
SMD 10	38.75	88.50	49.75	17.6	1320	535	0.35 interval of rich vein
	38.75	76.00	37.25	19.2	1300	500	excluded
SMD 11	31.00	74.00	43.00	16.9	1525	1020	1.35 interval of nil core recovery
	31.00	60.70	29.70	19.4	1540	1010	excluded.
SMD 15	33.00	105.00	72.0	18.0	1470	1250	

NOTE: All intervals are approximate true thickness.

C. SUMMARY.

The total wrigglite resource is

"indicated" tonnage	15,500,000
grade	approx. 18% CaF_2, 0.1% Sn, 0.08% W

Overburden over "indicated" wrigglite 11,200,000 tonnes

Ratio "indicated" wrigglite to overburden 1.4: 1

(Data for inferred wrigglite is not reliable and so is not quoted here).

The deposit is probably the largest single fluorite resource in Australia.

18. BENEFICIATION STUDIES.

A. GENERAL.

Beneficiation studies have been carried out on wrigglite by The Australian Mineral Development Laboratories (AMDEL) in Adelaide and by the Tasmanian Department of Mines in Launceston.

Amdel initially did work on chemical extraction of the fluorite from surface samples of wrigglite and subsequently carried out mainly flotation testing of a composite sample from drill core.

The Department of Mines has investigated, and is continuing to investigate, fluorite, tin and tungsten extraction on a bulk sample of wrigglite collected off dumps at the Shepherd and Murphy mine.

B. AMDEL INVESTIGATIONS.

(a) Chemical extraction of fluorite:

Aim and method

Amdel offered in late 1975 to try a roasting process to extract fluorite from Moina wrigglite. They had previously used their method in an ore that had not given acceptable flotation results.

The exact method remains confidential to Amdel, but is believed to be a roast at 400°C with ammonium sulphate, and collection of gaseous fluorine compounds. Details of what is probably the method used are at the end of Appendix 28. (This method was published in Amdel report No.1178 in 1977).

Two samples, Moina 1 and Moina 2 were submitted to Amdel. Both are samples collected by A.H. Bartlett of wrigglite from outcrop adjacent to the old Shepherd and Murphy mill site.

Results

Only sample Moina 1 was tested. Full results are in Appendix 28.

Extraction of fluoride was 70%, but this initial work did not attempt to modify and optimize the method. The tin remained in the residue and so was not extracted.

The high iron content of the wrigglite caused a major wastage of the reagent, so no further work was carried out with this method; all further work was by more conventional

flotation means.

(b) Flotation extraction of fluorite:

Aim and methods

The aims were:

(i) to produce an acid grade fluorite concentrate at maximum recovery;

(ii) to study the feasibility of extraction of magnetite, cassiterite and scheelite.

A quotation was received from Amdel and the Colorado School of Mines for an initial beneficiation study of the Moina wrigglite. Amdel's quote was accepted because

(i) it was cheaper;

(ii) they had previously carried out beneficiation work on the Mt. Garnet wrigglite; and

(iii) liaison with Amdel as work proceeded would be easier.

A bulked sample of crushed split core from drill holes SMD 4, 5, 6, 7 was made up according to a recipe, as listed in Appendix 29. The sample contains both wrigglite and calc-silicate rock. The overall fluorite content was 16%, and Sn was 0.13%, WO_3 - 0.11%.

Full details of the mineralogical studies, magnetic separations and flotation testing are in Amdel's final report No.1147 (Feb 1977), in Appendix 29. (Data from Amdel's preliminary report and progress reports 1 and 2 are all covered in the final report and so only the final report is reproduced here).

Results

Full results of testing are in Amdel's final report in Appendix 29.

Amdel's final conclusions are:

(i) By fine grinding and flotation a concentrate assaying 92.5% CaF_2 with a recovery of 64.9% was produced from a feed assaying 16.5% CaF_2.

(ii) Removal of magnetite by magnetic separation applied to either the flotation feed or concentrate resulted in a high loss of fluorite due to the poor liberation characteristics of magnetite.

(iii) Grinding to 17 μm is necessary for good liberation of fluorite. Scheelite is well liberated at flotation sizes but little cassiterite is liberated above 9 μm. Much of the magnetite is liberated at 33 μm but finely disseminated magnetite is evident in grains finer than 15 μm.

(iv) The chief contaminants in the flotation concentrate are magnetite, calcite, scheelite and amphibole. Further flotation cleaning stages should reject calcite and amphibole, but the magnetite occurs as fine inclusions in the fluorite. Differential flotation of the scheelite from the fluorite has not been studied.

(v) It is unlikely that a high grade magnetite concentrate suitable for heavy medium use could be recovered from the ore because of the poor liberation characteristics of this mineral.

(vi) The composite ore sample contains 0.13% Sn, 45% of which is present in solid solution in garnet. The remaining tin is present as cassiterite which is too finely disseminated for physical beneficiation.

(vii) The composite ore sample contains 0.11% WO_3 as scheelite. This scheelite concentrated to 0.4% WO_3 with the fluorite during flotation but it is doubtful whether the scheelite could be recovered economically from this concentrate unless chemical treatment was carried out.

Discussion and recommendations

Amdel believe that it may be feasible to treat the Moina wrigglite by flotation to produce a low-grade concentrate at high recovery and then leach this concentrate to extract fluoride. They recommended that sufficient work be carried out to enable a preliminary assessment of the technical and economic feasibility of this process. This work has not been carried out.

My comments are:

(i) The bulk sample which Amdel investigated included some calc-silicate rocks from holes 4 and 5; these calc-silicate rocks are not within the "indicated" wrigglite of the resource tonnage calculations (Section 17). This can throw a bias on the bulk sample by changing grades, minerals present etc. The bulk sample when it includes calc-silicate rock is probably more representative of the deposit as a whole, but the initial investigation would have been simpler if only pure wrigglite had been studied.

(ii) Amdel have found that about half the tin is in solid-solution in garnet. I suspect that most of this garnet comes from the calc-silicate rocks and relatively little from the wrigglite (See table 3). Much of the calc-silicate rock occurs at the base of the wrigglite

so with selective mining calc-silicates can probably be largely eliminated, and a relatively pure wrigglite product mined. This relatively pure wrigglite possibly would have more tin in a recoverable form.

To resolve the tin problem work needs to be done on a purer bulk sample of wrigglite. Such a bulk sample was prepared for investigation by the Department of Mines.

(c) Pyrohydrolysis

Amdel have a method of extracting fluorite by pyrohydrolysis a process which yields HF using steam as the only reagent. When approached by us they suggested that trials be carried out. (Appendix 29A). No such work has yet commenced but it is recommended.

C. DEPARTMENT OF MINES INVESTIGATIONS.

(a) Exploratory project, investigation R 731:

Sample type and analyses

A sample of wrigglite uncomplicated by the presence of calc-silicate rocks was submitted to the Department of Mines for investigation. The sample was a 140 kg parcel of wrigglite made up of cobbles and boulders selected off the dump about 50 m NNE of the old main shaft of the Shepherd and Murphy Mine, and off a bulldozer costean spoil heap immediately adjacent to the old main shaft.

An analytical comparison of the bulk samples submitted to Amdel and the Department of Mines is:

	Amdel sample.	Mines, (R 731).
CaF_2	16.1%	20.0%
Sn	.13%	.22%
W	.08%	.14%

Full lists of available analyses of the Mines bulk sample are in Appendix 30. Analyses by Comalco laboratory, Amdel and the Mines Department are listed. The Mines Department analysis for fluorine (7.8%) is much lower than that of both Amdel and Comalco ($\sim$10%), however the Mines Department admit that their fluorine analyses are not consistent and "should be taken as indicative only".

Aim and methods

Comalco's aim when submitting the sample was the recovery of acid grade fluorite. The Department of Mines decided that an exploratory project was necessary, to fully determine the nature of the material submitted.

They carried out a series of scout tests applying normal concentration methods at three degrees of grinding. The concentration methods included heavy liquid separation, magnetic separation, gravity separation, and flotation of sulphides and fluorite. Full details are in Appendix 30.

Results

A full discussion of results is in a report entitled R 731, dated 21st December 1976, which is in Appendix 30. It was clear that very fine grinding of the wrigglite would be necessary to future work.

Conclusion

It was decided by H.K. Wellington (Mines Department) and A.H. Bartlett and P.W. Askins (Comalco) that the information emerging from the investigations would basically only mirror what Amdel had already determined. Therefore the future investigations were designed to accentuate problems which Amdel had not tackled, in particular scheelite recovery.

(b) Scheelite concentration project, investigation R 745.

Preliminary

As concluded above the exploratory project was requested to be continued with an emphasis on scheelite.

Thorough evaluation of scheelite recoveries had not been carried out by Amdel, partly because their bulk sample had a low tungsten content. Scheelite occurs in two main styles in the wrigglite: (i) as very fine grained scheelite disseminated within the fabric of the wrigglite. Under UV this fluoresces lime green and so is probably Mo rich; (ii) relatively coarse grained scheelite within the pink felspar veins which commonly penetrate the wrigglite. This generally fluoresces white and so is probably purer, with less Mo content.

It was thought that a thorough metallurgical evaluation of these styles of scheelite mineralization were warranted; scheelite could be more important economically than the tin. Assuming a mined recoverable grade of 0.1% WO_3 and prices of scheelite concentrate of $120/m.t.u. then this is worth $12 in-the-ground; - possibly sufficient to pay mining and milling costs.

Sample

The same sample as R 731 was used.

Methods

Successive grinding and concentration techniques were used to reduce a sample bulk of 95 Kg to a final particle size of - 53 μm. The ore was initially ball mill ground to -

-300 µm, producing the fractions + 170 µm, + 76 µm, + 53 µm and - 53 µm. The + 170 µm fraction was then subjected to magnetite separation on an Eriez drum type wet magnetite separator. The resulting non-magnetic fraction was then gravity concentrated on a laboratory Deister table and the concentrate produced was subjected to sulphide flotation. The flotation tailing was then dried and fed to a Rapid high intensity dry magnetic separator. This total process was repeated for each size fraction together with its appropriately sized and reground table tailing and Eriez magnetics. Thus by successive stages, the ore was concentrated at the following particle sizes.

Primary section	- 300 µm	+ 170 µm
Secondary section	- 170 µm	+ 75 µm
Tertiary section	- 75 µm	+ 53 µm
Quaternary section	- 53 µm	

Results

Full results in are Appendix 31 in the report entitled Comalco Limited - Moina wrigglite, R 745, dated 9th May 1978.

Both scheelite and cassiterite were difficult to recover. Scheelite overall recovery by tabling was only 33% producing a concentrate of only 22.7% WO_3. The problems are presence of scheelite-magnetite composites and that 39% of the overall WO_3 present was finer than 16 µm and unrecoverable by gravity concentration methods. (Furthermore 26% of the over-all WO_3 was finer than 9 µm!).

Recovery of tin in table concentrate was only 7%, producing a concentrate of only 7% Sn. This grade was raised to 21% Sn after dry magnetic separation, but recovery was then only 4% overall. 53% of the original tin was found to be in the sample in smaller than 16 µm particle size and 19% is locked in magnetite in a - 50 µm cleaner magnetics product.

A magnetite concentrate was produced which contained 82.5% magnetics (Davis Tube determination) and 63% HCl soluble iron. This could approach specifications suitable for coal washery purposes - (See Appendix 32 for such specifications)

Fluorite flotation was not attempted. 87% of the original fluorite was contained in the - 53 micron final gravity tailings.

Conclusion

These results are disappointing, but in view of the fine grain size of the wrigglite, not surprising. Considering

that the concentration methods were merely conventional gravity techniques, the results are somewhat encouraging.

Insufficient sample remained for future investigation.

(c) Scheelite concentration project, investigation R 758:

Because no further sample remained a one tonne parcel of wrigglite again from surface dumps was supplied.

This sample was collected by P. Askins and deliberately slightly biased by selection of pieces of wrigglite containing felspar veinlets. This was done in an effort to raise the head grade of scheelite and so make beneficiation tests a little easier.

A head assay by the Mines Department R 758 is:

Sn	0.3 %
WO_3	0.16%
S	0.8 %
Ca	12.7 %
Fe (total)	21.2 %
Fe (soluble)	20.4 %

No results of investigations have yet been received.

New techniques of flotation of scheelite in the presence of fluorite have been devised, see Appendix 33. Hopefully these methods can be attempted on the Moina wrigglite.

19. FINANCIAL FEASIBILITY STUDIES.

A. CAPTIVE SOURCE OF FLUORITE.

The beneficiation work done to date on the wrigglite has shown that it is feasible to remove fluorite, though with low recoveries. The wrigglite is therefore a satisfactory strategic captive source of fluorite for Comalco Limited's aluminium smelting operations at Bell Bay, Tasmania and Bluff, New Zealand.

The deposit is possibly the largest single fluorite resource in Australia.

B. COMMERCIAL RESOURCE.

(a) General:

The wrigglite is potentially a very large source of fluorite for the commercial market.

Based on purely commercial, largely overseas, sales of acid grade fluorite and by-product tin, a preliminary DCF study was carried out in July 1975, by G. Weste. (Appendix 34). This study was done before any drilling and showed that the wrigglite body was potentially economic. This was an encouragement for all the future investigations.

The beneficiation studies have since shown that the main problems affecting the wrigglite are the low attainable recoveries of CaF_2, Sn and W.

A recent brief financial analysis taking into consideration the available drilling and beneficiation information and revised metal prices is attached, Appendix 35.

(b) Conclusion:

The wrigglite body at Moina is a subeconomic source of CaF_2, Sn and W with presently known technology and commodity prices. It relies on all or some of the following factors to become economic.

(i) increased prices of CaF_2, Sn and W.

(ii) new beneficiation method to increase fluorite recoveries, and recoveries of by product cassiterite and scheelite.

(iii) higher Sn content in recoverable form in areas so far not drilled.

(iv) higher W content in recoverable form, say coarse scheelite in areas so far not drilled.

(v) sale of magnetite for say coal beneficiation.

(vi) sale of basalt overburden for say road-aggregate.

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Table 1

CHEMICAL ANALYSES OF GRANITES ASSOCIATED WITH WRIGGLITES

	(1) Dolcoath Granite	(2) Elizabeth Creek Granite Mt.Garnet Q'land	(3) Rapakivi Granite	(4) Granite Lost River Mine Alaska	(5) Average Stanniferous Granite
SiO_2	76.32	74.5%	75.6	75.73	73.12
Al_2O_3	12.35	13.1	14.1	13.63	14.03
TiO_2	0.03	0.19	0.21	0.02	0.18
MnO	0.03	0.02	0.07	0.11	0.04
MgO	0.20	0.23	0.34	0.19	0.46
CaO	1.45	1.59	0.72	0.63	1.22
Na_2O	3.16	3.40	3.42	3.47	3.47
K_2O	5.00	4.80	3.94	4.70	4.43
H_2O^+		0.29		1.17	
H_2O^-		0.14			
FeO		1.33	0.99	0.56	1.19
Fe_2O_3	0.51	0.36	0.44	0.24	0.83
FeS_2	-	-	-	-	-
P_2O_5	0.03	0.05	0.12	0.00	
CO_2		0.12	-	0.07	
F	0.50	0.02 - 0.6	0.6	0.55	

Notes: (1) Granite from diamond drill holes ML 1 at 860 ft. Sample 42244 in Webb (1974). F analysis by fusion/specific ion electrode, Comalco Laboratory, Melbourne, of granite at 862.5 ft in ML 1.

(2) Biotite adamellite from 26 km SW of Mt. Garnet (from Blake, 1972). The F analyses are range in unweathered granite from percussion cuttings at Pinnacles area. Analyses by fusion/specific ion electrode Comalco Laboratory, Melbourne.

(3) Rapakivi Granite. Recalculated from Eskola (1963), which quotes analyses as cations % and anions per 100 cations. Composite mixture of 54 samples from Finland.

(4) Granite, Lost River Mine from mine cross cut. From Sainsbury (1964 b).

(5) Average Composition of "stanniferous" granites in Kolyma, Eastern Siberia, Ural, Altay, and non Russian occurrences. (Shcherba, 1970)

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Table 2

Rock analyses, Moina Area, %

	Moina Sandstone (1)	Gordon Limestone (2)	Calc-silicate rock (3)
SiO_2	76.94	12.71	36.51
Al_2O_3	9.57	1.33	9.57
TiO_2	0.73	0.12	0.38
MnO	0.04	0.01	1.34
MgO	1.68	1.51	1.11
CaO	1.74	46.46	29.88
Na_2O	0.98	0.13	0.89
K_2O	4.67	0.47	0.25
Fe_2O_3	2.15	0.47	18.49
P_2O_5	0.30	0.03	0.11
Loss	0.73	36.70	1.35
	99.53	99.94	99.88

(1) Sample 42283 in Webb (1974). From diamond drill hole ML 3 at 612 ft.

(2) Sample 42246 in Webb (1974). From diamond drill hole DOM 1 at 70 m. (DOM 1 is about 500 m NW of hole SMD 9 on Fig.2).

(3) Sample 42279 in Webb (1974). From diamond drill hole ML 3 at 590 ft. (Analyses are expressed as oxides but presence of fluorite means that analyses particularly CaO might be slightly inaccurate).

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TABLE 3

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SMD 6 QUALITATIVE X-RAY ANALYSIS

BY DR. IAN GREY, C.S.I.R.O. DIV. MIN. CHEMISTRY.

DEPTH	Fluorite	Magnetite	Feldspar	Mica	Garnet	Amphibole	Quartz	Clays	Others	DESCRIPTION
26.0 m	3	2	1	0	2	2	0	0	0	Wrigglite (Wr) with spotty garnet
27.2 m	1	3	3	0	0	1	3	0	0	Pale calc-silicate (CS) rock
30.5 m	3	3	1	0	0	1	0	0	0	Wr with felspar veinlets
34.5 m	3	1	3	0	2	1	0	0	Pyrite 1	CS, chlorite, highly altered
37.2 m	2	3	2	0	0	1	0	0	0	Wr, mottled
41.2 m	3	3	1	0	0	2	0	0	0	Wr with felspar veinlets
44.5 m	3	2	1	0	0	3	0	1	0	Wr and CS interlayered
47.0 m	3	3	1	0	0	3	0	1	0	Wr
51.5 m	2	3	0	3	0	2	0	1	0	Wr
55.9 m	3	2	1	3	0	0	0	1	Pyrite 1	Wr, pure
59.1 m	3	3	1	3	0	2	0	1	0	Wr with felspar veinlets
65.0 m	3	2	1	1	0	3	0	1	0	Wr
67.9 m	1	2	3	0	3	1	3	1	0	CS rock
70.9 m	3	1	2	0	3	1	0	0	0	CS rock
74.5 m	3	2	3	0	3	1	0	1	0	Wr and CS interlayered
77.0 m	3	3	2	0	3	1	0	1	0	Wr
81.0 m	3	2	3	1	3	1	0	1	0	Wr
83.0 m	3	3	2	0	3	1	0	1	0	Wr
86.0 m	1	2	3	0	0	1	3	1	0	Pale CS rock
87.7 m	3	2	3	3	2	1	3	1	MoS_2 2	CS rock with greisen veinlets
92.2 m	2	3	2	3	0	1	0	1	Pyrite 1	Pale CS rock, dark veinlets
101.5 m	0	1	3	0	0	2	3	1	0	Siltstone

KEY: 0 Absent
1 Minor
2 Intermediate
3 Major

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TABLE 4.

CHEMICAL COMPOSITION OF WRIGGLITES

Element	Moina - Tas. Bulk sample from dumps, ppm.	Mt. Garnet - Qld. Bulk sample from percussion cuttings, ppm.	Kazah - U.S.S.R. from Zasedatelev (1973), ppm.
F	9.9%	16%	9 - 12%
Ca	15.2%	20%	14%
Si	11.3%	10%	2.5 - 9%
Al	3.9%	4%	4 - 6%
Fe	20.7%	14%	30 - 40%
Mg	1.4%	0.3%	0.26 - 0.6%
Mn		0.3%	0.09 - 0.15%
Cu	120	300	
Pb	75	100	
Zn	810	3000	0.7 - 1.2%
Ag	(0.1)	4	
Au	.035	<0.5	
Be	30	250	1400 - 2500
Sn	2250	1500	400 - 500
W	1400	300	300 - 2500
Bi	350	250	
Mo	115	25	300 - 400
Cd		5	
Ni		30	
Cr		(50)	
V		(10)	
Ta		(<100)	
Nb		(< 20)	
Th		(<100)	
Pt,Pd,Os		(< 10)	
Ir,Rh,Ru		(< 2)	
Ga		(10)	
Ge		(3)	
As		(< 50)	
Sb	45	(<30)	
Na		(1500)	
Ba		(200)	
Sr		(50)	
Y		(30)	
La		(<100)	
Ce, Nd		(<300)	
Pr		(<100)	
Ti		(600)	
Er		(<100)	
Sc		(< 50)	100 - 200
Eu		< 50	
S			300 - 400
Density		3.5	

NOTES: 1. Fluorine determined by distillation, fusion and specific ion electrode, and neutron activation analysis. Other elements by a variety of laboratories and methods, but chiefly atomic absorption spectrometry. Analyses quoted are approximate averages of various analyses.

2. Figures in brackets are semi-quantitative spectrographic analyses.

TABLE 5

COMPARISON OF WORLD WRIGGLITE OCCURRENCES

NATURE, OCCURRENCE, ETC.	KNOWN WRIGGLITE OCCURRENCES													
	Moina Australia	Mt. Garnet Australia	Mt. Bischoff Australia	Lost River Alaska	Iron Mtn USA	Chugoku Japan	Sthn China	Kristiana Norway	Pitkaranta USSR	Kazakh USSR	Central Asia USSR	Siberia USSR	Far East USSR	Primorye USSR
Texture fine grained, fine rhythmic layers	yes	yes	yes	yes	yes	yes	yes	yes	yes	yes	yes?	yes	yes	yes
Essential minerals: fluorite	yes	yes	yes	yes	yes	yes	yes	ves	yes	yes	yes	yes	yes	yes
magnetite	yes	yes	no	yes	yes	yes	yes	yes	yes	yes	yes	yes	yes	yes
idocrase	yes	yes	no	yes	yes	?	no	?	yes	yes	yes	yes	no & yes	yes
Accessory minerals including: cassiterite	yes	yes	yes	yes	no	no?	yes	?	yes	yes	?	?	yes	?
scheelite	yes	yes	no	yes	yes	no?	yes	?	?	yes	?	?	no?	?
beryllium minerals	?	?	yes?	yes	yes	no?	yes	?	yes	yes	yes	yes	yes	?
Geodes, or zones of coarse minerals, geometrically central to the layering	uncommon	yes	yes	yes	yes	?	?	?	yes	yes	?	?	yes	?
Unconformable relationships between adjoining layered sequences	yes	yes	yes	yes	yes	?	?	?	yes	yes	?	?	yes	?
Replaced rock is limestone, marble or dolomite	yes	yes	yes	yes	yes	yes	yes	yes	yes	yes	?	yes	yes	yes
Evidence that argillaceous rock is not replaced by wrigglite	yes	indirectly,yes	indirectly,yes	yes	?	?	?	?	yes	?	?	?	?	?
Spatially closely associated with leucocratic or 'alaskite' granites	yes	yes	yes	yes	yes	yes	yes	yes	yes	yes	?	yes	yes	yes
Greisenization common in granite	yes	yes	not exposed	yes	?	?	yes	?		yes	?	yes	yes	yes
Within a metallogenetic province of: tin	yes	yes	yes	yes	no	no?	yes	?		?	?	?	yes?	yes
tungsten	yes	yes	yes	yes	yes	yes	yes	?		?	?	?	yes?	yes
beryllium	yes?	yes?	no	yes	yes	no?	yes	?		yes	yes?	yes	yes?	yes

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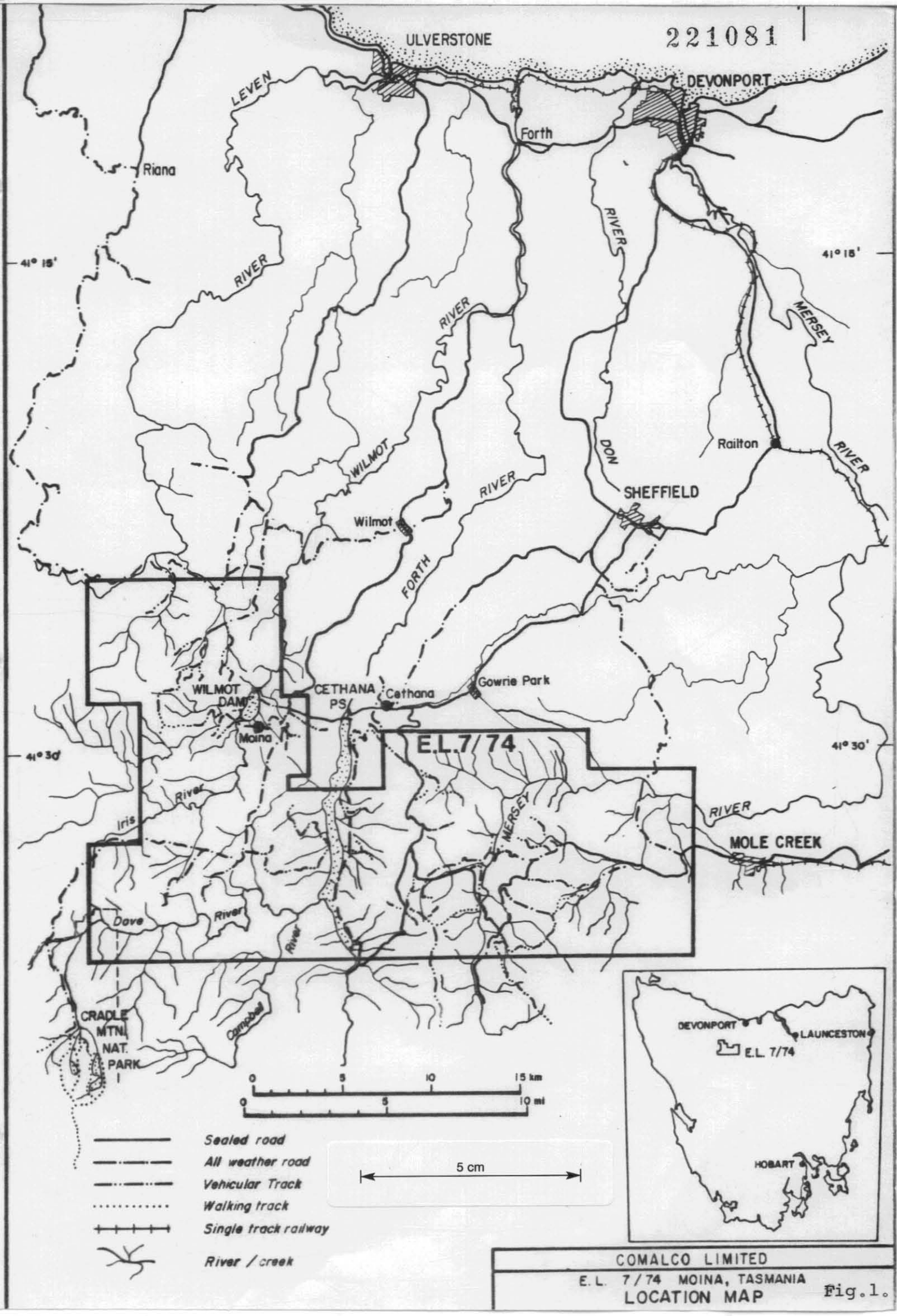


COMALCO LIMITED

E.L. 7/74 MOINA, TASMANIA

LOCATION MAP

Fig.1.

COMALCO

E.L. 7/74 - MOINA
GRIDDED AREAS AND
SHEET LOCALITIES

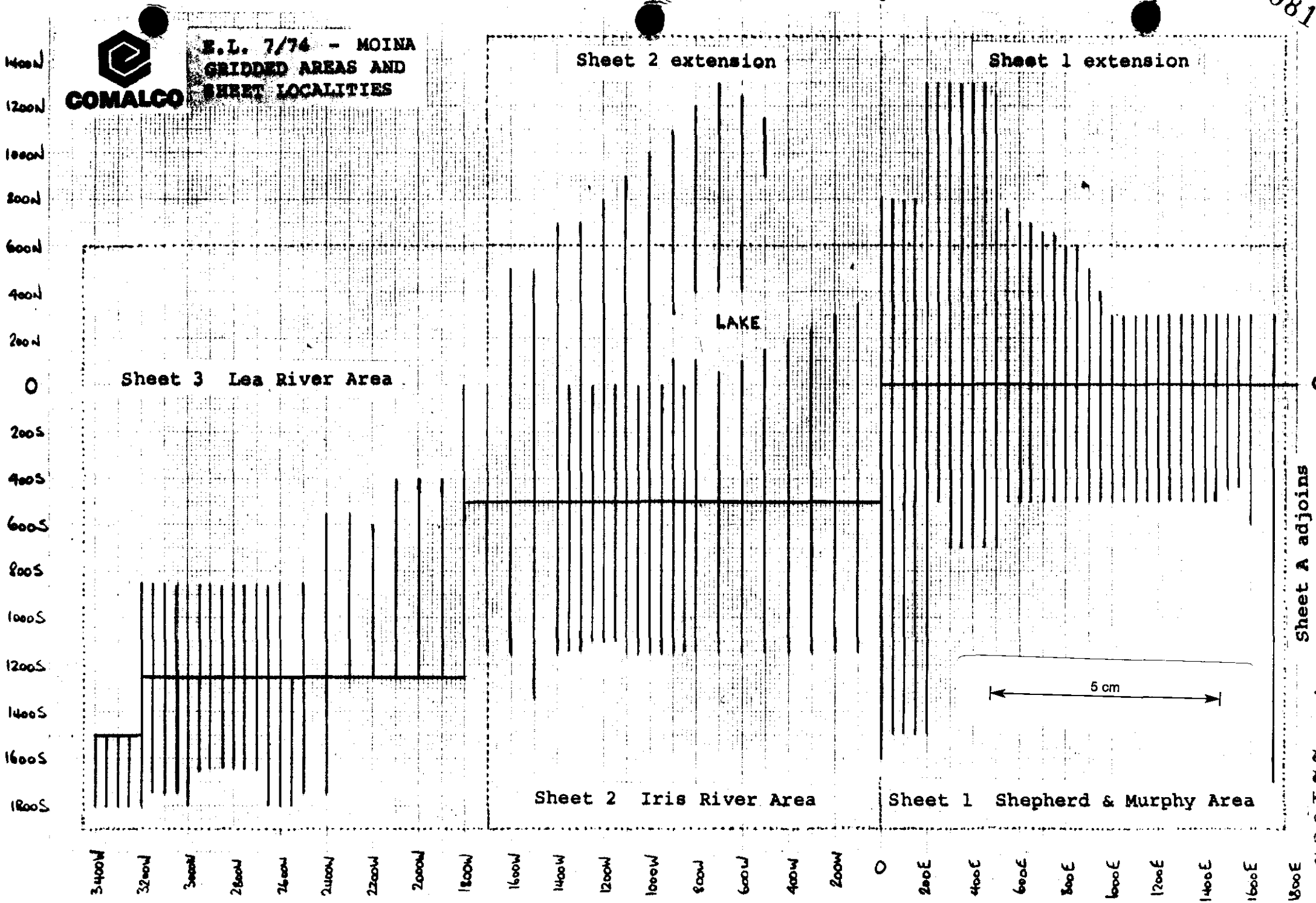
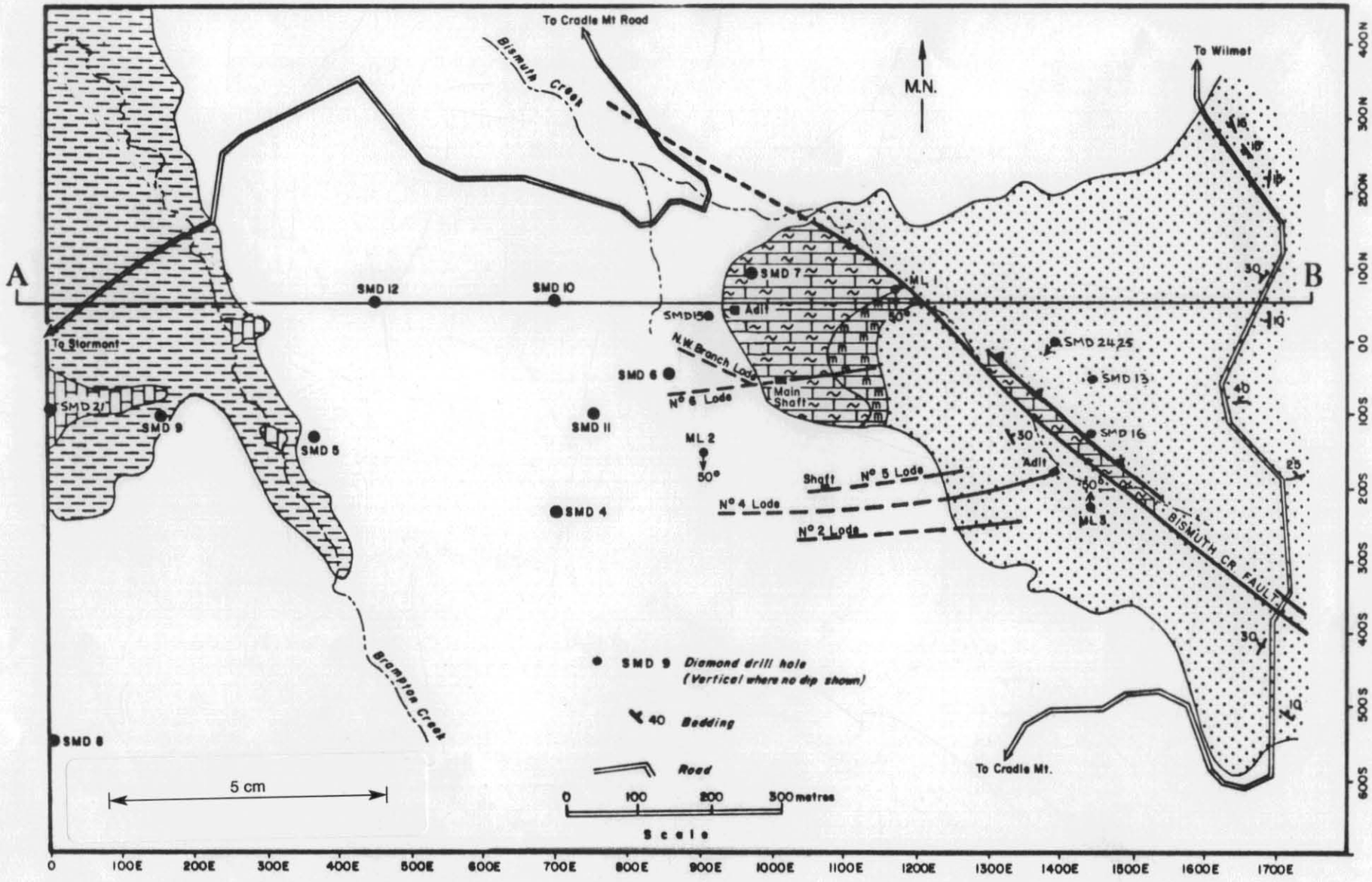


FIG.2.



GEOLOGY, SHEPHERD AND MURPHY MINE AREA, MOINA FIG. 3

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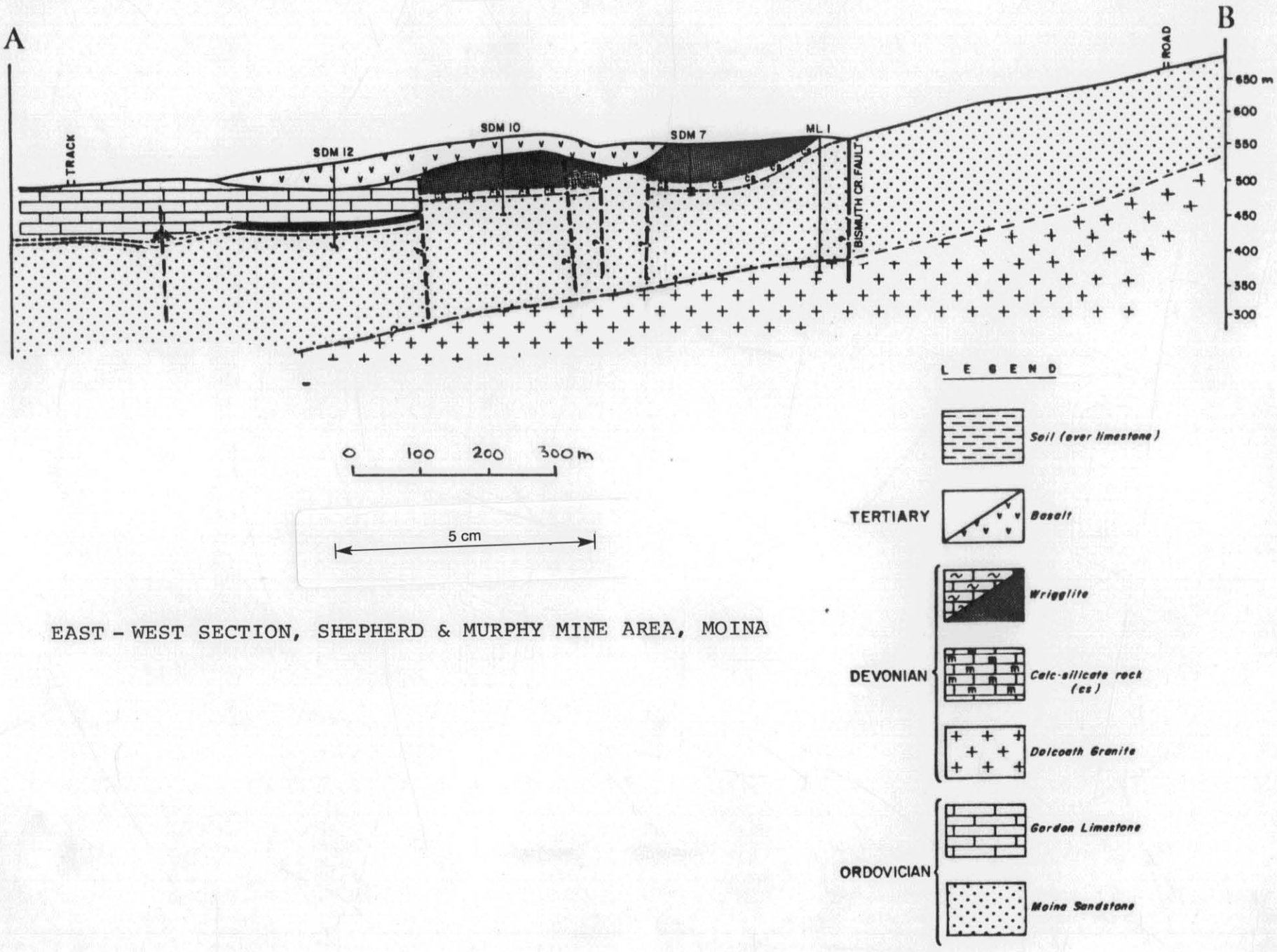
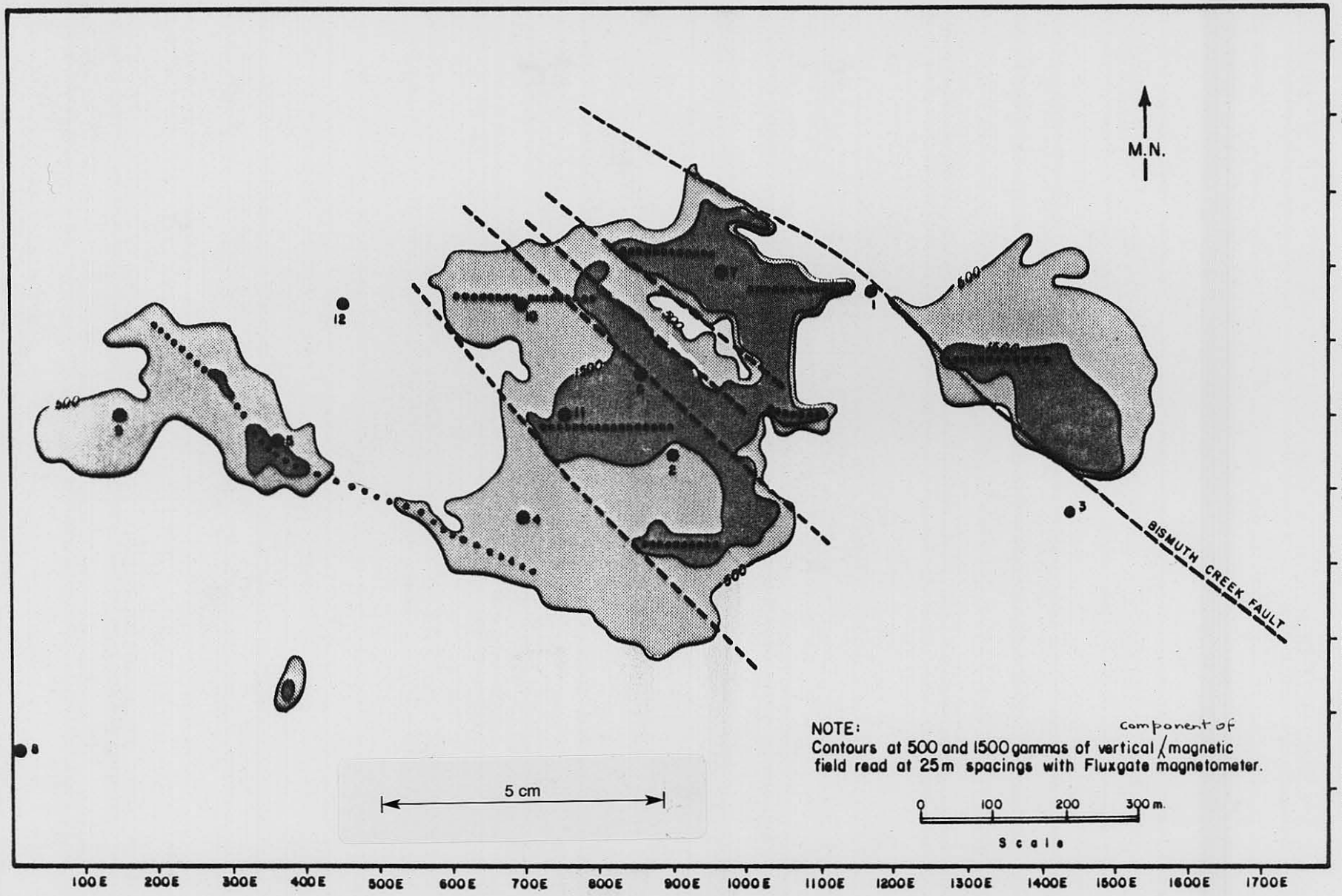


EAST - WEST SECTION, SHEPHERD & MURPHY MINE AREA, MOINA

FIG. 4.



MAGNETIC CONTOUR PLAN SHOWING INTERPRETED N.W. FAULT-FRACTURE & E.W. FOLD &/OR LODE FRACTURE SYSTEMS

FIG. 5.

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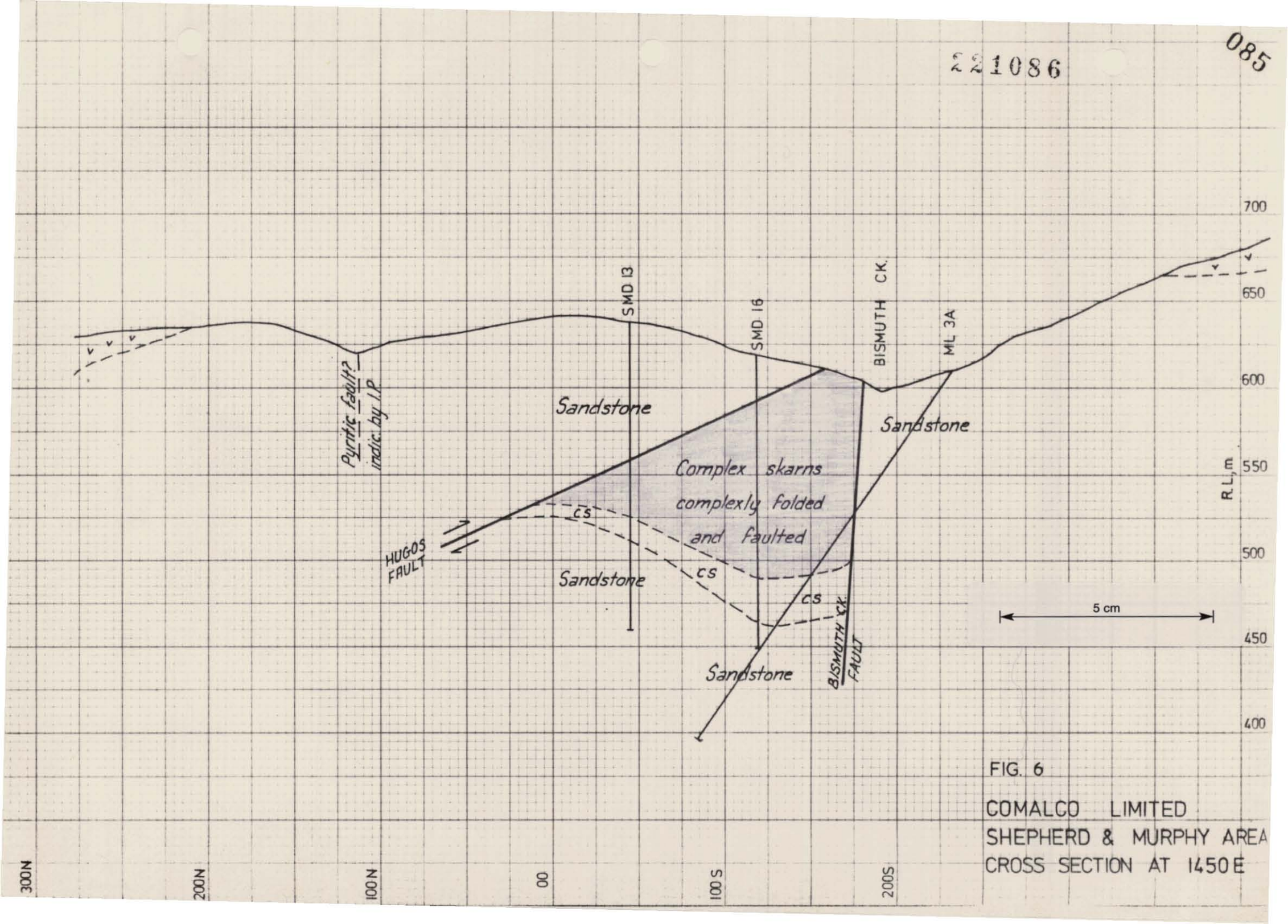

FIG. 6

COMALCO LIMITED
SHEPHERD & MURPHY AREA
CROSS SECTION AT 1450E



WRIGGLITE WITH CALC-SILICATE INTERBEDS, SMD 4. Fig.7.

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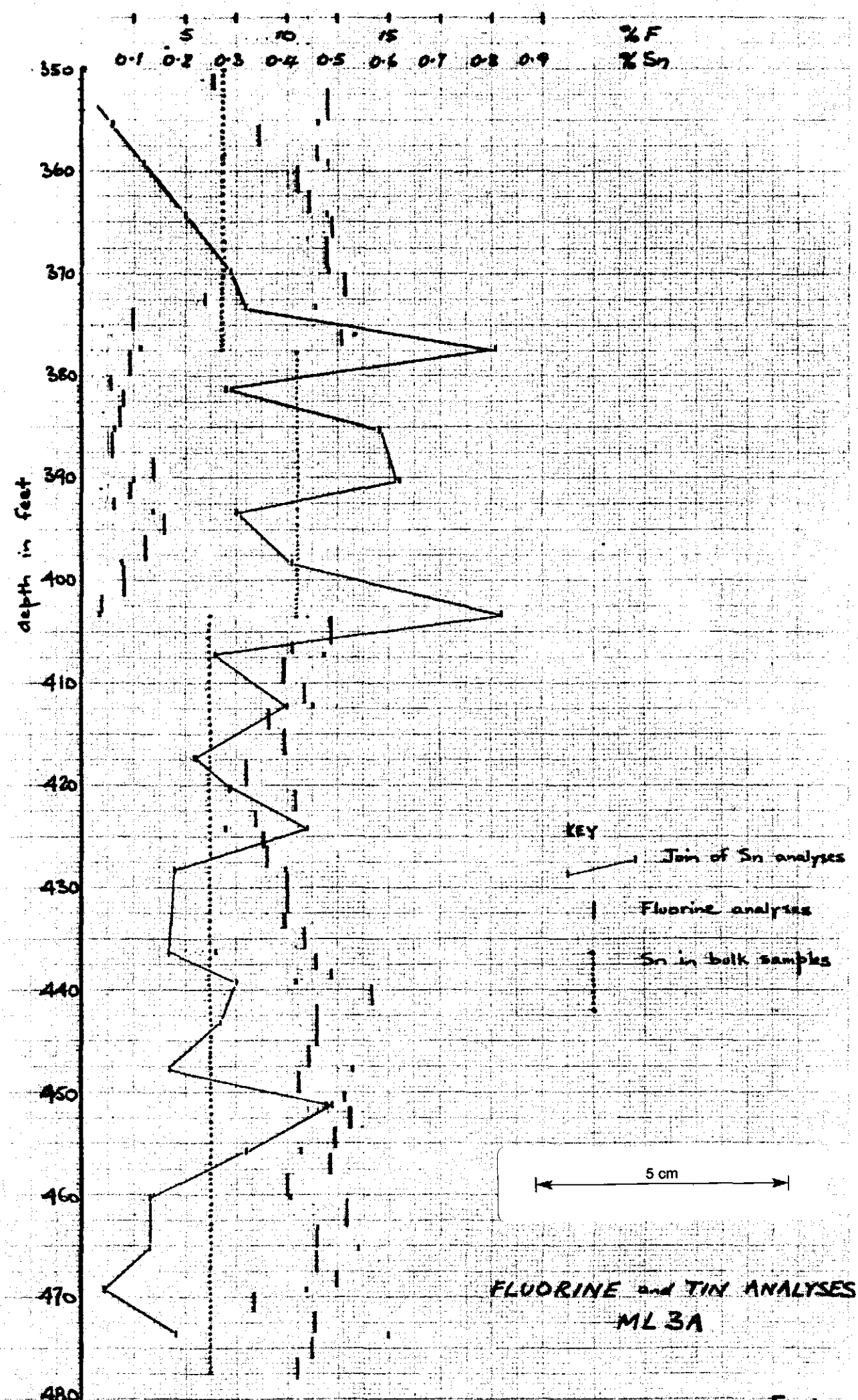

FLUORINE and TIN ANALYSES
ML 3A

Fig 10.

08-

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1. Examination of wrigglite for presence of cuspidine.
2. Wrigglite petrology by Amdel.
3. Thin and polished section study of wrigglite by W. Fander.
4. Thin section description by W. Fander of chlorite rich skarn, ML3A.
5. Known wrigglite occurrences in the world.
6. Origin of layering in wrigglites.
7. Nature of the mineralizing fluid.
8. Analytical techniques used by Comalco, Amdel.
9. Analyses of samples Shepherd and Murphy area collected by Gove 1974.
10. Analyses Moina area collected by G. Weste.
11. Shepherd and Murphy area. Samples collected by A.H. Bartlett, 1976
12. Costean sampling and analyses Shepherd and Murphy area.
13. Mines Department samples, Shepherd and Murphy area.
14. Analyses of rock samples, Lea River and Iris River areas.
15. Post Office anomaly, hand auger sampling, - analytical results.
16. Geophysical rock property tests.
17. A report on Gradient electrical induced polarization surveys over the Shepherd and Murphy grid. Scintrex Pty. Ltd.
18. Investigation of an I.P. anomaly Shepherd and Murphy grid.
19. Investigation of "Post Office" I.P. anomaly.

* 20.A A report on M.I.P. survey, Moina, by A. Howland-Rose, Scintrex Pty.
* 20.B Ltd. A: report B: plans. (*bound separately)

21. Electrical logging of diamond drill holes, Moina.
22. Density of various rock types, Moina.
23. Seismic and Resistivity Trials, Moina.
24. Diamond drill logs, Moina.
25. Analyses of split core.
26. Recovery logs, diamond drilling, Moina.
27. Sedimentological logs by P.W. Stainton.
28. Amdel. Chemical extraction of fluoride from wrigglite.
29. Amdel. Physical extraction investigations of Moina wrigglite.
30. Department of Mines. Investigation R731 - Exploratory investigations, Moina wrigglite.
31. Department of Mines. Investigation R745 - Scheelite concentration project.
32. Specifications for magnetite for heavy media use.
33. New techniques for scheelite flotation.
34. Preliminary DCF study, Moina fluorite prospect July 1975.
35. Financial analysis, Moina wrigglite. Sept, 1978.

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amdel

AN 2227/75

APPENDIX 1.

EXAMINATION OF SAMPLE FOR CUSPIDINE

Sample M2 180-180'6" was examined:*

1. XRD powder trace of the sample showed the presence of fluorite, magnetite, (?) vesuvianite and a trace of mica.

   Cuspidine was not detected. The limit of detection is not known, but should be less than 5%. Since cuspidine contains 10% F, this would correspond to <0.5% F present in cuspidine, which would correspond to a small error only compared with the F analysis figure of 11.6%.

2. Under the microscope no pink or red mineral was detected, but cuspidine may be uncoloured and would be difficult to detect.

* pure wrigglite from core of hole ML2 at 180 ft.

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APPENDIX 2

221091

C6-3(ii)

amdel
The Australian Mineral Development Laboratories

Flemington Street, Frewville, South Australia 5063
Phone Adelaide 79 1662, telex AA82520

Please address all correspondence to Frewville,
In reply quote: MP3/1/4/0

20th October, 1975

Comalco Exploration Dept.,
P.O. Box 2773Y, G.P.O.,
MELBOURNE, Vic. 3000.

Attention Mr. G. Weste

REPORT MP887/76

YOUR REFERENCE:	G.W./T/SM.
MATERIAL:	2 rocks.
IDENTIFICATION:	M2/194/1; M2/200/4. *
DATE RECEIVED:	22.9.75.
WORK REQUIRED:	As specified.

Investigation and Report by: Dr B.G. Steveson.

Officer in Charge, Mineralogy/Petrology Section: Dr K.J. Henley.

K.J. Henley.

for F.R. Hartley
Director

grs

* both samples of wigglite from diamond drill hole ML2 at 194 ft and 200 ft

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221092

EXAMINATION OF METASOMATIC FLUORITE-BEARING ROCKS

1. INTRODUCTION

Two samples (labelled M2/194/1 and M2/200/4) were received from Comalco with a request for mineragraphy, X-ray diffraction examination and spectrographic analysis. The samples are fragments of drill-core collected from a skarn locality.

2. RESULTS

2.1 X-ray Diffraction Results

X-ray diffraction analysis of a relatively large sample of each rock gave the following results:

Sample M2/194/1

Biotite	Dominant
Magnetite	Subdominant
Fluorite	Accessory

Sample M2/200/4

Fluorite	Dominant
Magnetite	Subdominant
Amphibole	Subdominant
Chlorite	Trace
Mica	Trace

D = Dominant. Used for the component apparently most abundant, regardless of its probable percentage level.

SD = Subdominant. The next most abundant component(s) providing its percentage level is judged above about 20.

A = Accessory. Components judged to be present between the levels of roughly 5 and 20%.

Tr = Trace. Components judged to be below about 5%.

Both X-ray diffraction traces contain several small peaks which were not identified; however, a check was made of numerous minerals which might be expected in skarn-type rocks.

2.2 Mineragraphy

M2/194/1; (PS24002)

The polished briquette of this sample contains about 20-30% magnetite, less than 5% pyrite and trace amounts of chalcopyrite, cassiterite, marcasite and (?)hematite. Fluorite can be recognised in the polished briquette by its extremely low reflectivity and this mineral comprises of the order of 20-30% of the briquette also.

Magnetite occurs as irregular patches which are up to about 1 mm in size but more commonly occurs as small equant crystals which range in size down to only a few microns. Macroscopic examination of the surface of the briquette shows that the magnetite tends to form swirling irregular patches which have a banded appearance. Pyrite forms both small crystals and diffuse and porous patches up to about 0.5 mm in size. The pyrite is commonly associated with magnetite and in one or two places both these minerals occur in thin fractures which cut across the main swirling banding referred to above. Chalcopyrite occurs both as fine-grained material with pyrite and also, in one part of the briquette, is present as a group of crystals 0.7 mm in size.

Poorly polished pale yellow anisotropic marcasite is associated with this relatively coarse-grained chalcopyrite as aggregates up to 0.8 mm in size, the lamellar texture of which suggests formation from pre-existing pyrrhotite.

Cassiterite is present only in very small amounts and forms equant crystals approximately 0.03 mm in size. The cassiterite is associated with the non-opaque minerals and shows no special association with either pyrite or magnetite. If the cassiterite crystals intersected in the polished briquette are typical of cassiterite in the sample as a whole, then crushing to a particle size of less than 0.03 mm would be required to liberate the tin from the adjacent silicates and fluorite.

Fluorite is a notably abundant mineral in this sample and it occurs in a wide range of associations and grain sizes. In some places in the rock fluorite is clearly present as a vein mineral and occurs in thin stringers which are generally less than 0.1 mm in size. Elsewhere in the rock the fluorite is present as numerous equant crystals within silicates where it ranges in size from a few microns to about 0.2 mm. If it were required to obtain a high-grade fluorite concentrate from the sample (of the order of 90-100% fluorite), then grinding to an extremely small grain size would be required if reasonable recoveries were to be obtained; probably grinding to a grain size of 0.1 mm would effect the liberation of only approximately 20-40% of the fluorite in the sample.

M2/200/4; (PS24003)

Apart from the trace amounts of hematite, pyrite, and cassiterite, magnetite is the only opaque mineral in this sample. Magnetite comprises approximately 10-20% of the rock and fluorite about 20%. The remainder of the sample consists of a non-opaque silicate which, from the X-ray diffraction results, is probably amphibole. Magnetite is notably fine-grained in this rock and generally occurs as equant crystals which are commonly 0.02 to 0.05 mm in size. Much of the polished briquette is densely speckled with this magnetite and, as in the preceding sample, the magnetite tends to occupy curved swirling areas which define an extremely irregular banding. In some places in the polished briquette the small magnetite crystals form almost continuous patches and monomineralic areas up to about 0.7 mm in size.

Fluorite is widely dispersed among the magnetite and amphibole and occurs as equant crystals or groups of crystals which range in size from approximately 0.05 mm to 1.5 mm. Fluorite is particularly coarse-grained where magnetite is least abundant and in some parts of the polished section patches of fluorite more than 0.2 mm in size occupy about 40% of the rock. Elsewhere the fluorite is considerably finer-grained and commonly closely associated with fine-grained magnetite. Fluorite also occurs in the polished briquette in very thin veinlets, but only a small proportion of the fluorite occurs in this fashion.

A small proportion of the fluorite in this rock could be liberated by relatively coarse grinding to particle sizes of the order of 0.2 mm but moderate to large recoveries of high-grade fluorite concentrates would require grinding to much smaller sizes than this (probably about 0.02 to 0.04 mm).

Cassiterite is a little more abundant in the polished briquette of this sample than in the briquette of M2/194/1 but, even so, may be described as being present in trace amounts only. The cassiterite forms equant and anhedral grains which range in size from 0.03 to 0.06 mm. The cassiterite is generally associated with fluorite and amphibole rather than magnetite and most crystals are free from inclusions. As a result of these observations it is clear that liberation of the cassiterite from the rock would require grinding to particle sizes of the order of 0.02 to 0.03 mm.

2.3 Emission Spectrography Results

Emission spectrography analyses were carried out on portions of each sample and the results are as follows:

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SPECTROGRAPHIC ANALYSIS

Detection-Limit Concentrations of Elements
DC Arc Excitation

Element	%	ppm	Element	%	ppm
Ag	0.00001	0.1	Na	0.005	50
Al	0.02	200	Nb	0.02	20
As	0.005	50	Nd	0.03	300
Au	0.0003	3	Ni	0.0005	5
B	0.001	10	Os	0.001	10
Ba	0.005	50	P	0.01	100
Be	0.0001	1	Pb	0.0001	1
Bi	0.0001	1	Pd	0.001	10
Ca	0.002	20	Pr	0.01	100
Cd	0.003	3	Pt	0.001	10
Ce	0.03	300	Rb	0.001	10
Co	0.0005	5	Re	0.001	10
Cr	0.002	20	Rh	0.0002	2
Cs	0.003	30	Ru	0.0002	2
Cu	0.00005	0.5	Sb	0.003	30
Dy	0.01	100	Sc	0.005	50
Er	0.01	100	Si	0.02	200
Eu	0.005	50	Sm	0.03	300
Fe	0.005	50	Sn	0.0001	1
Ga	0.0001	1	Sr	0.001	10
Gd	0.03	300	Ta	0.01	100
Ge	0.0001	1	Tb	0.01	100
Hf	0.02	200	Te	0.002	20
Hg	0.01	100	Th	0.01	100
Ho	0.01	100	Ti	0.01	100
In	0.001	10	Tl	0.0001	1
Ir	0.0002	2	Tm	0.01	100
K	0.0005	5	U	0.1	1000
La	0.01	100	V	0.001	10
Li	0.0001	1	W	0.005	50
Lu	0.03	300	Y	0.001	10
Mg	0.01	100	Yb	0.005	50
Mn	0.001	10	Zn	0.002	20
Mo	0.0003	3	Zr	0.02	200

THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES

REPORT AN.....9716/76.

Page.....

QUALITATIVE SPECTROGRAPHIC ANALYSIS

Sample No.	Major	Minor	Heavy Trace (0.1—1%)	Trace (0.01—0.1%)	Faint Trace (0.001—0.01%)	Very Faint Trace (0.0001—0.001%)
M2/194/1	Ca Fe Mg	Al K Si	Rb Sn Zn Cs	W B As Ti Li Bi Sb Na Cu	Be Zr Ga Cr Ni Pb V Y Ge	Co Mo Ag
M2/200/1	Ca Fe	Al Si Mg	K	Sn As W Rb Sb Bi B Ti Na Zn	Be Cr Li V Zr Cs Pb Ga Y	Ni Ag Cu Ge

221096

Elements not sought: Hg, U

Other elements not detected at limits quoted in attached sheet: Co, Mo, Nb, Re, Ta, Th, Cd, In, Au, P, Te, Tl, Ba, Ce, Er, Eu, La, Nd, Pr, Sc, Se

096

221097

APPENDIX 3

Central Mineralogical Services Pty. Ltd.

231 Magill Road
Maylands, S.A. 5069
Telephone 42 5659

3rd June 1976

Mr. A.H. Bartlett,

Exploration Manager,
Comalco Limited,
95 Collins Street,
MELBOURNE. VICTORIA. 3000

REPORT CMS 76/5/16

YOUR REFERENCE:	Verbal request, A.H. Bartlett, 21.5.76
DATE RECEIVED:	24.5.76
SAMPLE NOS.:	Four chip samples (Moina 3, 4, 5, 6)
SUBMITTED BY:	Mr. A.H. Bartlett
WORK REQUESTED:	Mineralogy/Petrology

H.W. Fander, M.Sc.

Copy to:

Mr. P. Askins,
Comalco Limited,
P.O. Box 691,
Devonport,
Tasmania. 7310

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221098

NOTES

Sample No's

03/0701/

MO3 Chip sample of wrigglite from 3 outcrops at Shepherd and Murphy Mine.

MO4 Tailings Shepherd and Murphy Mine. Consists of 1/8 inch mixture of quartz, wrigglite, etc.

MO5 Garnet rich skarn, Shepherd and Murphy Mine, exact locality unknown.

MO6 Pale calc-silicate rock, Shepherd and Murphy Mine, exact locality unknown.

Fander's descriptions of MO5, MO6 are more typical of wrigglite. He has probably examined wrigglite occurring in the specimens interlayered with the calc-silicate rock or garnet skarn.

For analyses of these rocks see Appendix 5.

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REPORT CMS 76/5/16

MOINA FLUORITE SAMPLES

Introduction

Four chip samples from the Moina prospect were submitted for detailed mineralogical-petrological examination, with particular reference to the following :

a. Nature and occurrence of all minerals
b. Determination of presence or absence of cuspidine
c. Nature and quantity of cassiterite, and the possibility of other potential Sn-bearing phases
d. Liberation of fluorite
e. Presence and nature of other economic minerals (Cu-Pb-Zn, W, Mo and others).

Mr. Bartlett emphasized in the discussion that the information should be as representative as possible, and therefore it was agreed that several thin-sections should be prepared of each sample. Four samples (Numbers 3, 4, 5, 6) were supplied, together with analytical information.

Procedure

One polished section and four thin-sections were prepared for each sample, using smaller chips for the polished sections, larger ones for the thin-sections. Potash-staining tests were carried out on chips.

Because of the general similarities in composition and fabric, the samples are described as a group, with comments on variations in individual samples. It was decided that photomicrographs would be more valuable in presenting the important characteristics than lengthy descriptions, particularly in regard to metallurgical assessment of the samples. There is nothing to be gained from repetitious descriptions of many thin-sections; it will be more important to examine mill-products resulting from beneficiation experiments.

Results

(a) General Comments

Very generally, the rocks are metasomatic in character, and it is evident that there have been several phases of replacement, introduction of material, and late-stage veining; the phases are believed to be genetically linked.

The main problem is that of the origin of the rocks and their primary composition (i.e. before metasomatism). Metasomatic

alteration was so intensive that none of the original minerals are thought to have survived; however, the present fabrics are believed to have been inherited, to a large extent, from the primary rocks. Also, the present composition in some measure reflects the original composition.

Thus, in fairly speculative terms, the original rocks could well have been in the nature of finely-banded or laminated ferruginous cherts, quite possibly intercalated with argillites and carbonate rocks. The dominant feature of all the samples is the presence of magnetite bands, and in some these are very thin, and are intricately folded on a fine scale akin to colloform-banding and crustification. These distinctive magnetite bands are thought to represent original ferruginous layers, to account for their present structure. The only other explanation is that the banding represents some type of rhythmic, metasomatic deposition, perhaps coupled with highly selective replacement processes; this seems far-fetched at this stage, though field observations would have an important bearing on the interpretation.

An interesting feature is the similarity between these rocks and the Mt. Garnet material, in fabric, and broadly in composition. Cuspidine was not detected.

(b) Descriptions

M3 The major rock type, represented by the chips in four thin sections, is a finely-banded magnetite-fluorite rock, with varying amounts of other minerals. The banding is often very fine, and is intricately folded, even forming closed structures (depending on direction of sectioning). The distances between similar bands (say magnetite) are often as little as 0.1 - 0.3mm, and thus the bands themselves are very thin. This in turn means that the constituents are mostly very fine-grained, except where lenses and patches of more coarsely-granular material have formed, fairly haphazardly.

The magnetite-fluorite rock generally contains diopside, ferro-hastingsite, green biotite and feldspar (some of this is a later addition, see below). Subordinate rock types include medium-grained ferrohastingsite-fluorite-garnet-magnetite intergrowths, with and without andesine. Rocks with diopside, and with minor wollastonite, are also present. Garnet is generally a markedly anisotropic andradite-grossularite variety.

Adularia is common in some fragments and is believed to be a late-stage, second metasomatic phase replacing some earlier-formed

minerals; this probably also applies to some of the plagioclase. Both types of feldspar also constitute late-stage, cross-cutting veins and patches; they are rather lacking in the distinctive characteristics usually associated with them, and in consequence are easily confused with each other and with quartz. Quartz is in fact a very minor mineral only.

The importance of the feldspar veins, especially the plagioclase (andesine) veins, is that they contain most (perhaps almost all) the scheelite which is virtually the entire source of the tungsten. The scheelite occurs as irregular and anhedral grains, sporadically distributed and ranging from 20μ to 300μ in size.

Cassiterite is also sporadic in its distribution, as irregular patches up to 100μ across, intergrown with fluorite, quartz, and feldspars. No other Sn-bearing phases were detected, though traces of "stannospinel" may be present (extremely fine-grained). Spinels are absent, in contrast to Mt. Garnet, with the exception of magnetite.

Magnetite occurs as individual crystals 2μ to 100μ (mostly < 20μ), forming thin bands and larger aggregates. There are also patches of magnetite with "colloform" microtexture, and thin films (shells) of magnetite on other minerals; these shels are often 5μ or less thick. Traces of bismuthinite were detected, as < 20μ grains, embedded in non-opaque minerals.

M4 This sample is very similar to M3, especially in regard to the fabric and also in general composition. In addition to the finely-banded magnetite-fluorite types, some carry green biotite. A few massive garnet and garnet-ferrohastingsite rocks occur, and muscovite is fairly widespread. It seems to be equivalent to adularia, constituting a late stage of greisening. There are relatively coarsely crystalline assemblages of quartz-topaz-fluorite-muscovite-feldspar representing a distinctly different phase younger than the main metasomatism.

Veins and patches of medium- to coarse-grained crystal-mosaics have formed, consisting of quartz-fluorite, adularia-fluorite, and andesine-fluorite. Scheelite is present in fluorite-andesine veins, as anhedral grains 20 - 250μ and larger patches up to 700μ with minute magnetite inclusions.

Cassiterite was detected as 3 - 18μ grains, and small aggregates up to 30μ, in fluorite.

The polished section shows that the magnetite is very similar to

that already described. The main feature of the magnetite is its generally extremely fine grainsize and pervasive distribution through the other constituents. Traces of bismuthinite (< 20μ) and molybdenite (20 x 40μ - 5 x 100μ flakes) were seen, in non-opaque phases. A possible wolframite-scheelite intergrowth was also noted.

To summarise, these rocks are generally fine-grained and finely-banded, with coarser veins and greisen patches.

M5, M6 These two samples, whilst closely similar to M3 and M4, have a slightly different fabric. They are on the whole coarser-grained (though still rather fine) and less markedly banded. Intricately-folded fine banding is quite subordinate.

Compositionally the rocks are microcrystalline intergrowths of magnetite, fluorite, green biotite (altered to chlorite in places), plagioclase and sporadic garnet, diopside, ferrohastingsite and ?stilpnomelane (with epidote).

There are comparatively coarsely-crystalline veins and patches of muscovite-fluorite-quartz-oligoclase. Some of these contain scheelite, as irregular grains up to 750μ across though generally finer. Bands and lenses of matted pale phlogopite and pale-green biotite occur, studded with abundant brown pleochroic haloes surrounding minute ?brannerite grains.

Cassiterite is very sporadic and scarce, as scattered grains up to 50μ in size in fluorite and green biotite(-chlorite). The greisening phase described in M4 is apparently absent though possibly represented by the muscovite-bearing veins.

In polished section it was seen that the distribution and form of the magnetite is more uniform and granular. Individual crystals range from 2μ to 200μ, often forming aggregates and semi-continuous masses. Chalcopyrite was detected, as irregular patches up to 100μ across. Pyrite is conspicuous in places and up to 1mm across (poikiloblastic crystals). Marcasite forms fine dendritic growths, and bismuthinite occurs as small, rounded grains seldom exceeding 50μ in size. Sphalerite is abundant in places, intergrown with magnetite and forming grains < 50μ, to semi-continuous patches. Simple intergrowths of cassiterite/magnetite were also observed, in which the cassiterite is up to 50μ in size.

Summary of Economic Minerals

1. Fluorite

The bulk of the fluorite is very fine, individual grains seldom

exceeding 0.1 - 0.2mm and generally much finer (< 50μ). Where it occurs in more coarsely-crystalline patches and veins, it attains sizes up to 500 - 600μ and may be fairly clear and "clean". Fluorite is generally intimately intergrown with other minerals, mainly magnetite but also a variety of silicates such as feldspars, micas, diopside, ferrohastingsite, and others.

Aggregates of fluorite crystals seem to be rare; most occur as individuals interspersed with, and isolated by, other minerals. The fluorite is clear and colourless or pale mauve, except for occasional strongly coloured crystals.

2. Scheelite

This mineral is regarded as of some significance and potential interest. It is consistently associated with late-stage plagioclase-fluorite veins which tend to be pinkish in hand specimen. Most scheelite is reasonably coarse (up to 700μ) and is easily detected under short-wave UV radiation. Its fluorescence is white, suggesting a low Mo content.

Further exploration for this mineral should be seriously considered, especially in view of rising tungsten prices (about £62/tonne unit, 65% WO_3)

3. Cassiterite

This mineral is present in minor amounts in all the rocks. Because of its very fine grainsize, it is difficult to detect, but its content would be much less than 0.5%. Grainsizes seldom exceed 100μ, and are more commonly in the range 5 - 50μ, with some finer material.

If the assays should be unexpectedly high, then the presence of Sn in other minerals must be considered. No "stannospinel" was detected with certainty, and Sn in solid solution would need to be sought by electron-probe microanalysis. In view of the absence of spinels and corundum (in contrast to Mt. Garnet), it may well be that "stannospinel" did not form.

The cassiterite appears to have been introduced with the first metasomatic phase but may also be related to the greisening; this may be worth following up, because the greisening phase is distinctive.

4. Sulphides

Bismuthinite occurs in all samples but is generally very fine-grained; it may be recoverable as a by-product.

Molybdenite was detected as sporadic small flakes, and chalcopyrite occurs as fine, irregular patches. Sphalerite is the most abundant sulphide (apart from pyrite) but is erratic in its distribution.

Metallurgical Comments

The major problems in the recovery of fluorite are the fine grainsize and the small-scale intergrowths with other minerals. Magnetite is likely to be particularly troublesome, because of composites of fluorite/magnetite leading to fluorite losses. In view of these factors, fluorite flotation may have to be considered, but it is doubtful whether acid-grade fluorite could be produced economically from this ore (i.e. as examined).

However, the material should perhaps be considered as a whole, with the possibility of producing high-grade magnetite (for use as a dense medium?), metallurgical grade fluorite, minor scheelite and a bulk sulphide concentrate (for further separation). The cassiterite appears to be too fine and sparse to be considered economically, and if Sn occurs in other minerals, extraction would be problematical. This situation may be different with better grades.

The coarser fluorite is naturally more amenable to beneficiation, but constitutes a subordinate proportion of the ore; thus grades would be obtained at the expense of recovery, and vice versa.

Photomicrographs

These are intended to give a general impression of the material, showing fabrics, textures and distribution of minerals; illustrations of particular features were also prepared.

H.W. Fander, M.Sc.

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REPORT CMS 76/5/16

PHOTOMICROGRAPHS

Photo 1. Sample M3 Thin Section x30

General view of typical banded "wrigglite" rock showing alternating magnetite (black) and fluorite-feldspar bands. An area of coarse fluorite occurs, and there are feldspathic veinlets, partly conformable.

Photo 2. Sample M3 Thin Section x125

Part of same field as in Photo 1, showing fine intergrowths of minerals. Fluorite = small clear grains.

Photo 3. Sample M3 Thin Section x125

Coarse fluorite (more or less clear), garnet (centre, high relief), ferrohastingsite (dark) and magnetite.

Photo 4. Sample M4 Thin Section x125

Part of fluorite area in which feldspar has been replaced by muscovite. Black = magnetite; clear = muscovite; cloudy (with low relief) = fluorite with minute hastingsite inclusions.

Photo 5. Sample M4 Thin Section x125

Irregular scheelite grains (cloudy, high relief) in a medium-crystalline patch of fluorite, plagioclase, magnetite. Note small magnetite inclusions in scheelite (scheelite is younger). Fluorite = clear grains with sharp boundaries.

Photo 6. Sample M5 Thin Section x30

Rock fragment showing vein of coarsely-crystalline fluorite, plagioclase and green biotite, and coarse scheelite (very dark, with magnetite inclusions). Fluorite = grains with heavy borders. Note dark spots in biotite; these are minute radioactive grains (?brannerite) surrounded by pleochroic haloes.

Photo 7. Sample M5 Thin Section x125

Dark cassiterite grain (arrowed) with magnetite and green

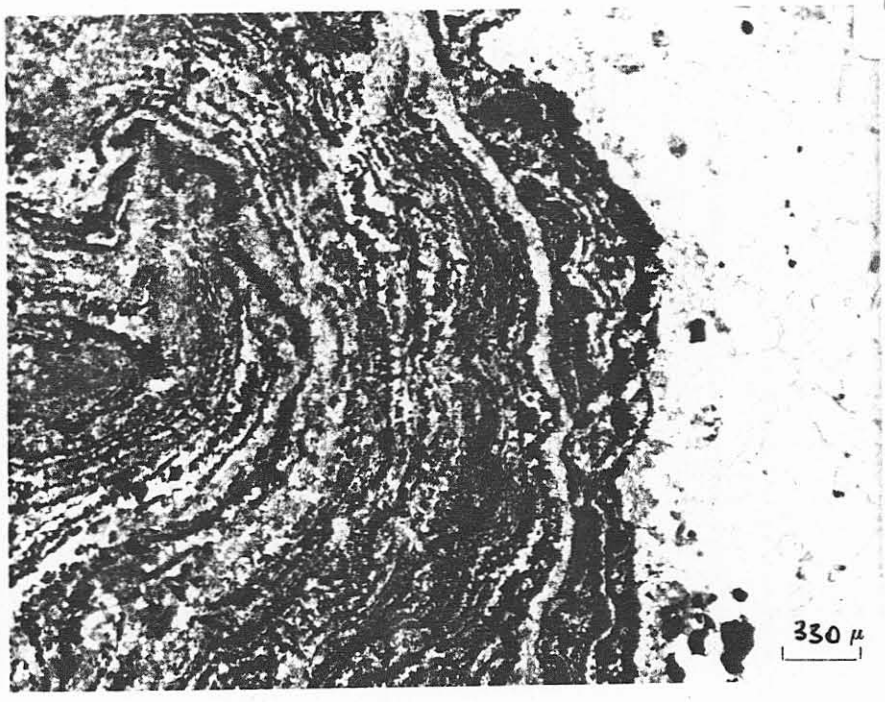
APPENDIX 3

biotite, fluorite. Photo purposely overexposed to show cassiterite.

Photo 8. Sample M3 Polished Section x100

Illustrates fine magnetite, some showing colloform and atoll textures, forming bands. Darker and lighter grey areas are fluorite and silicates, magnetite is very pale grey.

1



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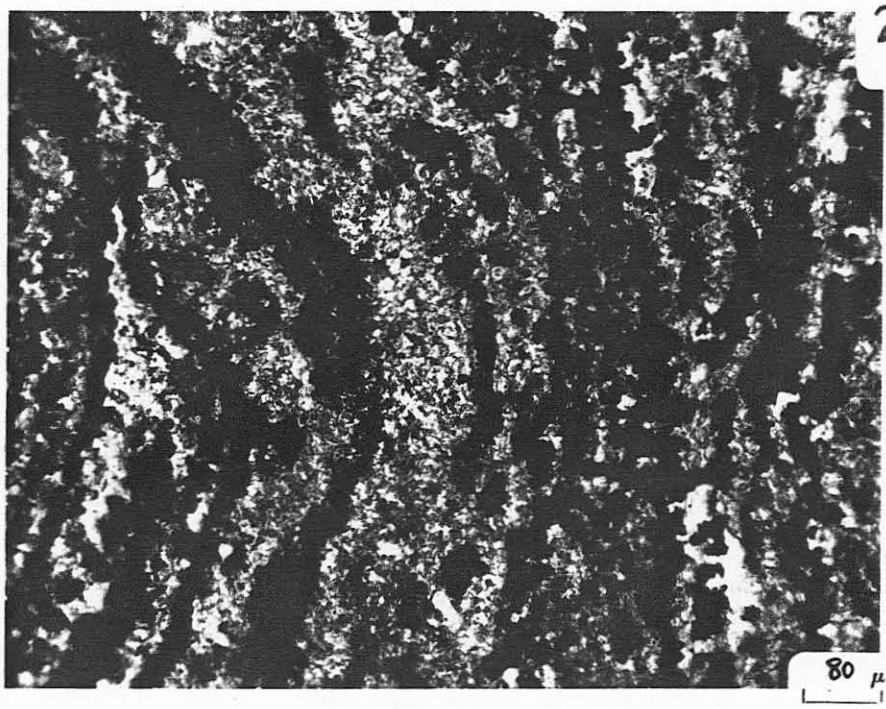

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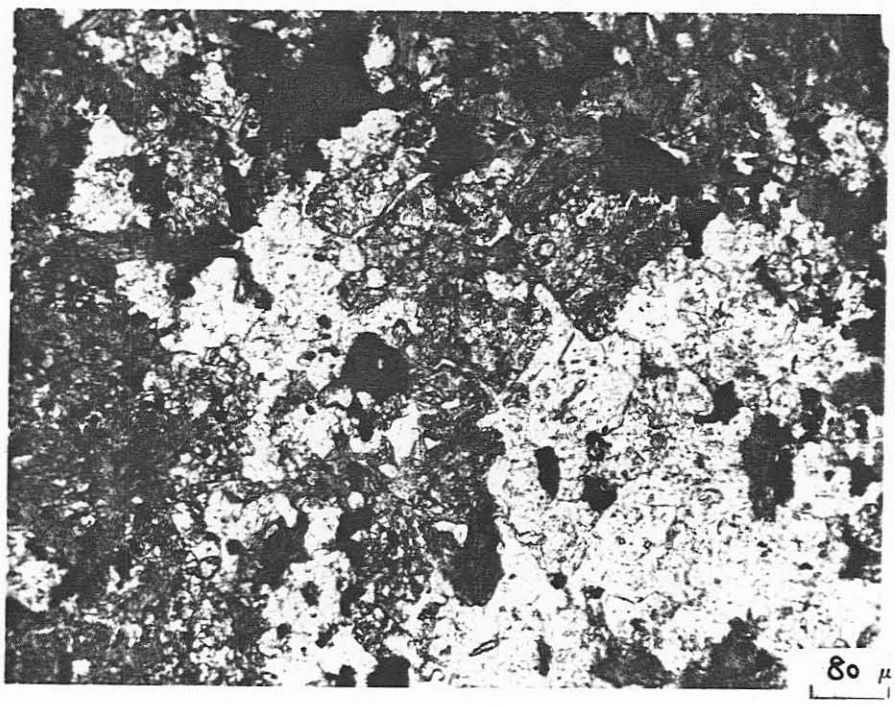

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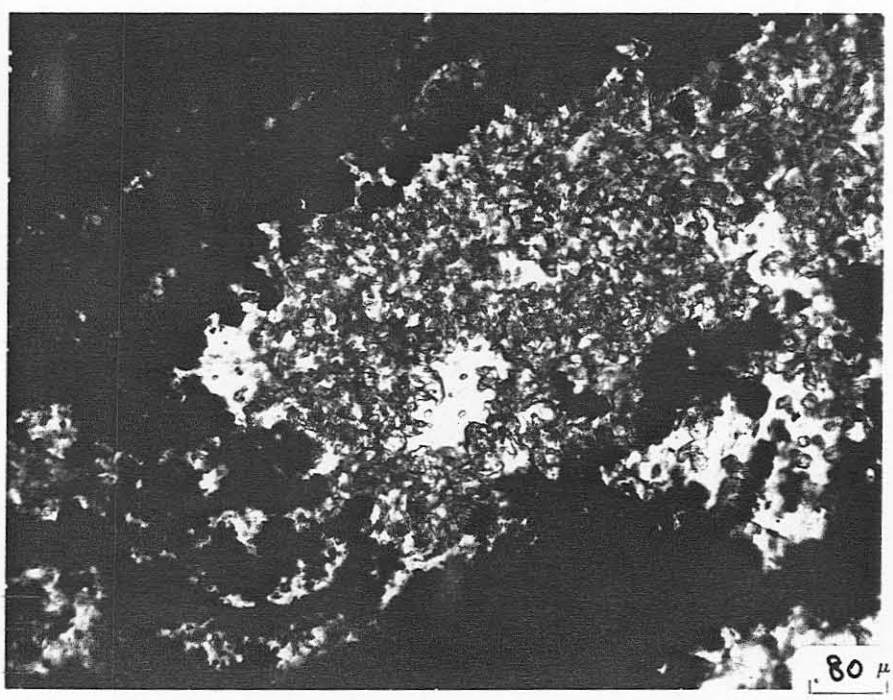

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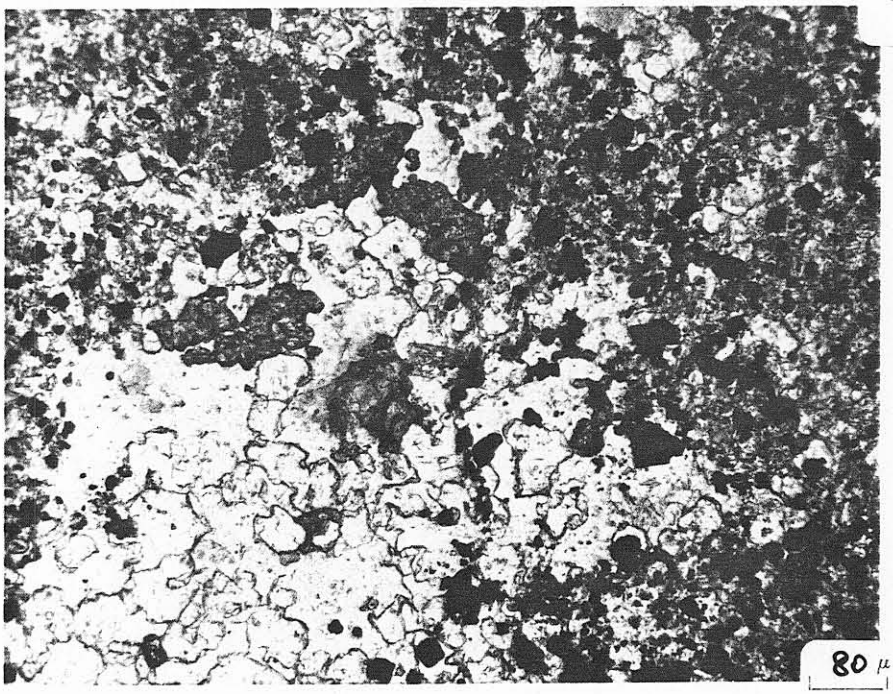

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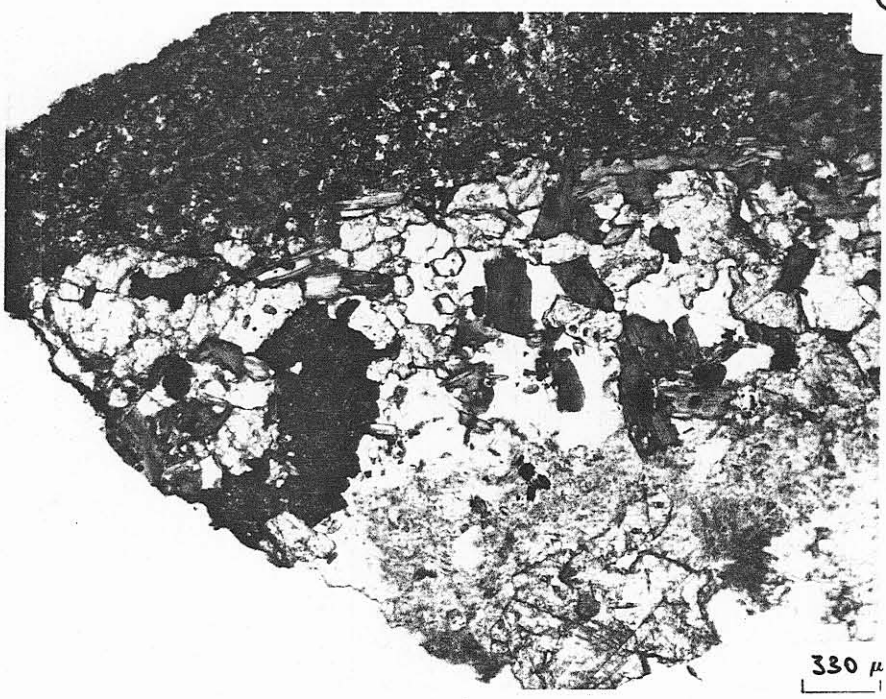

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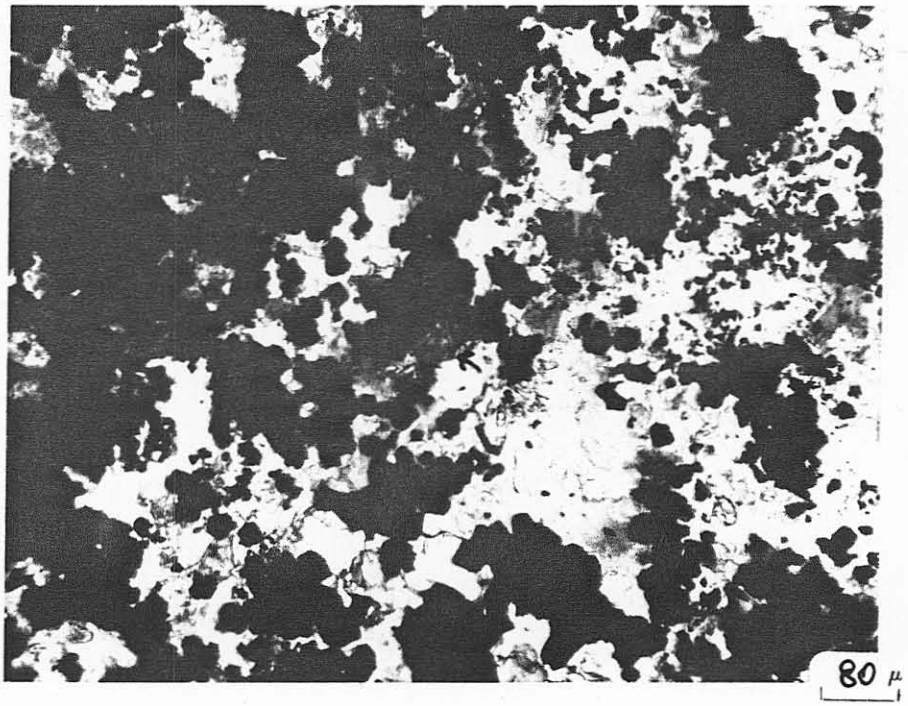

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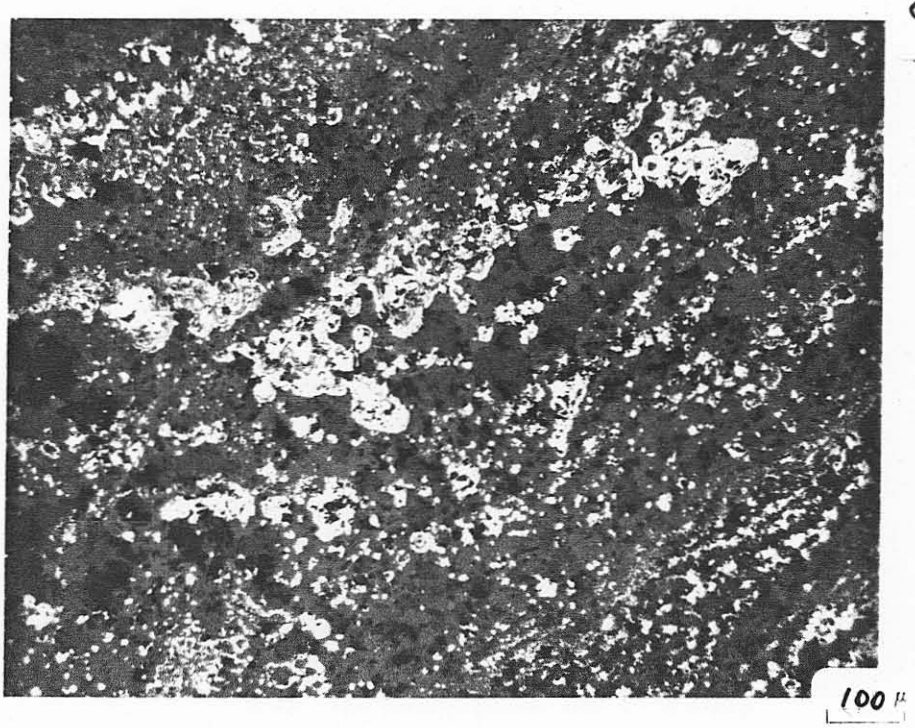


110

APPENDIX 4

221111

Central Mineralogical Services Pty. Ltd.

231 Magill Road
Maylands, S.A. 5069
Telephone 42 5659

25th November 1976

Mr. P. Askins,
Comalco Limited,
P.O. Box 691,
DEVONPORT,
TASMANIA. 7310

REPORT CMS 76/11/25

YOUR REFERENCE:	PA/TAS/MOINA
DATE RECEIVED:	18th November 1976
SAMPLE NOS.:	LOR1, LOR2 and ML3A 388
SUBMITTED BY:	P. Askins
WORK REQUESTED:	Petrology

H.W. Fander.

H.W. Fander, M.Sc.

111

221112

CENTRAL MINERALOGICAL SERVICES PTY. LTD.

Date 25th November 1976

SAMPLE REPORT (Mineralogy, Petrology, Ore Microscopy)

IDENTIFICATION
ML3A 388
MINERALISED CHLORITE-MAGNETITE ROCK (WITH CASSITERITE)

Job No. CMS 76/11/25 Date Received: 18.11.76

Reference PA/TAS/MOINA

Sample No. ML3A, 388 ft.

Nature of Sample: D.D. Core

DESCRIPTION **SECTION No.** 20234

a. Hand Specimen:

Black, fine-grained rock with sulphides. Weakly magnetic.

Microscopic:

Essentially this is a chlorite-magnetite rock, with subordinate fluorite, pyrite and ?cassiterite. Its fabric is reminiscent of a fine sediment, and it may well be a metasomatised argillite. It is not known to what extent the features outlined or accentuated by fine magnetite (folding, bedding/lamination) are primary and depositional, but seems logical to assume that they are.

The rock consists mainly (85-90%) of fine, pale green, matted chlorite flakes with random orientation. The chlorite may represent recrystallised chemically deposited material or may replace primary clay. Magnetite occurs as very small single crystals (mostly < 10μ) and small clusters, as lenses and thin, folded laminae (or laminae outlining pre-existing folds?). Small, very irregular patches of purplish fluorite are scattered through the rock; individual patches, which have an almost fibrous habit, seldom exceed 100μ in size, and may be pseudomorphous. Fine carbonate is generally associated with the fluorite, as well as microgranular crystals of cassiterite (3μ-30μ) singly and in small (up to 30μ) clusters. At these grainsizes, cassiterite is not definitely distinguishable from sphene, and thus an Sn assay is essential. Cassiterite is quite plentiful in places, but the total cassiterite content of the rock is not high (?<1%)because of its fine grainsize.

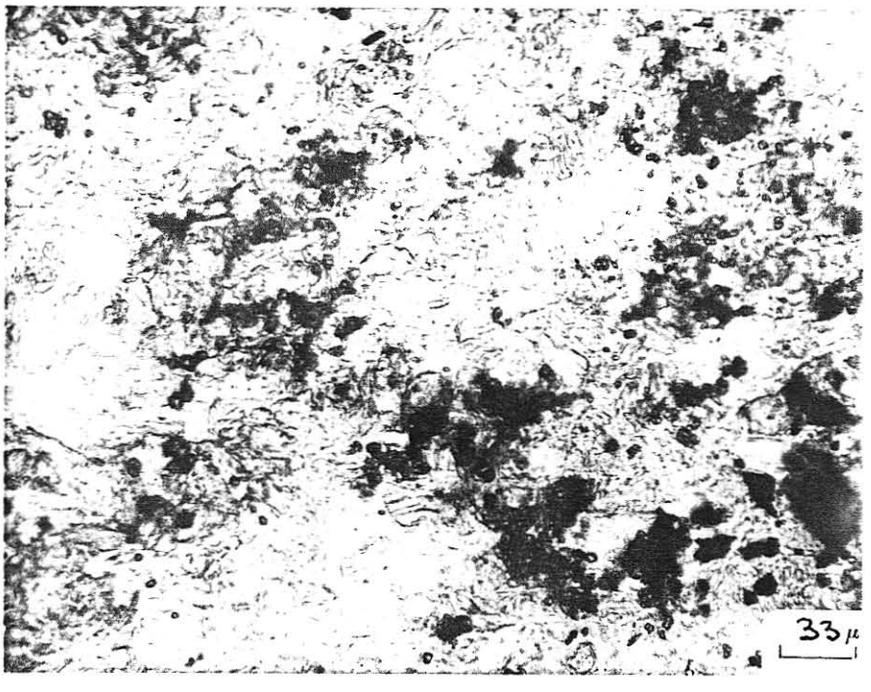
There is a close genetic association between the fluorite and the cassiterite, and in fact the whole phase of metasomatism of the rock.

Small patches of semi-opaque leucoxene occur and cause slight pleochroic haloes in the chlorite.

Opaque minerals include magnetite with thin hematite rims, shapeless patches of pyrite, subhedral to euhedral arsenopyrite crystals (the most abundant sulphide), traces of fine chalcopyrite, traces of bismuthinite, and small flakes (up to 10 x 50μ) of molybdenite flanking coarser chlorite veins. Some of the coarser magnetite shows atoll textures.

H.W. Fander, M.Sc.

The photomicrograph (x300) shows granular cassiterite, minor magnetite (black euhedra), poorly defined fluorite (low relief) and a chlorite matrix.


33μ

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APPENDIX 5

KNOWN WRIGGLITE OCCURRENCES IN THE WORLD

At least 14 wrigglite occurrences including Moina are reported throughout the world. In approximate order of reporting, the localities are:-

1. Pitkäranta, U.S.S.R.
2. Lost River, Alaska, U.S.A.
3. Kristiania, Norway
4. Southern China, (People's Republic)
5. Iron Mountain, New Mexico, U.S.A.
6. Kazakh, U.S.S.R.
7. Siberia, U.S.S.R.
8. Chugoku district, Japan
9. Central Asia, U.S.S.R.
10. Far East U.S.S.R.
11. Primor'ye district, U.S.S.R.
12. Mt. Garnet, North Queensland, Australia
13. Moina, Tasmania, Australia
14. Mt. Bischoff, Tasmania, Australia

A very brief resumé of each occurrence follows and some summary data is on Table 5.

Pitkäranta, U.S.S.R. Trustedt (1907); Eskola (1951).

Precambrian Rapakivi granite, a porphyritic leucocratic granite, with which tin tungsten mineralization is associated, is intrusive into Precambrian dolomite, marble, amphibolite and schists, in the area around the Lodoga Sea. At the time of discovery this area was part of Finland. In one mineralized field, Hopunvaara, pipes and veins of wrigglite, resembling branches of a tree, occur in marble within 30 m of the granite contact. Other skarn types also occur in the area. The wrigglite is texturally similar to that of Moina, and it consists of magnetite, fluorite, and idocrase or garnet.

Lost River, Alaska. Knopf (1908); Steidmann & Cathcart (1922). Sainsbury (1964 a,b, 1969), McOuat (1973).

The Lost River area is the site of the potentially largest fluorite mine in the world - reserves of over 38 million tonnes exceeding 16% CaF_2 and about 0.1% Sn have been blocked out. Cretaceous stocks

of biotite granite intrude thick sequences of pre Ordovician and Ordovician limestone bearing sediments. A greisenized rhyolite porphyry dyke, about 2-4 m thick and 3000 m long (the Cassiterite Lode) ends in a granite cupola in the lower levels. The enclosing limestone contains a stockwork of veins and veinlets of fluorite and of magnetite-fluorite-pyroxene and of fluorite-sericite. The last two groups of veins have fine wrigglite-like texture.

Wrigglite has been recorded at Tin Creek, a locality where a contact of granite and limestone is exposed. The limestone is marmorized, and wrigglite, consisting of fluorite-hornblende-idocrase-plagioclase-magnetite, occurs as interfering pipes and veins.

Kristiania, Norway. Twelvetrees (1913).

Twelvetrees in a report on the Moina area refers to communications with V.M. Goldschmidt and a reference Goldschmidt V.M. (1911) "Die Kontaktmetamorphose im Kristianiagebiet" p.213. Goldschmidt (in Twelvetrees, 1913) says of the Moina skarns: "Rocks similar to this, consisting of fluorite and iron ores, occur also in the Kristiania region, and originate from limestone through pneumatolytic contact-metamorphism. They are genetically related to the skarn rocks."

No other references to the geology and mineralization of the area have been consulted, but the description is consistent with the occurrence of wrigglite in this area.

Southern China. Meng, (1937), Hsieh (1963), Beus (1966).

A large section of south-east China has thick limestone sequences of Devonian to Carboniferous age, intruded by biotite granites. Contact metasomatic deposits containing cassiterite occur as bedded to lenticular bodies, veins, pipes and stockworks. At Hsianghua-ling in Hunan province and Kochiu in Yunnan province, in the immediate roof parts of the limestone in contact with the granite, or in limestone blocks engulfed in granite, small amounts of wrigglite are reported. It is described to have a similar texture to the Moina wrigglite, and consists of magnetite, lithium mica, fluorite, garnet, epidote and spinel.

Iron Mountain, New Mexico. Jahns, (1944).

A thick section of Carboniferous limestone is intricately cut by Miocene rhyolite and aplitic granite plugs sills and dykes. Massive bedded magnetite-andradite skarn and wrigglite are metasomatic alterations of limestone; and calc-silicate rocks are alterations of less pure calcareous rocks. Two types of wrigglite are described: a rarer dark variety, of magnetite-fluorite-helvite, and a lighter variety, of fluorite-biotite-chlorite-idocrase-magnetite-hematite. The rarer variety occurs in marble as thick pods, pipelike masses and tabular masses one to several metres thick.

The common variety occurs as roughly lenticular bodies parallel to the strike of the adjacent beds and has a tendency to be between massive magnetite-andradite skarn and the nearest intrusive contact. Both varieties have textures identical to the Moina type.

Chugoku District, Japan. Miyake (1965).

At the Mihara mine, presumably a copper producer, skarns occur replacing Paleozoic limestone in a sequence of limestone and shale. No granite is mentioned by Miyake but he refers to pre-skarn diorite, "porphyrite", and quartz-porphyry, which may be related to granite. These igneous rocks are classed as pre-skarn because they are altered, but they are possibly penecontemporaneous. "Porphyrite" is locally altered to garnet-idocrase-scawtite rock, continaing some cuspidine. The main skarn mass is 6 m wide, has a strike length of 130 m and dips steeply to a depth of 250 m. It tends to be chalcopyrite bearing in the upper part, pyrrhotite bearing in the middle part and magnetite bearing in the lower part. A photograph of wrigglite, composed of garnet, diopside, fluorite, actinolite, magnetite, is included in the paper, and this presumably comes from the lower part of the skarn.

At the Takinomaru mine there is a skarn zone 30 - 50 cm wide at a contact of limestone with granite. There are no definite boundaries between skarn and granite; the granite is altered to grossular rick skarn and the limestone is replaced to an unstated degree by wrigglite.

At the Yamato mine various skarns are reported, but not wrigglite. Chalcopyrite, molybdenite and scheelite ores are stated to occur. It seems possible therefore that the area is a metallogenic province of tungsten.

Kazakh, U.S.S.R. Beus (1966), Zasedatelev (1973).

Lenses of marble in Devonian sedimentary rocks are intruded by granite, which in its margins is greisenized. A zone of meta-somatized rocks about 100 m thick and 5 - 6 km long contains several different skarn types, including micaceous-fluorite-magnetite rocks containing idocrase. These rocks have textures similar to the Moina wrigglite. Table 3 shows analyses of these rock types.

Siberia, U.S.S.R. Getmanskaya (1972).

In this area bedded dolomitic limestones, calcareous shales and micaceous shales of Lower Paleozoic age are intruded by Jurassic granite, which is characteristically greisenized. In the exocontact zone there are rocks composed of fluorite-idocrase-magnetite, being "fine grained with an odd kind of thinly banded structure". These are doubtless wrigglites.

Central Asia, U.S.S.R. Beus (1966)

Beus (1966, p.52) lists an occurrence in "Central Asia" of magnetite-fluorite skarns with helvite, danalite, idocrase and pyroxene, but he does not describe the deposit in the text. It is possibly a wrigglite occurrence.

Far East U.S.S.R. Sainsbury (1969), Shcherba (1970)

Sainsbury (1969) and Shcherba (1970) mention an area possibly adjacent to Alaska, over the Bering Sea, where the geology is similar to the Lost River area. Aleksandrov (1975) refers to this area as the Iten'-yurginsk deposit, Chukotka and includes a photograph of wrigglite. Greisenized granite and skarns in limestones occur in a tin-beryllium metallogenic province. The fluorite rich skarns are described as "fine grained, crudely banded". They have the same composition as the wrigglites at Tin Creek, Alaska. Their thickness 'reaches tens of metres'.

Primor'ye District, U.S.S.R. Aleksandrov (1975)

Aleksandrov (1975) lists references in Russian which describe wrigglites - one is by Stepanov G.N. (1966) with a translated title of "'Finely banded idocrase - magnetite - fluorite rocks in the skarns of one of the regions of Primor'ye', Geochemistry and Mineralogy of Magmatic Products, Vladivostok". This district is probably Dal'negorsk, where skarns replace limestone. Very little other useful information is given. It is a tin mining district.

Mt. Garnet, Queensland Askins (1975)

Roof pendants composed of Silurian siltstones, sandstone, basalt and lenses of limestone occur in Carboniferous leucocratic granite, with which tin-tungsten mineralization in the region is associated. The granite is extensively greisenized and characteristically contains accessory fluorite. A complex array of skarn types, and calc-silicate rocks occurs. Wrigglite occurs as a replacement of marble lenses within 70 metres of granite contacts. The lenses seldom exceed 300,000 tonnes in size. The wrigglite in general is coarser grained than the Moina type, and the most common mineralogy is magnetite-fluorite-idocrase-spinel.

Mt. Bischoff, Tasmania Groves et al (1972)

Groves et al (1972) describe the Mt. Bischoff deposit in detail but no mention is made of the unusual rhythmically banded rocks which occur prominently outcropping in one of the open pits. A description of the rock by Fander is appended. The rock is clearly a variant of wrigglite, where sellaite (MgF_2) occurs instead of fluorite, pyrrhotite occurs instead of magnetite, and tourmaline occurs instead of calc-silicate minerals.

At Mt. Bischoff a ? Cambrian dolomite bed in siltstones is partly replaced by cassiterite bearing pyrrhotite. The sequence is intruded by aplite dykes which are probably related to a Devonian granite occurring at depth.

(APPENDIX 5 CONTINUED)

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REPORT CMS 76/12/10

SAMPLE PA/TAS/MB1 MOUNT BISCHOFF

Thin and polished sections were prepared and examined and two photomicrographs were prepared to illustrate some of the main features.

The rock is of considerable interest and may be of economic significance because of the occurrence of tin minerals. Total Sn content is in the 0.5 - 1% range, though the metallurgy would not be easy.

Essentially the rock is crustiform-banded, with alternating bands of contrasting mineralogy. This geological setting is particularly significant because of the occurrence of cassiterite in this context.

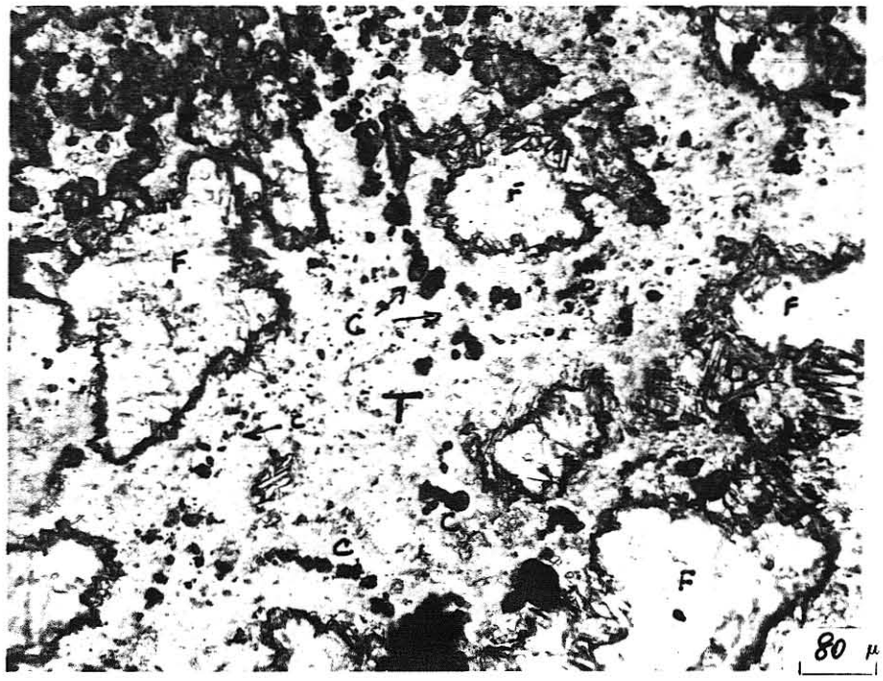
The bands are composed of sellaite (MgF_2), elbaite (alkali tourmaline), sulphides and fluorite, with minor amounts of carbonate and some late-stage iron staining due to oxidation of sulphides. The bands are more or less monomineralic, with limited marginal intergrowths and composite patches. Of the minerals listed, sellaite and elbaite are the major constituents; this in itself is unusual, because of their rarity.

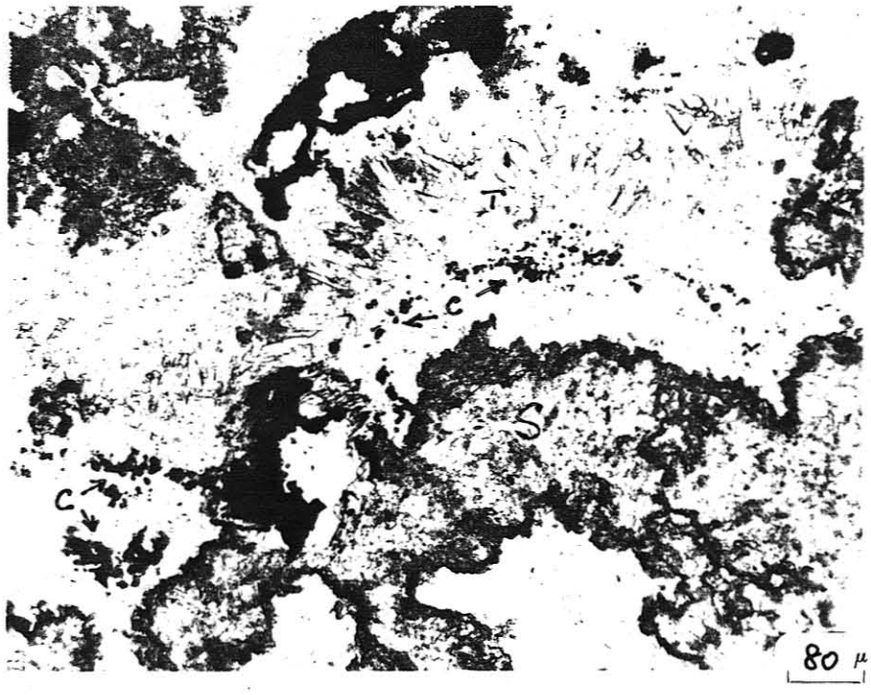
Cassiterite is more abundant than would be suspected at first; because of its fine grainsize, and the abundance of sellaite (its low R.I. of 1.38 is responsible for grains with dark borders due to low relief, especially where embedded in tourmaline), it is difficult to detect. Much of the cassiterite occurs as individual crystals and small "stringers" (see photos) parallel to the banding, and is mostly associated with tourmaline and pyrrhotite but not exclusively so. Cassiterite crystals range from 1 - 2μ to 30μ; maximum measured crystal size was 50μ.

Sulphides are dominantly pyrrhotite, showing all stages of classical, text-book style, alteration to marcasite/pyrite. There are masses of granular pyrite/marcasite intergrowths, all probably derived from pyrrhotite. Traces of sphalerite and stannite occur, commonly associated and intimately intergrown; composite stannite-sphalerite crystals up to 1mm across are occasionally seen. The stannite is believed to be the "isostannite" variety (isotropic) in part; this could explain its close association with sphalerite.

In the photomicrographs (mag. = 125x), S = sellaite, F = fluorite, C = Cassiterite, T = tourmaline.

H.W. Fander, M.Sc.


F
F
F
F
C
T
C
80 μ


T
C
S
C
80 μ

APPENDIX 6

ORIGIN OF LAYERING IN WRIGGLITES

Several different hypotheses for the development of the rhythmic layering in wrigglites have been advanced. These are that the layers

(1) represent bedding in an unusual sediment, which has been subsequently metamorphosed. (Zasedatelev, 1973)

(2) are replacement, layer by layer, of pre-existing stromatolites. (Stainton, pers comm)

(3) are colloform layers, representing a solidified gel precipitate, which has replaced limestone (Stevenson & Jeffery, 1964)

(4) are crustiform layers, rhythmically deposited in cavities in the limestone. (Trustedt, 1907)

(5) are deposited by replacement of limestone at diffusion "fronts" moving outwards from a fracture along which mineralizing fluid infiltrates. (Knopf 1907, Jahns 1944, Eskola 1951, Beus 1966, Sainsbury 1969).

The last hypothesis is the one which best fits the observed facts and it is the only process capable of building up the observed geometry. (Fig.8). The mechanism in more detail is:-
Fluids infiltrate along fractures and pipe-like openings in limestone. The components in the fluid move into the limestone diffusion along grain boundaries, down a concentration gradient. The fluid, assumed to be acidic, fluoride rich, and containing iron, silicon, aluminium, tin, tungsten, etc. as fluoride complexes, reacts with the calcite. Layers of fluorite, magnetite and idocrase etc. deposit by volume-for-volume replacement of limestone. The fluid continues to diffuse from the fracture, through these deposited layers, and a new set of layers forms in the same way. The process continues until limestone is entirely replaced by wrigglite or until the effective limiting distance of diffusion from the fractures is reached, or until the fluid infiltrating along the fractures is spent. The process is analogous to the formation of Liesegang Rings. There are irregularities in the advancing diffusion fronts caused by slightly differing rates of diffusion from one place to another - this produces local thickening and thinning and the irregular contortions in the layers. The last phase in the process is the filling of the fracture itself, to produce usually a coarser grained central vein or coarser grained patch of fluorite and other minerals. This central vein might contain late stage minerals or might be the locus for alteration of wrigglite by later fluids of different composition. The geometry of interfering sequences produced by this process is illustrated in Fig. 8.

Some of the evidence against the other processes postulated is:-

(a) partial replacement of limestone by wrigglite occurs as veinlets or pipe shapes, which precludes a sedimentary origin. The limestone being replaced contains no stromatolites, so stromatolites are not a necessary intermediary in the process;

(b) if the wrigglite was once a gel, the rock at that time would have had nearly zero strength and would have collapsed under the load of the overlying rocks. A gel process on a large enough scale to form a whole rock like wrigglite is therefore not possible;

(c) Crustiform layers built up in cavities would not occur if there were so many cavities that the rock collapsed under the load of the overlying rocks. This means that there would need to be a very involved specialized process of cavity making, filling with rhythmic crustified layers from a fluid with rhythic changes in composition, then new cavity making and filling and so on, to ensure total replacement of limestone by wrigglite. The possibility of such a process operating on a large scale is very remote.

(Caption to Fig.8)

FORMATION OF WRIGGLITE BY DIFFUSION PROCESS.

Fractures, through which mineralizing fluid infiltrates, traverse limestone (A). At right angles to the direction of infiltration, components of the fluid diffuse into the limestone and replacement takes place to build up layers of magnetite, fluorite, etc, as outlined in the text. A later fracture, heavy lines in (C), passes through both limestone and previously formed wrigglite. Only limestone is replaced when components diffuse from this new fracture, and so the geometry in (C) is built up.

Many of the fractures are pipe-like in shape, (D). The wrigglite so formed from pipe P_1 by diffusion is like that in (E). Subsequently formed wrigglite from pipe P_2 impinges on the previously formed wrigglite to form "unconformable" geometric relationships, (F).

The geometry of (G) is sometimes seen in the wrigglites. This can only form when P_1 and P_2 are contemporaneous and diffusion takes place from both simultaneously.

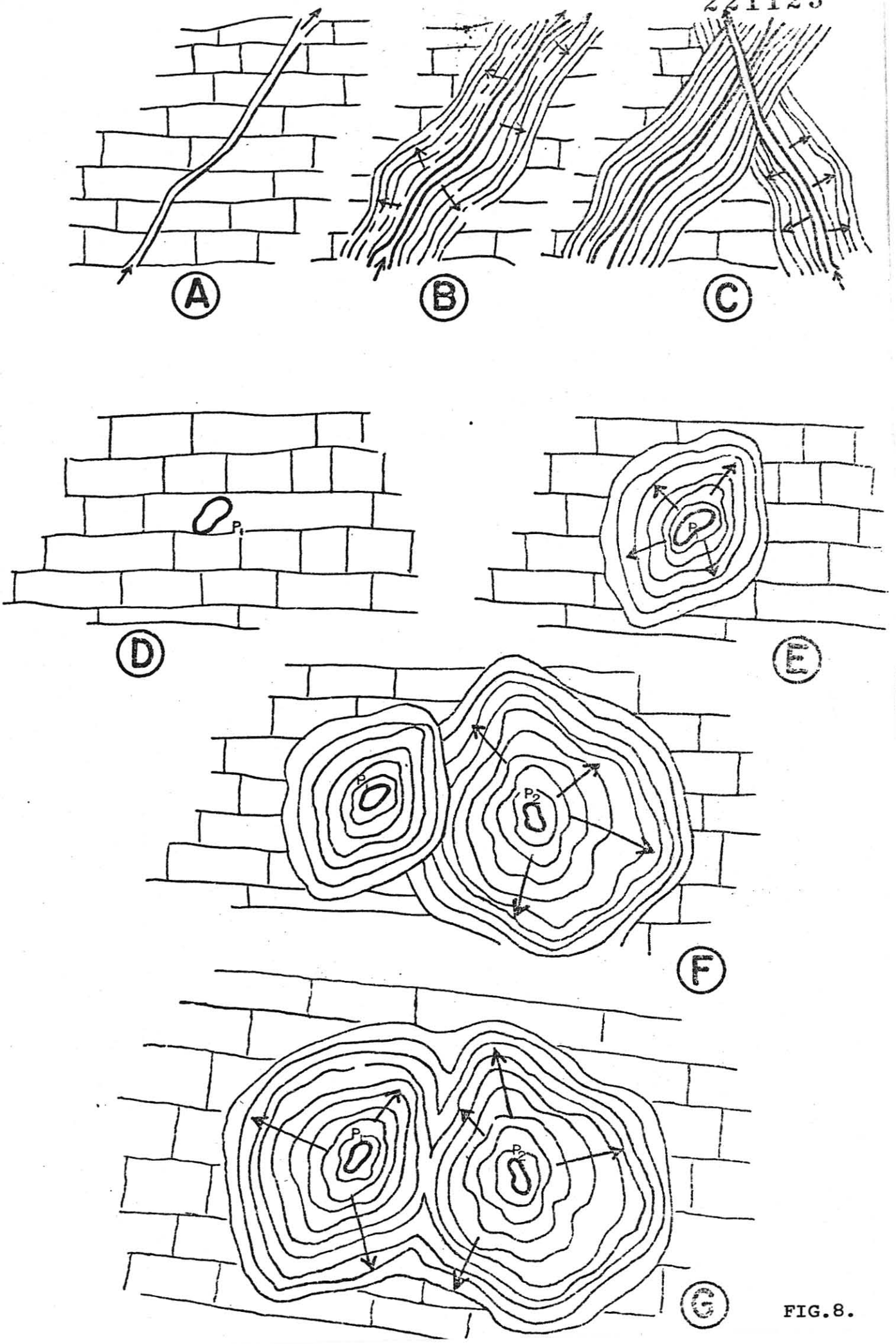


FIG.8.

APPENDIX 7

NATURE OF THE MINERALIZING FLUID

Field evidence from Moina and other world occurrences clearly indicates that wrigglite results from replacement of relatively pure calcite or dolomite, therefore the mineralizing fluid must introduce all foreign elements including H_2O, F, Fe, Al, Si, K, Mg, Sn, W and Bi.

The most significant associated alteration type throughout the workd is greisenization, in which H_2O, F, Fe, Al, Si, Sn and W are introduced into a granite. West (1974) found from studies of fluid inclusions in a fluorite lode about 30 km NNW of Mt. Garnet, Queensland, that the fluid associated with greisenization was possibly of high salinity (14-41% NaCl) and at temperatures of 260 - 380° C.

It is not possible to use data such as that of Meyer and Hemley (1967) on the systems K_2O - Na_2O - Al_2O_3 - SiO_2 - H_2O, to deduce the fluid composition during the alteration stages of greisenization, because greisenization would involve systems with high F concentrations, which would have an unknown quantitative effect on stability relationships. It is clear however that the fluid had a high F^- and H^+ activity and therefore possibly carried fluoride complexes like $Sn(OH,F)_6^{2-}$.

For reference list see Appendix 5.

APPENDIX 8

Analytical techniques used by Comalco, Amdel.

A. COMALCO LABORATORIES (COMALCO RESEARCH CENTRE).

1. STREAM SEDIMENT AND SOIL SAMPLES.

Cu, Pb, Zn, Bi, Mo, Co, Ni:

Determination by AAS after dissolution of 1 g sample in hot HCl + HNO_3 (Scheme 1).

Ag, Mo, Bi:

Determination by AAS after dissolution of 5 g sample in hot HNO_3. (Scheme 2).

Fluorine: (total)

Determination by specific ion electrode after fusion of 0.25 g sample with flux of Na_2CO_3, KNO_3, K_2CO_3, SiO_2 and water leach. (Scheme 8).

2. ROCK SAMPLES.

Cu, Pb, Zn, Bi, Mo, Co, Ni, Cr:

Determination by AAS after attack of 1 g sample in hot HCl + HNO_3 + $HClO_4$, evaporated to fuming $HClO_4$, and dissolution in HCl. (Scheme 3). Where silicates are suspected the method is the same but attack is by HF, HNO_3 + $HClO_4$. (Scheme 5).

Ag, Mo, Bi, Sb:

Determination by AAS after dissolution of 5 g sample in hot HCl + HNO_3. (Scheme 4).

W:

Fusion of 0.25 g sample with $KHSO_4$, leach with HCl, add $SnCl_2$ solution, determined with Zn-dithiol indicator in a spectrophotometer, compared with known solutions. (Scheme 7).

Au:

10 g of sample roasted, leached to dryness with HCl + HNO_3, dissolved in HCl, transfer to organic solvent and determine by AAS.

Fluorine:

As for stream sediment and soil samples.

3. WATER.

Fluorine:

Read directly by specific ion electrode after addition of buffer solution.

B. AMDEL.

CODE A. Semi quantitative analyses by emission spectroscopy.

CODE B. XRF. All XRF analyses are by Code B1 unless otherwise stated.

B1 XRF, accuracy ± 5%, detection limits as quoted.

B2 XRF, accuracy ± 3%, detection limits as quoted.

CODE C. AAS

C1 Bi, Cd, Co, Cr, Cu, Fe, Mn, Ni, Pb, Zn, accuracy ± 5%, detection limits as quoted. Dissolution in hot $HClO_4$.

C2 Ag, Mo, accuracy ± 5%. Dissolution in hot HCl + HNO_3?

C3 Au, accuracy dependent on sample, detection limit 0.05 ppm. Details of method not available, but dissolution in HCl + HNO_3.

CODE E. E4 F as CaF_2. Method is not available but is suspected to be leaching in near boiling aluminium sulphate and determination by specific ion electrode. The method is more or less specific for CaF_2 and does not give total F in the sample i.e. F tied up in micas, topaz, amphibole, etc).

CODE K. K4/2 Au by fire assay and AAS, detection limit 0.005 ppm.

APPENDIX 9

SAMPLES - MOINA, TASMANIA

221129

collected by J. Gove 1974

Sample No.		Comalco Laboratory				AMDEL Analysis
		(F%)	Cu (ppm)	Pb (ppm)	Zn (ppm)	
03/0701/ M-01	Grey limestone (chip sample) - ¼ mile west of Moina, near road	0.13	10	50	30	Yes
M-02	As for M-01	3.6 *	10	40	110	No
M-03	Chip sample of wrigglite from 3 localities at S. and M. Mine. Often mineralized with pyrite, silvery sulphide, minor chalopyrite	6.5	365	50	6000	Yes
M-04	Tailings (?) - S. and M. Mine. Mostly quartz with some wrigglite	2.4	50	40	1100	Yes
M-05	Garnet rich skarn with very minor disseminated sulphides - S. and M. Mine	0.92	45	110	370	Yes
M-06	Calc-silicate (skarn?) rock - with disseminated sulphides - S. and M. Mine	0.15	25	40	20	Yes
M-07	As for M-06	0.12	15	30	45	No
M-08	Banded brown garnet-green pyroxene (?) rock with pink feldspar veinlets and minor disseminated sulphides (?) - S. and M. Mine	0.035 *	10	45	10	Yes
M-09	Wrigglite with abundant pyrite (and disseminated Sn/W ?) - S. and M. Mine		Not	Analysed		Yes
M-10	As for M-09	5.2	1090	55	2380	No
M-11	Wrigglite with disseminated Sn/W(?) - S. and M. Mine		Not	Analysed		Yes
M-12	As for M-11	10.1	10	75	65	Yes
M-13	Magnetic - Garnet (?) skarn with pyrite	4.0	110	50	3880	Yes

* these samples could have been interchanged in error

AMDEL ANALYTICAL SE[illegible]CE

JOB: ..3176/74...

Semi-Quantitative Spectrographic Analysis Schemes A1,A2,A3,A4,A5 & A6 BATCH .....1..... 221130

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Form 60

Results in ppm unless otherwise stated. Detection limits in brackets

Sample No	05/10/01 M-01	M-04	M-06	M08	M-09	M-11	M-13	Sample No.							
A1								A2 Contd.							
Co (5)								Ge (1)							
Ni (5)								As (50)							
Cr (20)								Sb (30)							
V (10)								A3							
W (50)	x	300	x	300	500	1,000	300	Te (20)							
Mo (3)	x	50	30	3	5	3	5	Tl (1)							
Mn (10)								P (100)							
Ta (100)								A4							
Nb (20)								Na (50)							
Be (1)	x	10	1	10	80	50	30	Li (1)							
Th (100)								A5							
Pt (10)								K (5)							
Pd (10)								Rb (10)							
Os (10)								Cs (30)							
Ir (2)								A6							
Rh (2)								Ba (50)							
Ru (2)								Sr (10)							
A2								Y (10)							
Cu (0.5)	10	50	30	10	1,000	20	500	La (100)							
Pb (1)	50	10	100	80	80	50	50	Ce (300)							
Zn (20)	x	200	100	200	500	100	2,000	Nd (300)							
Sn (1)	30	250	1,000	300	1,000	800	1,000	Pr (100)							
Cd (3)								Ti (100)							
Bi (1)	1	30	20	50	800	250	50	Er (100)							
Ag (0.1)	0.1	x	0.2	0.2	0.2	0.1	0.2	Sc (50)							
Au (3)	x	x	x	x	x	x	x	Eu (50)							
Ga (1)															

Results are semi-quantitative. Elements apparently present in concentrations of economic interest should be redetermined by an appropriate accurate analytical technique. X = Not detected at limit quoted

[illegible] (%10) = 90

AMDEL ANALYTICAL SERVICE

JOB: ...3179/74..

Semi-Quantitative Spectrographic Analysis Schemes A1,A2,A3,A4,A5 & A6 BATCH .....3.....

Form 60

Results in ppm unless otherwise stated. Detection limits in brackets

221131

Sample No	M-03	M-05						Sample No.							
A1								A2 Contd.							
Co (5)								Ge (1)							
Ni (5)								As (50)							
Cr (20)								Sb (30)							
V (10)								A3							
W (50)	1,000	X						Te (20)							
Mo (3)	1,000	3						Tl (1)							
Mn (10)								P (100)							
Ta (100)								A4							
Nb (20)								Na (50)							
Be (1)	30	3						Li (1)							
Th (100)								A5							
Pt (10)								K (5)							
Pd (10)								Rb (10)							
Os (10)								Cs (30)							
Ir (2)								A6							
Rh (2)								Ba (50)							
Ru (2)								Sr (10)							
A2								Y (10)							
Cu (0.5)	300	10						La (100)							
Pb (1)	80	100						Ce (300)							
Zn (20)	3,000	500						Nd (300)							
Sn (1)	1,000	50						Pr (100)							
Cd (3)								Ti (100)							
Bi (1)	400	30						Er (100)							
Ag (0.1)	0.1	0.1						Sc (50)							
Au (3)	X	X						Eu (50)							
Ga (1)															

Results are semi-quantitative. Elements apparently present in concentrations of economic interest should be redetermined by an appropriate accurate analytical technique. X = Not detected at limits

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THE A' 'RALIAN MINERAL DEVI PMENT LABC TORIES

REPORT AN 3176/74

Page 2

221132

QUALITATIVE SPECTROGRAPHIC ANALYSIS

Sample No.	Major	Minor	Heavy Trace 1,000 – 10,000 ppm (0.1—1%)	Trace 100 – 1000 ppm (0.01—0.1%)	Faint Trace 10 – 100 ppm (0.001—0.01%)	Very Faint Trace 1 – 10 ppm (0.0001—0.001%)
03/0701 M-03	Si, Fe, Al, Ca,	Mg,	Na, Zn, K, Ti,	W, Sn, Bi, Cu, Li, Mo, Mn,	Zr, Pb, Cr, Co, V, B, As, Ni, Cd, Sr, Be, Ga,	Ge, Ag, Y,
03/0701 M-05	Si, Al, Ca,	Fe, Mg,	Mn, Ti,	Zn, Cr, K,	Pb, Zr, V, Na, Li, Ni, Bi, P, Sn, Y,	Co, Be, B, Mo, Cu, Ga, Ag,

Elements not sought: Hg, Re,

Other elements not detected at limits quoted in attached sheet: Ta, Nb, Th, Pt, Pd, Os, Ir, Rh, Ru, Au, Sb, Te, Tl, P, Rb, Cs, Ba, La, Ce, Nd, Pr, Er, Sc, Eu, Tb, As, Ge, Sr, W,

Complete Qual 2x.

SPECTROGRAPHIC ANALYSIS

Detection-Limit Concentrations of Elements
DC Arc Excitation

Element	%	ppm	Element	%	ppm
Ag	0.00001	0.1	Na	0.005	50
Al	0.02	200	Nb	0.02	20
As	0.005	50	Nd	0.03	300
Au	0.0003	3	Ni	0.0005	5
B	0.001	10	Os	0.001	10
Ba	0.005	50	P	0.01	100
Be	0.0001	1	Pb	0.0001	1
Bi	0.0001	1	Pd	0.001	10
Ca	0.002	20	Pr	0.01	100
Cd	0.003	3	Pt	0.001	10
Ce	0.03	300	Rb	0.001	10
Co	0.0005	5	Re	0.001	10
Cr	0.002	20	Rh	0.0002	2
Cs	0.003	30	Ru	0.0002	2
Cu	0.00005	0.5	Sb	0.003	30
Dy	0.01	100	Sc	0.005	50
Er	0.01	100	Si	0.02	200
Eu	0.005	50	Sm	0.03	300
Fe	0.005	50	Sn	0.0001	1
Ga	0.0001	1	Sr	0.001	10
Gd	0.03	300	Ta	0.01	100
Ge	0.0001	1	Tb	0.01	100
Hf	0.02	200	Te	0.002	20
Hg	0.01	100	Th	0.01	100
Ho	0.01	100	Ti	0.01	100
In	0.001	10	Tl	0.0001	1
Ir	0.0002	2	Tm	0.01	100
K	0.0005	5	U	0.1	1000
La	0.01	100	V	0.001	10
Li	0.0001	1	W	0.005	50
Lu	0.03	300	Y	0.001	10
Mg	0.01	100	Yb	0.005	50
Mn	0.001	10	Zn	0.002	20
Mo	0.0003	3	Zr	0.02	200

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amdel

AN 3176/74

221134

1.

ANALYSIS

%

Sample Mark	Fluorine F	Calcium Fluoride CaF_2
M-03	10.3	20.1
M-05	0.90	1.75
M-06	0.10	0.2
M-09	4.8	9.35

NOTE: The quotation of $4-00 per sample given by telephone was made on the understanding that 20 or so samples were to be submitted. Also that the values expected were of "geochemical" significance i.e. less than 5000 ppm. The values found in these samples necessitated the use of alternative methodology for which our price is $8-00 per sample.

221135

APPENDIX 10

MOINA AREA, SAMPLES COLLECTED BY G. WESTE 1975.

07/0705/01	Beside collar ML 1 calc-silicate garnet rock MW-HW.	Shepherd & Murphy Area
02	3 m SW of 01.wrigglite with bismuthinite veinlets and minor quartz veins.	
03	Near 01, calc-silicate rock ,HW.	
04	6 m W of ML 1 on track, fine grained wrigglite MW.	
05	11 m W of ML 1 on track, fine grained wrigglite MW.	
06	15 m W of 05 on track, fine grained wrigglite MW.	
07	20 m W of 06, fine grained wrigglite HW.	
08	15 m N of S & M shaft, fine grained wrigglite from waste rock dump SW.	
09	15 m W of S & M shaft, quartzite from waste rock dump SW.	
10	15 m W of S & M shaft, calc-silicate and wrigglite from waste dump MW.	
11	15 m W of S & M shaft, wrigglite from waste rock dump SW.	
12	At main S & M shaft, wrigglite SW.	
13	At main S & M shaft, wrigglite SW.	
14	10 m S of 13, calc-silicate and wrigglite MW.	
15	3 m W of 14, calc-silicate and wrigglite MW.	
16.	50 m W of S & M shaft top 3 cm of fine sediment in old water tank.	
17	50 m W of S & M shaft bottom 2 cm of fine sediment in old water tank.	
18	10 m S of Stormont Mine, calc-silicate on track MW.	Stormont
19	Few m SW of Stormont Mine, skarn on track MW.	
20	Few m NW of 19 skarn MW.	
21	Stormont cut W end massive skarn MW.	
22	SW along strike and 2 m NE across strike from 21,MW	
23	NE side of Stormont adit and 3-4 m NE across strike from 22 MW.	
24	Tin spur mine skarn.	
25	Bell Creek where it crosses HEC road to Wilmot dam wall - panned concentrate (not analysed).	
26	Bismuth Creek outlet to Lake Gairdner - panned concentrate (not analysed).	
27	Brampton Creek at crossing of Moina road - panned concentrate (not analysed).	
28	Iris River at crossing of Moina road - panned concentrate (not analysed).	
29 & 30	Boulders of calc-silicate from Bismuth Creek outlier E of Bismuth Creek, MW.	Shepherd & Murphy Area

13c

221136

07/0705/1001			Rock chip grab sample from E end S & M main crusher outcrop.
1002			Rock chip grab sample from W end S & M main crusher outcrop.
1003			Boulders of banded calc-silicate on track S of mine.
1004	0- 1 m	HW-CW	Continuous chip sample across wrigglite (+ minor calc-silicate) behind S & M primary crusher. Each sample take over 1 metre interval on bearing 170° M commencing at N end of exposure. Sampling carried out to observe effect of weathering on F content of wrigglite.
1005	1- 2		
1006	2- 3		
1007	3- 4		
1008	4- 5		
1009	5- 6		
1010	6- 7	HW	
1011	7- 8		
1012	8- 9		
1013	9-10	MW	
1014	10-11		
1015	11-12		
1016	12-13		
1017	13-14		
1018	14-15		
1019	15-16		
1020	16-17		
1021	17-18		
1022	18-19	SW	
1023	19-20		
1024	20-21		
1025	0- 1 m		Continuous chip sample traverse through SW hard wrigglite exposed 1 metre S of S & M main shaft W to E traverse at about 60° M.
1026	1- 2		
1027	2- 3		
1028	3- 4		
1029	4- 5		
1030	5- 6		
1031	0- 1 m		Continuous W to E chip sample traverse located immediately to E of S & M main shaft at 30° M. Outcrop poor. Rocks MW – HW.
1032	1- 2		
1033	2- 3		
1034	3- 4		
1035	3- 5		
1036	5- 6		
1037	6- 7		
1038	7- 8		
1039	8- 9		
1040	9-10		
1041	10-11		

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221137

Sample No.	F ppm	W ppm	Bi ppm	Mo ppm	Au ppm	Ag ppm	Cu ppm	Pb ppm	Zn ppm	Ni ppm
07/0705/01	3600	60	25	10	ND	3.3	-10	30	100	55
02	8.80%	1100	645	10	-0.10	3.3	145	30	185	70
03	4400	ND	115	-10	-0.10	2.2	-10	35	65	65
04	11.50%	775	475	10	0.13	3.3	-10	30	330	55
05	9.0 %	120	95	-10	0.15	2.2	15	25	345	35
06	12.2 %	870	680	10	0.15	3.3	190	25	250	20
07	19.0 %	460	480	-10	0.11	1.8	120	30	360	20
08	12.8 %	2145	285	35	-0.10	2.8	10	30	120	55
09	0.8 %	120	15	ND	-0.10	ND	10	20	25	155
10	8.5 %	1240	235	-10	0.10	2.2	-10	30	430	60
11	8.4 %	3120	50	45	-0.10	2.8	15	30	240	70
12	13.0 %	220	415	20	0.19	2.2	20	35	200	20
13	9.0 %	720	1040	-10	0.16	2.4	115	25	500	25
14	6.6 %	1630	80	-10	0.22	1.7	-10	25	145	75
15	2.0 %	415	60	-10	0.11	3.5	-10	20	340	45
16	3.4 %	3680	435	20	0.11	2.2	145	135	1160	65
17	0.6 %	415	135	-10	-0.10	1.8	45	385	340	145
18	4600	70	165	-10	-0.10	1.1	-10	30	70	65
19	2200	25	125	ND	0.10	1.3	-10	35	115	45
20	2000	160	270	ND	-0.10	1.1	10	25	90	70
21	1400	ND	275	ND	2.90	1.8	-10	85	70	60
22	1100	ND	2635	ND	2.57	4.0	-10	865	50	45
23	500	10	800	ND	0.23	2.4	-10	365	50	45
24	400	ND	30	ND	-0.10	ND	10	25	15	155
07/0705/29				-10	-0.10	0.36	-10	20	140	Sb 10
30				560	-0.10	0.60	-10	10	90	Sb 15

NOTE: -10 means less than 10

137

Sample: Expl^n - Rock - 07/0705/- Report No.: AK-30.

Submitted by: G.W. Requisition No.: 1815 Comm. Cont. Conc.

221138 MOINA, TAS.

ANALYSIS

	SAMPLE	F %	Zn ppm	Pb. ppm	Cu. ppm	Bi. ppm	Mo ppm	W ppm
1	0705/1001	10.0	215	15	75	65	1000	2210
2	/1002	3.4	1.00%	30	990	1060	35	150
3	/1003	1.5	510	20	15	150	15	30
4	/1004	0.5	660	30	370	960	30	107
5	/1005	0.4	265	25	210	455	15	690
6	/1006	0.3	470	30	790	830	15	670
7	/1007	1.0	590	25	860	1300	25	610
8	/1008	2.0	390	25	250	715	15	630
9	/1009	3.2	470	25	155	715	<10	210
10	/1010	7.8	540	25	95	650	10	405
11	/1011	9.2	270	20	100	470	<10	175
12	/1012	7.6	255	20	140	435	125	320
13	/1013	9.4	380	25	165	505	10	150
14	/1014	12.1	310	15	160	660	20	180
15	/1015	12.4	560	25	260	1108	75	225
16	/1016	9.5	510	25	130	650	50	150
17	/1017	9.1	850	25	205	415	45	430
18	/1018	8.8	850	20	125	480	30	295
19	/1019	13.0	810	30	375	715	25	345
20	/1020.	11.4	630	25	235.	610	65.	280
21	Continue - - - /2.							

Copies to: 1. W.H.A. 2. A.B. Chief Chemist:

3. G.W. 4. Date: 8/5/75.

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Sample: Expl^n - Rock - 07/0705/- Report No.: AK-30 ~~11.9~~

Submitted by: G.W. Requisition No.: 1815 Comm. K Cont. K Conc.

ANALYSIS

	SAMPLE	F %	Zn ppm	Pb ppm	Cu ppm	Bi ppm	Mo ppm	W ppm
21	0705/1021	13.5	740	25	175	505	25	295
22	/1022	7.2	690	25	160	415	20	330
23	/1023	8.0	1020	25	260	600	20	195
24	/1024	7.0	730	25	230	435	45	560
25	/1025	13.4	290	25	35	715	25	810
26	/1026	13.5	340	80	110	470	145	675
27	/1027	16.2	360	25	160	560	45	270
28	/1028	13.3	460	25	130	390	30	195
29	/1029	17.0	160	25	110	480	180	545
30	/1030	13.2	160	25	40	350	240	800
31	/1031	11.2	500	30	145	1070	85	500
32	/1032	11.0	800	25	55	470	45	1320
33	/1033	14.8	550	25	105	205	220	560
34	/1034	11.2	2320	25	185	805	100.	780
35			—	—	—	—	—	—
	Arithmetic average	8.91%						

Copies to: 1. W.H.A 2. A.B 3. G.W. 4.

(2)

Chief Chemist:

Date: 8/5/75.

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221140

Sample: Expl^n - Rock - 07/0705/- Report No.: AK-31. (1-7)

Submitted by: G.W. Requisition No.: 1815 Comm. ..... Cont. ✓ Conc. ✓

ANALYSIS

	SAMPLE	F %	Zn ppm	Pb. ppm	Cu ppm	Bi ppm	Mo. ppm	W ppm
1	0705/1035	7.4	820	25	160	520	120	560
2	/1036	5.6	1250	30	190	350	40	295
3	/1037	4.2	1200	25	145	430	65	195
4	/1038	7.0	660	25	125	260	55	630
5	/1039	10.0	680	30	100	215	115	1440
6	/1040	9.0	1140	30	185	310	300	408
7	/1041.	7.0	1230	35	230	1035.	580	1020
35.			–	–	–	–	–	–
	Arithmetic av. incl. 35-41	8.6%						

Copies to: 1. W.H.A. 2. A.B. Chief Chemist: [signature]

3. G.W. 4. Date: 8/5/75

APPENDIX 11.

SHEPHERD AND MURPHY AREA

Samples collected by A.H. Bartlett, 1976
Analyses by Comalco Laboratory

	← FLUORITE - MAGNETITE - SKARN →						SANDSTONE
Analysis	Moina 1	Moina 2	Moina 3	Moina 4	Moina 5	Moina 6	Moina 7
CaF_2	25.2 %	20.5 %	21.9 %	27.3 %	18.6 %	16.3 %	0.72 %
CaO	5.1	2.4	3.33	0.80	1.21	0.92	N.D.
$CaCO_3$	1.1	0.9	0.95	1.00	0.80	0.69	0.22
$MgCO_3$	0.2	0.2	0.15	0.10	0.19	0.17	0.09
$FeCO_3$ (?)	0.7	0.8	0.79	0.56	1.51	1.35	1.72
Total Fe as Fe_2O_3	29.9	34.1	22.1	24.3	33.3	38.9	3.9
SiO_2	20.0	22.9	29.0	24.8	24.3	23.8	83.3
TiO_2	0.1	0.1	0.25	0.16	0.20	0.16	<0.10
Al_2O_3	10.2	9.5	7.7	7.4	8.0	6.5	1.6
MgO			1.58	1.39	2.54	1.43	0.52
L.O.I.			3.9	4.2	3.1	4.7	
Sn							
Zn	700 ppm	1060 ppm	215 ppm	140 ppm	5350 ppm	295 ppm	30 ppm
Pb	15	15	80	75	70	60	35
Cu	115	70	25	15	265	140	20
Bi	605	325	420	310	415	440	30
Mo	50	110	130	180	145	145	145
Sb	55	45	40	40	35	35	10
W	1270	1190	930	825	1200	985	90
Au	<0.1	<0.1	0.02	0.02	0.04	0.04	<0.02
Ag			5.7	4.4	6.0	4.8	0.9
TOTAL			~91.0	~91.6	~93.0	~93.7	

Location of samples:

Moina 1, 2, 5 and 6 - from outcrop adjacent to old treatment plant
" 3 and 4 - main shaft mullock heap
" 7 - lower adit mullock heap

APPENDIX 12

SHEPHERD AND MURPHY AREA

COSTEAN SAMPLING AND ANALYSES.

A. WESTERN COSTEAN.

(a) Sampling method

After cleaning wall of costean a strip sample was collected down the wall. A sample was taken over each one metre from the base of the basalt to the floor of the costean. This strip sampling was performed every 2 metres along the costean. Where the costean had a hard floor one strip sample was taken over one metre across the costean floor immediately opposite the vertical strip but at the opposite side of the costean, where the costean had a soft floor some screw augering was carried as deep as was possible, sampling one metre intervals.

Samples were labelled with grid reference and depth from surface - e.g. RD/Tas/SM/9028E 26S/4-5 5-6 etc. or RD/Tas/SM/9029.5E 26S/0-1, 1-2 etc.

Costean sample sites were marked with pegs in the floor labelled with aluminium permatags as follows - 9028E 26S/4M, 9028E 26S/5M (last number indicates depth in metres).

(b) List of samples collected

All samples are of deep lead alluvium or highly to completed weathered wrigglite.

All samples inadvertently sieved to -80 mesh.

21S/945.5E/	2	- 3 metres
	3	- 3.6
21S/947 E/	3	- 3.8
21S/948.5E/	1.3	- 2
	2	- 3
	3	- 4
21S/950 E/	1.5	- 2.5
	2.5	- 3.5
21S/970 E/	2	- 3
	3	- 4
	4	- 5
	5	- 5.3

21S/972	E/	2 - 3	metres
		3 - 4	
		4 - 4.5	
21S/974	E/	2.3 - 3.3	
		3.3 - 4.3	
21S/976	E/	3 - 4	
		4 - 5	
		5 - 6	
		6 - 6.5	
21S/978	E/	1.5 - 2.5	
		2.5 - 3.5	
		3.5 - 4.5	
		4.5 - 5.5	
		5.5 - 6.2	
21S/980	E/	1 - 2	
		2 - 3	(missing)
		3 - 4	
		4 - 4.5	
21S/982	E/	0.7 - 1.7	
		1.7 - 2.7	
		2.7 - 3.7	
		3.7 - 4.7	
21S/954.5	E/	1.4 - 2.4	
		2.4 - 3.4	
		3.4 - 4.4	
		4.4 - 5.4	
		5.4 - 5.6	
21S/981	E/	2 - 3	

B. EASTERN COSTEAN.

(a) Sampling method

The method in general was as for the western costean, but this costean is in fresher wrigglite and calc-silicate rock, so some samples are chips gathered from a ripped section in the floor of the costean. These samples are distinguished in the attached list by the lack of depth numbers.

(b) List of samples collected

27S/1036	E/	1	- 1.5	- 2.5	m
1038		1	- 2	- 3	
1040		1	- 2	- 3	- 3.5
1042		1	- 2	- 3	- 3.5
1044		1	- 2	- 3	- 3.5
1046		1	- 2	- 3	- 3.5
1048		1	- 2	- 3	- 3.5
1050		1.4	- 2.4	- 3.4	- 4
1052		1	- 2	- 3	- 3.4
1054		1	- 2	- 2.7	
1056		1	- 2	- 2.2	

27S/1058 E
1060
1062 1 - 2 metres
1064
1068
1072
1075
1080
1085
1089
1091
1093
1095
1097

25S/1038 E
1040
1042
1044
1046
1048
1050
1052
1054
1056
1058
1060
1062

C. ANALYSES, ppm, (Western Costean only)

SAMPLE	Sn	W	Be	DESCRIPTION
21S/970E/5 -5.3	1850	1450	30	
21S/972E/4 -4.5	620	1100	20	All highly to completely weathered wrigg-lite, -80 mesh
21S/974E/3.3-4.3	980	1250	30	
21S/976E/6 -6.5	3950	1350	30	
21S/978E/5 -5.2	9700	1550	30	All lower-most sample collected in costean
21S/980E/4 -4.5	7300	1100	20	
21S/982E/2.7-3.7	1100	1150	30	Highly to complete-ly weathered wrigg-lite,-80 mesh.Lower most sample plus sample above.
21S/982E/3.7-4.7	1000	1300	30	
ANALYTICAL METHOD	AMDEL XRF		SEMI-QUANT. SPECTRO-GRAPHIC	

Sample adjacent to same costean, of gossan of wrigglite.

All analyses by Amdel, semi-quantatitative spectrographic.

	Sn	W	Be	Bi	Sb	As	Ag	Cu	Pb	Zn	Cd	Ge
PA/T/MOINA 6	1350	640	5	700	50	100	0.1	300	20	700	3	3

APPENDIX 13

MINES DEPARTMENT SAMPLES

SHEPHERD AND MURPHY AREA

ANALYSES BY	ELEMENT	WRIGGLITE AT SHAFT 736350	WRIGGLITE AT CRUSHER 736351
MINES DEPT.	Fe%	30.1	17.8
	WO_3%	0.07	0.52
	Sn%	0.21	0.20
	Cu%	0.01	0.01
	Bi%	0.05	0.03
	Mo%	Nil	0.14
	Au	Nil	Nil
COMALCO	F%	9.2	12.2
	W ppm	270	4480
	Bi ppm	425	260
	Mo ppm	15	1170
	Ag ppm	3	3
	Cu ppm	40	45
	Pb ppm	25	30
	Zn ppm	250	225

APPENDIX 14.

ANALYSES OF ROCK SAMPLES, LEA R AND IRIS R AREAS

ANALYSES BY AMDEL, NOV., DEC. 1976.

Au analyses by AAS; all others semi-quantitative spectrographic (Code A1)

X = below detection limit. Detection limit in brackets.
"Beryllium has been read allowing for suppression from CaF_2"

"It is likely that all refractory elements of Code A 1 will be suppressed to some extent by the CaF_2, a failure intrinsic to the method and there is no easy way to counter it. However, the relative levels are still valid".

SAMPLE NO.	Sn (1)	W (50)	Bi (1)	Be (1)	Au (0.05)	Ce (300) ppm	La (100)	Nb (20)	Y (10)	Sc (3)	Ta (100)	NOTES
600W-950S A	150	x	300	30	x	x	x	x	x	3	x	A,B,C are slightly
" B	10	x	30	30	x	x	x	x	30	3	x	different rocks from
" C	70	x	10	10	x	x	x	x	x	3	x	same locality
600W-600S	2	x	x	20	x	x	200	x	100	x	x	
840W-450S A	100	x	100	30	x	x	x	x	10	3	x	Note as above
" B	5	x	70	5	x	x	x	x	10	5	x	
900W-285S	100	100	500	100	0.10	x	x	x	30	x	x	
950W-1025S A	100	x	30	10	x	x	x	x	10	10	x	
" B	50	x	30	30	x	x	x	x	10	x	x	
1250W-550S	30	x	10	30	x	x	x	x	50	10	x	
1150W-240S	x	x	x	3	x	x	x	x	10	3	x	
000-1300S	80	x	70	1	x	x	x	x	10	3	x	
000-1420S	100	x	20	10	x	x	x	x	50	10	x	
1250W-1000S	30	x	30	30	x	x	x	x	x	3	x	
100E-1440S A	30	x	70	3	x	x	x	x	10	3	x	
" B	30	x	20	20	0.10	x	x	x	30	3	x	
100W-1400S A	5	x	800	3	x	x	x	x	x	x	x	
" B	30	50	10	5	0.05	x	x	x	20	10	x	
860W-940S	5	x	x	10	x	x	x	x	10	x	x	
1450W-1120S	10	x	10	3	x	x	x	x	x	5	x	
1200W-560S	3	x	x	1	x	x	x	x	10	x	x	
2730W-1100S	100	x	300	30	x	x	x	x	20	3	x	
2900W-1395S	1	x	3	3	x	x	100	x	30	10	x	
2600W-930S A	1	x	x	10	x	x	100	x	50	30	x	
" B	10	x	20	30	x	x	100	x	70	30	x	
2800W-1040S	1000	100	1000	150	x	x	x	x	20	3	x	near Fletchers adit
3115W-1430S A	3	x	30	5	x	x	x	x	x	10	x	
" B	30	x	30	70	x	x	x	x	10	x	x	
3270W-1670S A	30	200	100	200	0.25	x	x	x	30	5	x	mag. skarn S of Stormont
" B	30	x	50	10	x	x	x	x	30	3	x	
3175W-1600S	x	x	x	10	x	x	x	x	50	10	x	
3100W-1440S	50	x	5000	30	0.50	x	x	x	10	5	x	near Stormont Mine
3000W-990S A	2000	100	500	100	0.10	x	x	x	x	3	x	Area west of
" B	1000	50	200	100	x	x	x	x	10	10	x	Fletcher's adit
3185W-1500S	10	x	x	3	x	x	x	x	10	5	x	

Appendix 14, cont.

SAMPLE NO.	Sn (1)	W (50)	Bi (1)	Be (1)	Au (0.05)	Ce (300)	La (100)	Nb (20)	Y (10)	Sc (3)	Ta (100)	NOTES
							ppm					
3150W-1430S	10	x	30	30	x	x	x	x	50	5	x	
3250W-750S	30	x	5	10	x	x	x	x	30	15	x	
825W-850S	20	x	200	50	x	x	x	x	x	5	x	
1250W-800S A	30	x	5	3	x	x	x	x	x	x	x	
" B	50	x	10	30	x	x	x	x	x	x	x	
1350W-875S	250	x	300	200	0.25	x	x	x	x	5	x	Tea-Tree Ck area
900W-700S A	3	x	5	10	x	x	x	x	50	15	x	
" B	3	x	10	5	x	x	x	x	30	15	x	
400W-900S	20	x	100	50	x	x	x	x	30	10	x	
1400W-860S	300	x	50	80	x	x	x	x	50	15	x	
860W-340S A	50	x	10	20	x	x	x	x	50	15	x	
" B	200	x	x	10	x	x	100	x	50	20	x	
1150W-260S	10	x	x	150	x	x	x	x	x	3	x	
1300W-100S	10	x	20	70	x	x	x	x	50	5	x	
200W-1125S	10	x	10	30	x	x	x	x	50	5	x	
co - 1300S	100	x	10	1	x	x	x	x	30	10	x	
1350W-1000S	70	x	5	10	x	x	x	x	50	10	x	
co - 1360S	100	x	10	70	x	x	100	x	50	15	x	
1200W-1000S A	80	x	30	10	x	x	x	x	10	10	x	
" B	50	x	5	5	x	x	x	x	x	5	x	
800W-800S	10	x	100	40	x	x	100	x	70	10	x	
1250W-990S	5	x	200	30	0.15	x	x	x	x	5	x	near DOM 3
co - 1280S	100	x	10	30	x	x	x	x	30	15	x	
400W-960S	20	x	300	40	x	x	x	x	30	5	x	
1150W-675S	50	x	10	10	x	x	100	x	80	15	x	
900W-300S	50	x	30	100	x	x	100	x	30	5	x	
Lea R - 2450W	30	x	5000	5	0.10	x	x	x	x	x	x	galena rich vein
3100W-1400S	10	x	80	15	0.05	x	x	x	50	10	x	
850W-375S	70	x	100	80	x	x	x	x	80	20	x	
3150W-1550S	3	x	5	30	x	x	x	x	100	10	x	
2900W-1450S	x	x	10	x	x	x	x	x	30	3	x	
3250W-1650S	50	300	1000	200	0.15	x	x	x	150	15	x	mag skarn South of Stormont Mine
2950W-1025S	500	2000	100	150	0.10	x	200	x	70	10	x	Area W. of Fletcher's Adit
3150W-1500S	10	x	20	50	0.05	x	150	x	50	5	x	
3000W-1000S	200	x	100	80	x	x	x	x	10	10	x	
2700W-950S	300	x	10	40	x	x	x	x	50	10	x	
3100W-1550S	30	x	800	40	1.20	x	x	x	50	10	x	near Stormont Mine
												Costeans, road cuts etc:
FA 0 NW	150	x	500	30	0.05	x	x	x	10	10	x	Fletchers Adit
FA 8 NW	100	x	100	70	0.05	x	x	x	30	5	x	
T.T.CK 25NW	500	x	700	50	0.35	x	x	x	30	10	x	Tea Tree Ck
T.T.CK 6 SE	700	x	700	40	0.65	x	x	x	30	10	x	
T.T.CK 0	500	x	1000	30	0.50	x	x	x	x	5	x	
IRISH 980W 850S-865S	50	x	100	10	x	x	x	x	50	10	x	
IRISH 980W 830S-340S	30	x	500	50	x	x	100	x	100	15	x	
IRISH 980W 775S-795S	50	x	100	30	x	x	100	x	70	10	x	
STORMONT 312W-1500S	30	x	1000	40	1.95	x	x	x	10	5	x	
STORMONT 314W-1500S	30	x	5000	70	2.10	x	x	x	30	10	x	
STORMONT 316W-1500S	1	x	10	10	5.1	x	x	x	10	5	x	

1 for details see plan no. TAS-78-137

APPENDIX 15

POST OFFICE ANOMALY

HAND AUGER SAMPLING - ANALYTICAL RESULTS.

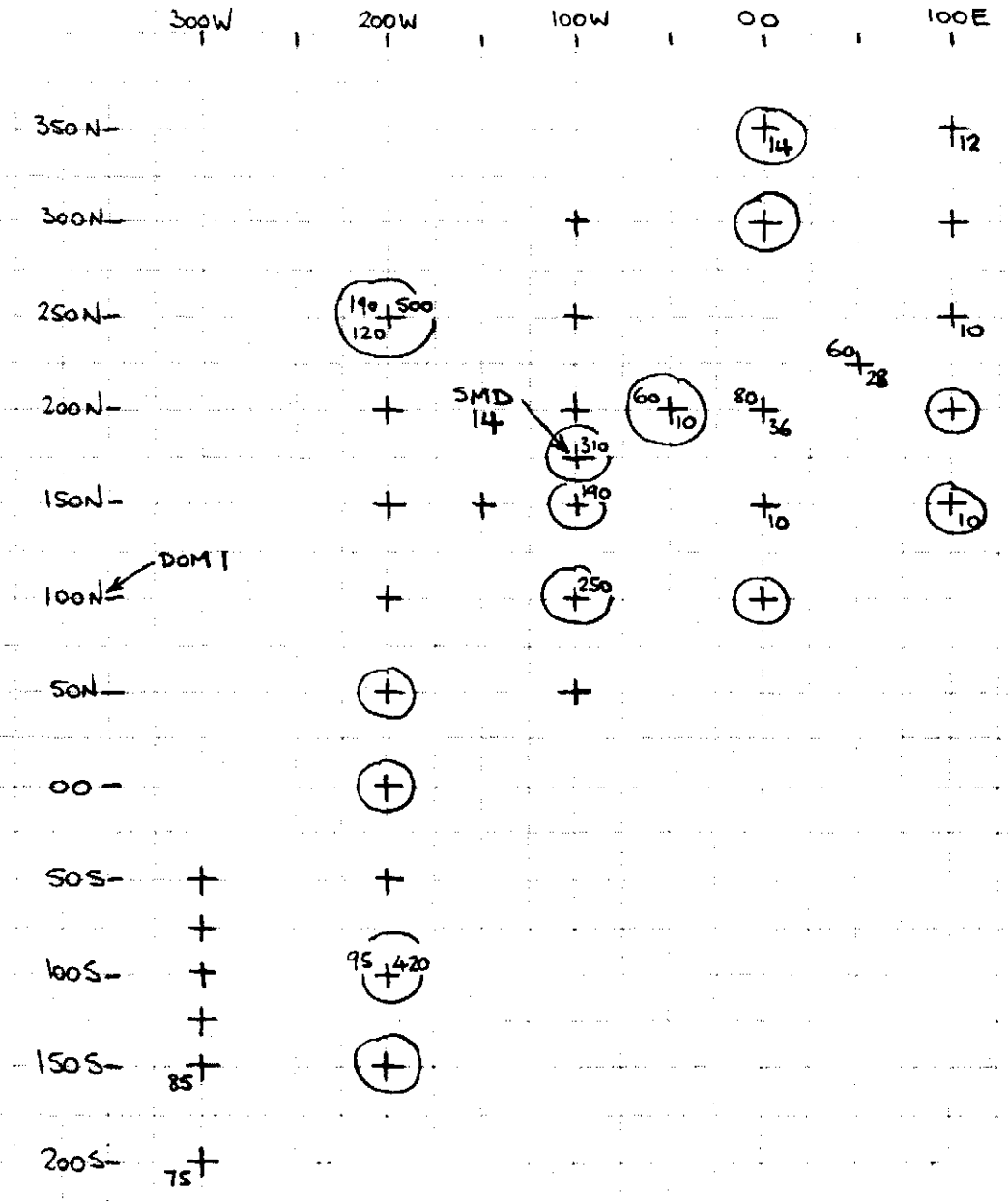
Samples sieved to -80 mesh.

Analyses by Amdel XRF Code B1 for Sn, W and AAS Code C1 for Cu, Pb, Zn and Bi.

Sample Co-ords	Sn ppm	W ppm	Cu ppm	Pb ppm	Zn ppm	Bi ppm	Depth Taken	Notes by collector
100E/150N	10	-10	40	42	110	-10	8 m	Hole still going
100E/200N	4	-10	32	30	100	-10	8	Hole still going
100E/250N	10	-10	55	25	120	-10	1.8 m	Hit rock
100E/300N	6	-10	38	18	140	-10	0.5 m	Hit rock
100E/350N	12	-10	55	30	120	-10	0.5 m	Hit rock
50E/225N	28	15	28	60	42	-10	2 m	Hit rock
000 /100N	4	-10	32	38	90	-10	7 m	Stones falling in
000 /150N	10	10	38	35	85	-10	0.6 m	Stones falling in
000 /200N	36	20	25	80	48	-10	2.3 m	Hit rock
000 /300N	6	-10	48	32	100	-10	?	No notes
000 /350N	14	-10	50	28	120	-10	0.6 m	Hit rock
100W/ 50N	6	-10	55	25	150	22	1.2 m	Hit rock
100W/100N	- 4	-10	35	28	250	-10	8 m	Hole still going
100W/150N	- 4	-10	32	28	190	-10	8 m	Hole still going
100W/175N	6	-10	30	35	310	-10	7.1 m	Hole still going but stones falling in.
100W/200N	6	-10	50	38	95	-10	1.6 m	Hit rock
100W/250N	4	-10	65	28	110	-10	2 m	Hit rock
100W/300N	4	-10	60	18	110	-10	0.8 m	Hit rock
200W/000	- 4	-10	40	30	140	-10	4.5 m	Hit rock
200W/ 50S	8	10	50	42	170	-10	1.5 m	Hit rock
200W/100S	- 4	-10	25	95	420	-10	5 m	Hole still going
200W/150S	4	-10	25	22	110	-10	5 m	Hole still going
200W/200S	8	-10	50	35	130	-10	5 m	Hole still going
200W/ 50N	- 4	-10	42	22	150	-10	3 m	Hole still going but stones falling in.
200W/100N	4	-10	45	25	100	-10	0.5 m	Hit rock
200W/150N	- 4	-10	50	28	130	-10	0.8 m	Hit rock
200W/200N	- 4	-10	60	25	140	-10	1 m	Hit rock
200W/250N	- 4	-10	120	190	500	-10	8 m	Hole still going
300W/ 50S	4	-10	32	28	60	-10	3 m	No notes
300W/ 75S	- 4	10	22	42	28	-10	3 m	No notes
300W/100S	- 4	-10	65	22	110	-10	3 m	No notes
300W/125S	4	-10	65	18	110	10	3 m	No notes
300W/150S	6	-10	85	12	110	-10	2.8 m	No notes
300W/200S	4	-10	75	12	110	-10	0.8 m	No notes
150W/150N	4	-10	25	25	130	-10	5.8 m	Hit rock
50W/200N	10	10	32	60	70	-10	6.1 m	Hole still going but falling in.

NOTES: (a) -4 means less than

(b) Rocks hit in hole are probably cobbles boulders and not limestone bedrock.



KEY :

Pb	Zn
Cu	Sn

Sample point showing Pb ≥ 60 ppm
Zn ≥ 190
Cu ≥ 75
Sn ≥ 10

Sample collected at satisfactory depth.

POST OFFICE ANOMALY - AUGER SOIL GEOCHEMICAL SAMPLING

FIG.9.

APPENDIX 16 - GEOPHYSICAL ROCK PROPERTY TESTS.

COMALCO LIMITED

COMALCO

Memorandum

Ref

Date 11th November 1975

To A. H. White, J. C. Goudie, M. E. Goudie, I. D. Picken.

From G. Weste

At Cairns

At Melbourne

Copies A. H. Bartlett

Subject Induced Polarization - Wrigglites

Please note the attached "Physical Rock Property Tests" carried out by Geoex.

Note Density of skarn	3.14 to 3.84
Frequency effect of skarn	high (at high frequency)
Resistivity of skarn	high
Metal factor of skarn	low
Magnetic susceptibility - basalt	low

As indicated by the ground magnetic survey over the S. and M. area, the basalt is causing no magnetic distortion (but only based on 1'6" section of core). However, basalts in Tasmania are generally magnetic.

Moina type wrigglite will give a reasonably effective frequency effect anomaly at higher frequencies (magnetite) but shows high resistivity, but only based on a 2'6" section of drill core. High frequency I.P. could therefore be used to locate Moina type wrigglite in Tasmania, particularly in areas with an overburden of magnetic basalt. In spite of weathering characteristics in North Queensland it is quite possible that frequency effect anomalies could be caused by wrigglite.

The ability to detect sulphide mineralisation (e.g. pyrrhotite) by I.P. beneath or adjacent to Moina type wrigglite is limited at high frequency and would be determined mainly at lower frequencies and by resistivity.

An I.P. survey will be carried out over the S. and M. area in late November this year by Scintrex (see attached notes). This survey will determine:

1. Whether I.P. is a suitable technique for locating wrigglite skarns in Tasmania.
2. Whether the wrigglite at Moina shows any variation in I.P. response.
3. If there is any high metal factor rocks associated with the wrigglite (possible pyrrhotite rocks)

More detailed magnetic susceptibility, and I.P. tests at various frequencies, will be carried out on the core at Moina using a CTU-2 core testing unit to be hired from Scintrex (notes attached).

G. Weste.

Enclosures
JMS

GEOEX PTY. LTD.

221152

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Submitted by G. WESTE
Company COMALCO
Area
Date Submitted 8.9.75
Date Tested 9-11.9.75
Tested by T. Wilson

PHYSICAL PROPERTY ROCK TEST

Description BASALT		Wet Density (gm/cc)	Dimensions: Cross Section X x Y or dam. (sq cm)	Dimensions: Length between measuring electrodes (cm)	Magnetic Susceptibility c.g.s. units	Resistance ohms. (KΩ)	Resistivity Freq 0.5 / 0.31 (Ω)metres	P.F.E. (%) Freq 0.5 / 0.31	Metal Factor
126'6" - 128'	ML2	2.88	4.7	7.2	< 200	1772.0	11,567.0	3.5	0.3
"	different pieces	2.89	4.7	6.1	< 200	1683.0	12,967.0	3.3	0.25
"	of basalt in	2.84	4.5	5.7	< 200	618.0	4,879.0	1.8	0.37
"	core from this	2.76	4.1	5.3	< 200	180.0	1,392.0	1.3	0.93
"	interval	2.80	4.1	5.3	< 200	261.6	2,024.0	1.9	0.94
"		2.90	4.7	3.4	< 200	939.6	12,988.0	2.8	0.22
"		2.69	4.0	3.0	< 200	-	-	-	-
10' M2 Large Core (Too Large ML2 basalt from near top of hole		2.84	6.8	7.8	-	1000.0	8718	0.8	0.09

GEOEX PTY. LTD.

221153

152

Submitted by G. WESTE
Company COMALCO
Area
Date Submitted 8.9.75
Date Tested 9.11.9.75
Tested by T. Wilson

PHYSICAL PROPERTY ROCK TEST

Description WRIGGLITE	Wet Density (gm/cc)	Dimensions: Cross Section X x Y or diam. (sq cm)	Dimensions: Length between measuring electrodes (cm)	Magnetic Susceptibility c.g.s. units	Resistance ohms. (KΩ)	Resistivity Freq 5.0 0.31 (Ω metres)	P.F.E. (%) Freq 5.0 0.31	Metal Factor
1 Sample too small 197'6" - 200' M2	3.79	3.8	2.9	-	35.6	466.0	26.5	57.0
2 "	3.58	4.0	6.8	47,369.9	509.2	2995.0	16.8	5.6
3 Sample too large	3.44	5.7	3.3	-	58.4	1008.0	24.6	24.0
4 "	3.84	3.92	6.4	87,598.7	176.0	1078.0	35.9	33.0
5 "	3.14	4.5	4.3	5,951.0	2000.0	20930.0	2.0	0.09
6 "	3.77	4.6	6.7	86,322.0	270.0	1853.0	22.1	12.0
7 "	3.57	4.5	7.7	55,485.0	1037.2	6061.0	9.5	1.6

NOTE : 1–7 are different pieces of wrigglite core from ML2 over the interval 197'6–200'

221154

APPENDIX 17.

A REPORT ON
GRADIENT ELECTRICAL INDUCED POLARIZATION SURVEYS
OVER THE SHEPHERD AND MURPHY GRID
E.L. 7/74, MOINA, NORTHERN TASMANIA
ON BEHALF OF
COMALCO LIMITED

221155

PRIVATE AND CONFIDENTIAL

A REPORT ON
GRADIENT ELECTRICAL INDUCED POLARIZATION SURVEYS
OVER THE SHEPHERD AND MURPHY GRID
E.L. 7/74, MOINA, NORTHERN TASMANIA
ON BEHALF OF
COMALCO LIMITED

BY

A.W. HOWLAND-ROSE
MSc,DIC,AMAusIMM,FGS.
GEOPHYSICIST

Sydney, N.S.W.

DECEMBER, 1975

TAS - 031

221156

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Summary	
Introduction	Page 1
Method and Equipment	Page 2
Discussion of Results	Page 6
Conclusions and Recommendations	Page 16

221157

SCINTREX PTY. LTD.
Formerly
SEIGEL ASSOCIATES AUSTRALASIA PTY. LTD.
GEOPHYSICAL CONSULTANTS AND CONTRACTORS

SUMMARY

The skarn zone at Moina was, as expected, found to be most resistive and chargeable. Two other such zones were located beneath basalt cover. However, the location of substantial induced polarization responses from within the Moina Sandstone which may be of formational origin, make a positive identification impossible.

The results require careful geological evaluation, but are considered encouraging.

1031 WELLINGTON STREET, WEST PERTH, W.A. 6005
TELEPHONE 21 6934 TELEGRAMS "SCINTREX" PERTH

6 TRAMORE PLACE, KILLARNEY HEIGHTS, N.S.W. 2087
TELEPHONE 451 5367. TELEGRAMS "SCINTREX" SYDNEY

221158

A REPORT ON
GRADIENT ELECTRICAL INDUCED POLARIZATION SURVEYS
OVER THE SHEPHERD AND MURPHY GRID
E.L. 7/74, MOINA, NORTHERN TASMANIA
ON BEHALF OF
COMALCO LIMITED

INTRODUCTION

On seven and a half production days between 1st and 8th December, 1975, a gradient array electrical induced polarization survey was carried out over a portion of the Shepherd and Murphy Grid, Moina (E.L. 7/74) on behalf of Comalco Limited. Some detailed work was carried out in addition to the gradient survey.

The survey was performed at the request of Mr. G. Weste, Senior Geologist in conjunction with Mr. A.H. Bartlett, Divisional Manager-Exploration Department, of Comalco Limited. The surveys were executed by Scintrex Senior Operator, Mr. B. Ekstrom assisted by fieldhands provided by Comalco Limited. These surveys were under the immediate geological direction of Mr. G. Weste, while the author undertook technical supervision. On 26th November, 1975 the author visited the survey site prior to commencement of the survey.

The objective of the survey was to define and delineate the geophysical characteristics of the skarn zone at Moina

without, and under basalt cover, and further, to locate and define additional skarn zones. The target mineralisation was expected to be both *chargeable* and *resistive* relative to the enclosing host rocks, while the nature of the basalt was expected to vary considerably depending on the degree of oxidation and mineral content.

METHOD AND EQUIPMENT

The major components of the equipment used consisted of two Scintrex IPR-7 induced polarization receivers and a Scintrex 2½ KW time domain transmitter.

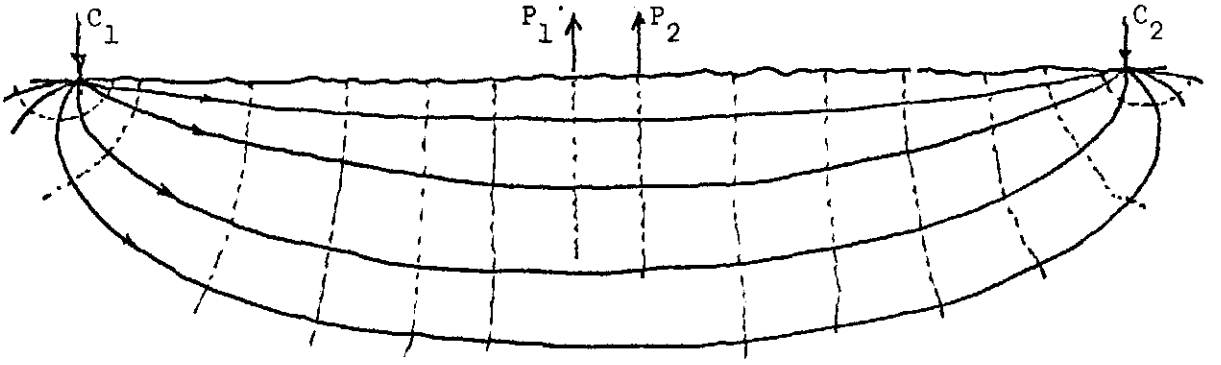
A very simple explanation of the parameters measured in the gradient array survey follows. This description is designed specifically for the geologist, in order to give a *visual picture* of the parameters measured.

In the case of the gradient array, the potential dipole records the electrical properties of the material defined by the two equipotential *surfaces* tapped by the potential electrodes. This is diagramatically illustrated in Figure 1(A). This diagram represents a *section*, however, it should be realised that the apparent resistivity measurements also record information *sideways*. For a three dimensional picture of the volume sampled, rotate the section *into* and *out of* the plane of the paper by 90°. Within the centre section of

ELECTRICAL PARAMETERS MEASURED

(A) RESISTIVITY MEASUREMENT

(taken during current 'on' time)



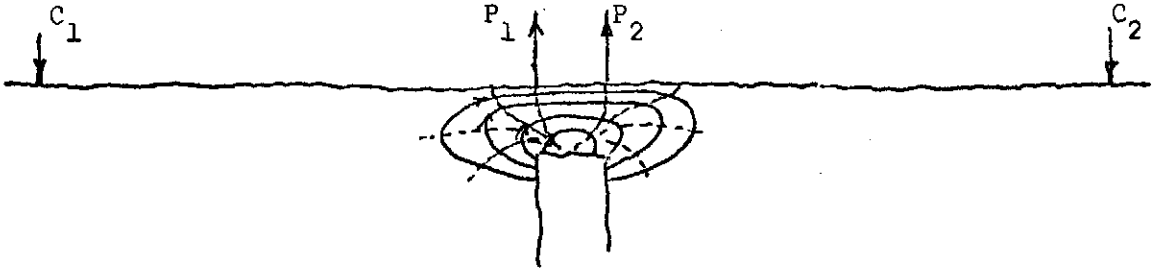
MEASUREMENT REPRESENTS:

ease with which primary current moves through ground

primary current flow
primary equipotential surface

(B) CHARGEABILITY (IP) MEASUREMENT

(taken during current 'off' time)



MEASUREMENT REPRESENTS:

discharge of stored energy

secondary current flow
secondary equipotential surface

FIGURE 1

the array the data represents the characteristics of the rock units *immediately below* and *immediately at right angles* to the survey line. The degree of *resolution* depends on the potential *dipole width*, the smaller the dipole the greater the resolution. Diagram 2 (the resistivity data) demonstrates the much improved resolution for a 10 metre dipole over that obtained from a 25 metre dipole over a sulphide occurrence whose charactersitics will be similar to the target mineralisation It is important to remember that only the *gross* properties are "seen". The potential dipole cannot resolve units whose effective width is less than half the size of that dipole.

The criteria affecting the chargeability reading are somewhat different. In diagram 1B the decay of a chargeable section is shown. The passage of current during the *current-on* phase, during which the resistivity measurement was taken, carried some energy to be stored in the rocks (and sulphides, etc.) through which it passed. On cessation of this imposed current flow, the energy so stored will discharge (IP). It will set up its own equipotential field as shown in Figure 1(B) which will be detected by the same two potentials which measured the resistivity. It should be noted that any chargeable source will have a width *greater than* the source due to the curvilinear nature of the discharge of the stored energy. It should be further noted that the volume defined by the *secondary*

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5 cm

IMPORTANCE OF RESOLUTION

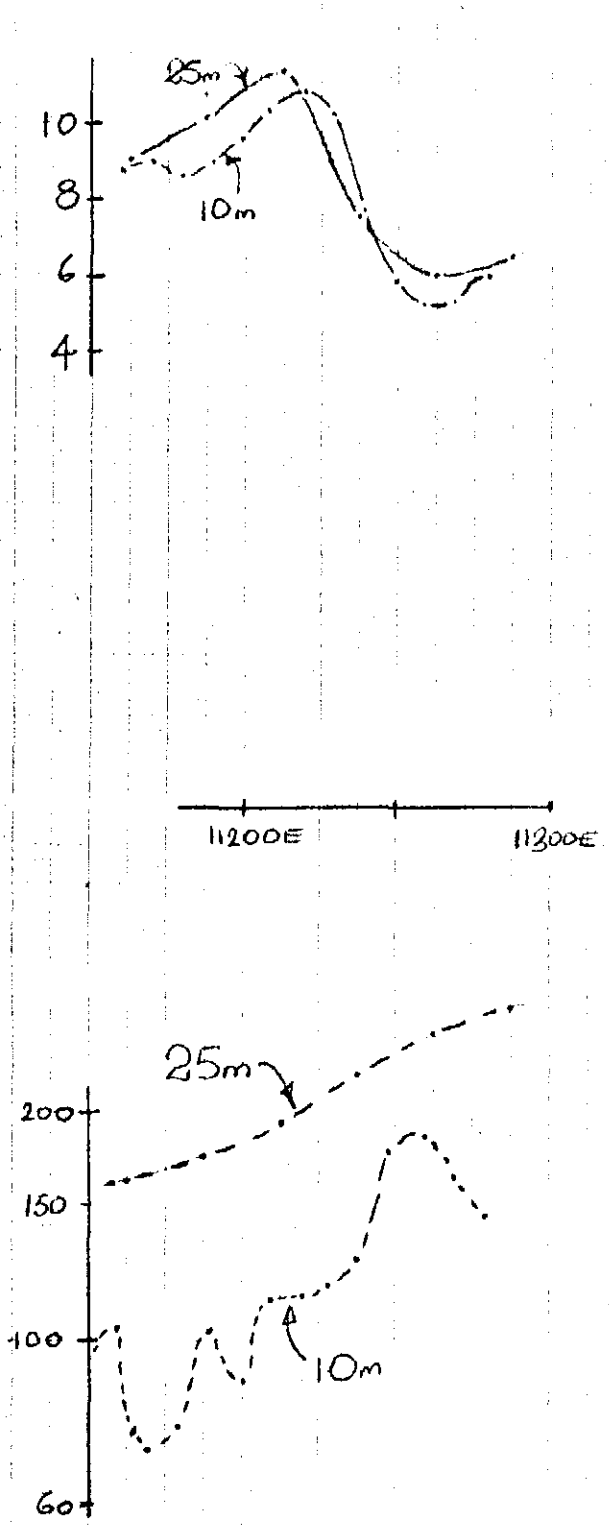

FIGURE 2

potential field caused by this discharge is *not necessarily identical* to that defined by the primary equipotential field. Figure 2 shows the difference in resolution *for chargeability* (top profile) over a known sulphide occurrence. The *form* is similar, but the positional information is far superior due to the more frequent reading interval.

In the present survey the 25 metre potential was considered quite adequate for defining target areas. Only three array surveys were preferred to assess the depths of some narrow targets.

This array has operational attributes which result in rapid coverage and excellent positional information. However, depth information is not well defined, it being possible only to assess "maximum depths".

With regard to the interpretation of gradient array data the following comments may prove to be of assistance.

In the gradient array the source of the reading lies between the two equipotential surfaces tapped by the two potential pots employed. For the most part then, when working in the centre section of a gradient array, the source will be "immediately below" the potential dipole used. The reliability therefore of *positional information* with gradient array is excellent,

however, the depth at which the response occurs is difficult to assess with accuracy. The maximum depth can be estimated from a consideration of the profile shape, but the accuracy of this approach will depend on a minimal current dipole length, and of course sharp boundaries to the body. The *resolution* therefore is not better than half to a quarter of the dipole. Therefore maximum depths of the order of 10 metres may in fact either outcrop or suboutcrop when a 25 metre potential dipole is used. Some moving source array would be required to obtain an *accurate depth estimate*.

Similarly the width of bodies is not easy to determine for zones having a width less than half the dipole spacing used. Thus, estimated maximum widths are educated guesses at best for narrow zones. However, wider bodies can be resolved more accurately.

The *attitude* of a chargeable zone can only really be gauged with any precision in the centre of the gradient array, and of course where the body has a strongly contrasting chargeability and apparent resistivity to that of the enclosing rock units.

All field measurements were taken between slope distances along lines. This will, in steep areas, produce errors in the calculated apparent resistivity data. However, these errors will be arithmetic, and as significant changes in

resistivity are logarithmic, this source of error is not significant. In assessing the position of the source in areas of extreme terrain, it does not lie vertically below the plotted position, but *normal to the "local slope"*. All positions in the text refer to source positions normal to the local slope.

Each current dipole block should be considered separately. As would be expected, the continuity along strike is generally good, especially in the chargeability data. However, *"end on"* current dipole blocks cannot be expected to give identical data due to the different base levels of the current dipoles and, in zones close to the current poles, the data will not sample identical volumes on the overlap between current dipoles. This phenomenon will result in more extreme divergence of data as the current dipole is approached. However, these factors are entirely predictable.

DISCUSSION OF RESULTS

The gradient array electrical induced polarization data has been contoured and drafted by Comalco Limited, and therefore is not appended to this report.* The scales utilised in that presentation were 1:2500 while the chargeability was presented on the scale of 1 centimetre = 2 milliseconds and resistivity was shown at the scale of 5 centimetres = 1 logarithmic cycle

* see DRAWINGS TAS-78 – 145
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A line by line review of the data follows:

LINE 050E:- The chargeability background over this line is about 10 milliseconds. Superimposed on this background, 6 to 8 millisecond chargeabilities were recorded from between 425S and 650S. The *maximum* depth to source is estimated to be of the order of 50 to 60 metres.

A second chargeability high of up to 10 millisecond above the local background was observed between 050S and 100N. The maximum depth to source is interpreted as being of the order of 50 metres.

In both the above cases there is no material change in apparent resistivity, and thus the source is considered to be *disseminated*.

LINE 150E:- A broadly similar profile form was observed for the induced polarization data, but of lower amplitude.

The depth to the 4 millisecond response between 450S and 575S cannot be ascertained from the profile form, while the 7 millisecond response centred at 060N has a maximum depth of about 35 metres.

LINE 250E:- High 4 to 6 millisecond responses were observed between 430S and 575S. The disseminated source is considered to have a *maximum* depth of the order of 50 metres. This zone correlates to similar responses seen at about the same co-ordinates on lines 150E and 350E.

A minor response of 3 to 4 milliseconds above background centred at 075N correlates with those seen on previous lines at similar positions.

LINE 350E:- Between 450S and 562S chargeabilities in excess of 6 milliseconds above the 10 millisecond background were observed with no material change in apparent resistivity. The source is considered to be disseminated sulphides and/or graphite within material beneath the basalt cover. The maximum depth inferred by the profile form is 50 to 60 metres. Whereas this response is clearly seen on the lines to the west, it is not seen to the east of this line.

LINES 400E to 950E:- No significant features were observed.

LINE 1000E:- High 20 millisecond chargeability responses recorded between 150N and 250N are inferred to have a disseminated source at a maximum depth of between 25 and 40 metres within the sandstones beneath the basalt cover.

A minor single point 4 to 5 millisecond response at 062N

coincident with a resistivity high was recorded over the tailings dump.

LINE 1050E:- A 13 to 14 millisecond anomaly recorded at about 188N is due to a disseminated sulphide and/or graphite source at a maximum depth of about 50 to 60 metres. This response comes from the zone to the immediate north of the western flank of the fault zone *but is not due to the fault as such*. However, should graphite and/or sulphide mineralisation be present within the fault zone, this may well give rise to the response.

A second 6 millisecond response centred at 088N also from a source showing no apparent resistivity contrast with the enclosing rocks is considered to have a maximum depth to source of 25 metres and a southerly dip. This is probably due to a sulphide enriched zone from within the skarn zone.

A small 4 to 5 millisecond response at 087S from a source twice as resistive as the enclosing skarn rocks could be due to sulphides and silicification within the skarn.

LINE 1150E:- The most striking feature observed was a 25 millisecond anomaly centred at 138N, but whose source lies between 088N and 175N. The maximum depth to the source is of the order of 25 metres and the source is some three times

more resistive than the enclosing rocks. The source lies within the sandstones to the immediate north of the Bismuth Creek Fault. The source is quite obviously *disseminated* sulphides and/or graphite.

A 6 to 8 millisecond anomaly above the 8 millisecond background centred at 062S is associated with a doubling of apparent resistivity. This source is considered to be near surface, less than 20 metres deep, and of disseminated origin, probably sulphide enrichment within the calc-silicate skarn zone.

A broad 6 millisecond anomaly centred at 138S is interpreted as originating in a *resistive* sulphide source beneath the basalt cover which overlies it.

LINE 1250E:- A 15 to 16 millisecond chargeability response from a source showing no resistivity contrast with the enclosing rocks, was noted between 025N and about 150N. Two distinct zones are inferred, one at 112N and the other at 037N. As in the case of previous lines, the sources lie to the *immediate* north of the Bismuth Creek Fault.

A broad chargeability anomaly centred at 137S of up to 6 milliseconds above background, correlates with responses to the north-west and south-east, semi-parallel to the Bismuth Creek Fault. The maximum depth may be as great as 75 metres, but

additional pole-dipole work would be required to confirm or deny this. The source is again disseminated as there is no appreciable change in the apparent resistivity profile.

LINE 1350E:- High chargeabilities of greater than 8 milliseconds above background were observed between 175N and 037S wholly from within the Moina Sandstones. No less than three distinct zones have been identified; at 137N, at 088N and at 00; all of which have a disseminated sulphide and/or graphite source.

The Bismuth Creek Fault zone crosses the traverse at about 095S and, under this, skarn has been mapped between the fault (at 095S) and about 075S *north-east* of the fault. Local 6 to 4 millisecond peaks at 088S and 130S probably mark the position of minor sulphide enriched zones from within the skarn, and chargeable zones within the Moina Sandstones respectively, either side of the fault. The depths to source of these two responses are less than 15 to 20 metres.

A broad high centred at 212S some 75 metres south-west of the fault, wholly within the mapped Moina Sandstone, is interpreted as originating in a disseminated sulphide source at a maximum depth of 40 metres. The the south-east, this high develops into a major anomaly centred at 288S on line 1450E.

LINE 1450E:- Substantial 15 to 20 millisecond chargeabilities were observed over practically the whole line surveyed. The exception was for about 100 metres north of the mapped

position of the Bismuth Creek Fault zone at 190S where *abnormally low* chargeabilities were recorded.

North-east of the fault between 075S and 200N the whole section was above 15 milliseconds and sulphides (or graphite) of the order of ½% to 1% over the whole width are inferred. The depth to source is considered to be shallow, from 10 to 25 metres at most.

South-west of the fault zone, chargeabilities reach 25 milliseconds at 275S, but remain above background between 250S and 350S.

From a study of the profiles together with the contour interpretation thereof and the associated geological map, the following conclusions and comments can be made.

1 - The most striking feature observed on the chargeability contour plan is the north-west/south-east strike of chargeable zones within the Moina Sandstones in the north-east quadrant of the area surveyed. This strike direction is semi-parallel to the Bismuth Creek Fault, but is clearly visible 200 metres south-west of it. The chargeability *is not* accompanied by any material change in apparent resistivity which clearly indicates the source to be "resistive", and the causative mineralisation to be disseminated sulphides (or perhaps graphite). The vertical field magnetic data very clearly shows that magnetite is

not the source, as magnetite percentages of the order of 10% would be required to produce the induced polarization responses observed here, and such magnetite segregations are not indicated by the magnetic fields observed. Interbedded graphite and/or sulphides are in fact the only possible explanation for the observed features.

2 - Over the abovementioned section, the resistivity contour plan should be recontoured to exhibit a parallel trend to the chargeability data, which is considered to be the actual trend of this data.

3 - On the whole, the apparent resistivities over the basalt covered areas are lower than those over the Ordivician sediments. This is due to the "dampening" effect of the more intensely oxidised near surface basalt and the conductive basalt-derived soils. The more resistive features observed over the basalt covered areas will tend to indicate the structural trends in subsurface Ordivician sediments, while the resistivity lows will tend to indicate only deeper and/or more intense oxidation of the basalt and its soil cover.

A study of the apparent resistivity data *bearing in mind* the dampening effect of conductive overburden/oxidation on the amplitude of *resistive features* clearly infers the

trend of the Ordivician sediments underlying the basalt cover to be north-west/south-east to west-north-west/east-south-east for the most part.

4 - The chargeability observed over the skarn zones is high, but is somewhat eclipsed by the unexpectedly high responses from sections within the Moina Sandstone unit.

5 - Chargeability background over the basalt cover remains fairly uniform between 10 and 12 milliseconds, which can be considered to be normal. Within these limits, higher backgrounds were noted for some hundreds of metres around two main areas:

[A] Centred at about 500S for 200 metres east and west of 200E, and about 200 metres wide,

and

[B] For some 200 metres around 650E/250S.

With regard to [A] above, two mutually perpendicular (FIG 1) electrical soundings carried out at 050E/500S inferred some 20 to 30 metres of conductive (60 ohm-metres) basalt cover to be present overlying resistive (1000 ohm-metres) Ordivician sediments below. An analysis of the chargeability data from these soundings inferred a chargeable source at a depth of the order of 50 metres. Thus the observed increase in chargeability background observed *IS* due to

increased chargeability *WITHIN* the underlying Ordivician sediments. However, there is no way of differentiating between skarn and chargeable Moina Sandstone sequences. The generally higher vertical magnetic field observed over the eastern section of the induced polarisation rise (i.e. around 400E/450S) may *infer* skarn. The signature seen through 20 to 30 metres of basalt cover could well be similar to this picture, as the very sharp, high localised fields over the outcropping skarn in the Mine area are due to near-surface magnetite outcrop.

The second zone centred at 650E/250S is also accompanied by higher regional magnetic fields and is also considered of interest, but the same reservations as expressed above also apply here.

6 - Two mutually perpendicular electrical soundings were (Fig 2) carried out centred at 900E/190S to ascertain the electrical characteristics of the basalt and underlying magnetite fluorite skarn zone in the vicinity of diamond drill hole Mt. Lyell No. 2 (ML-2). Due to vertical and lateral inhomogeneity the two profiles show gross divergence, however, the following tentative conclusions can be drawn.

(i) The soil cover had an intrinsic resistivity of about 150 ohm-metres and was about 4 metres thick

and showed background chargeabilities of the order of 28 to 30 milliseconds which are considered to be anomalously high.

(ii) The basalt cover was relatively conductive (38 ohm-metres) and was about 25 to 30 metres in thickness. (40 metres logged in ML-2). The intrinsic chargeability within the basalt varied between 10 and 18 milliseconds.

(iii) The underlying magnetic skarn zone was more resistive than the overlying basalt (200 ohm-metres plus) and considerably more chargeable (from 40 to 60 milliseconds) The depth to the chargeable zone was interpreted to be of the order of 35 to 60 metres.

(FIGS 3,4 SHOW INTERPRETIVE SECTIONS)

CONCLUSIONS AND RECOMMENDATIONS

1 - Both the gradient data and the electrical soundings clearly demonstrate the skarn zones to be both resistive and chargeable as expected. However, the substantial induced polarization anomalies unexpectedly located from within the Ordivician Moina Sandstones somewhat eclipse them.

2 - The highly chargeable zones within the Moina Sandstone are due to either disseminated sulphide or graphite. It is strongly recommended that the causative material be unambiguously identified should they be considered of

possible economic interest on other than geophysical grounds (i.e. due to associated geochemical responses etc.)

3 - A study of the highly chargeable zones within the Moina Sandstone located to the north-east of the Bismuth Creek Fault, indicates a strike direction for these zones, and presumably for the sandstone units also, of about 095°, while the Bismuth Creek Fault itself has a strike of about 135°.

4 - Two possible additional skarn zones may be present beneath the basalt centred at 500S/200E and at 650E/250S. However, both could be due to chargeable sequences within the Moina Sandstones such as those defined to the north-east of the Bismuth Creek Fault.

5 - No specific diamond drilling recommendations are made over any of the geophysical anomalies located, as it is considered that additional arguments based on structural analysis and/or geochemical indications would be required to justify this step.

The author looks forward to discussing the data in the near future.

Respectfully submitted on behalf of:

SCINTREX PTY. LTD.

A.W. HOWLAND-ROSE, MSc, DIC, AMAusIMM, FGS.

GEOPHYSICIST

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221177

Ohm-metres

700
600
500
400
300
200
100
80
70
60
50
40
30
20
10

N-S

E-W

MOINA

50E/500S

I.P. EXPANDER

a = 3-250m
b = 1-2m

chargeability - milliseconds

14
12
10
8
6
4
2

JOB No. TAS-030
8.12.75

E-W

N-S

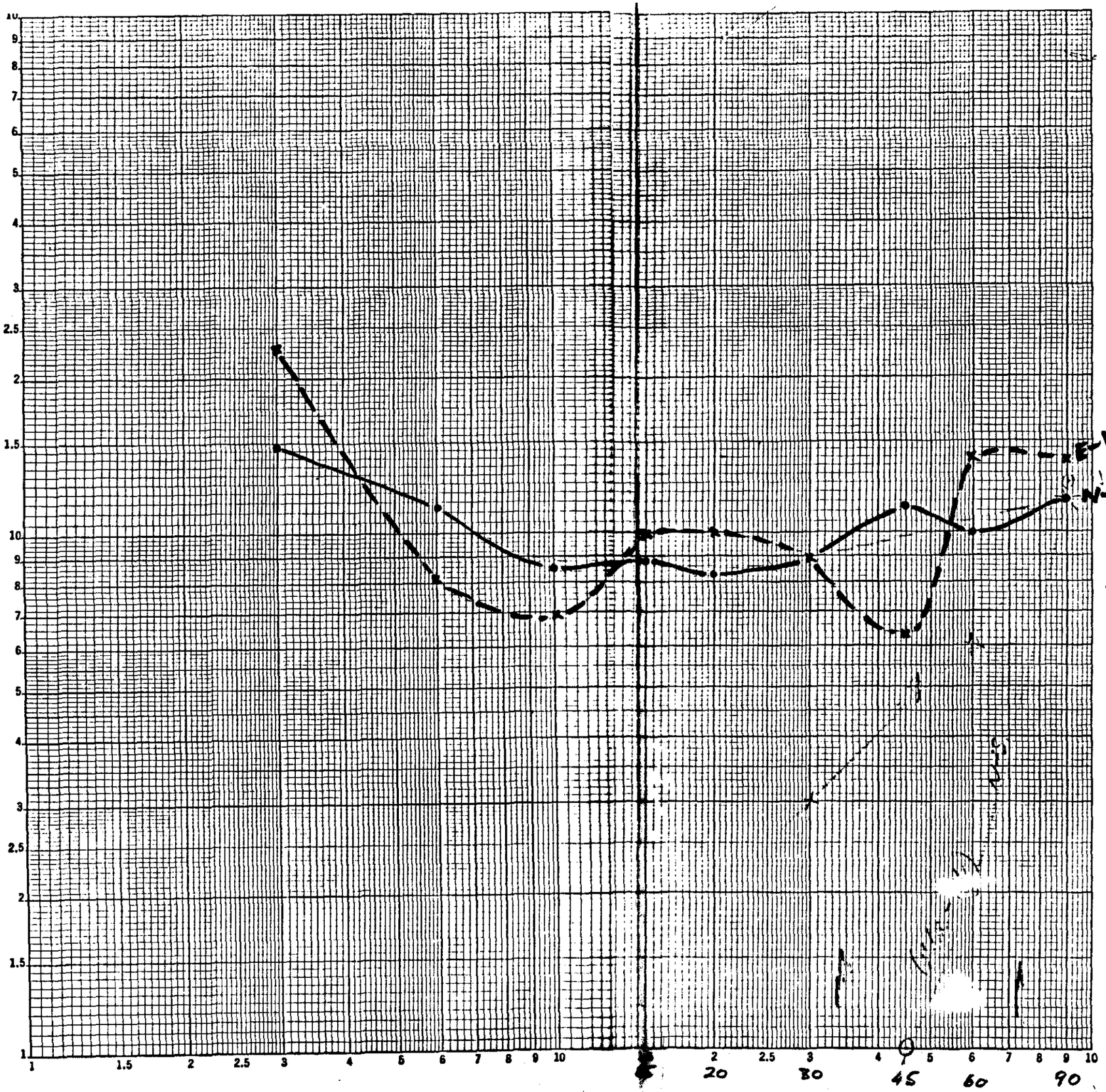
1.5 2 2.5 3 4 6 8 10 15 20 25 30 40 50 60 70 80 100 120 200 250 metres

5 cm

FIG 1

17.

221178



+ 900E/190S

(NS)

4M - 150 Ω.M.

28m

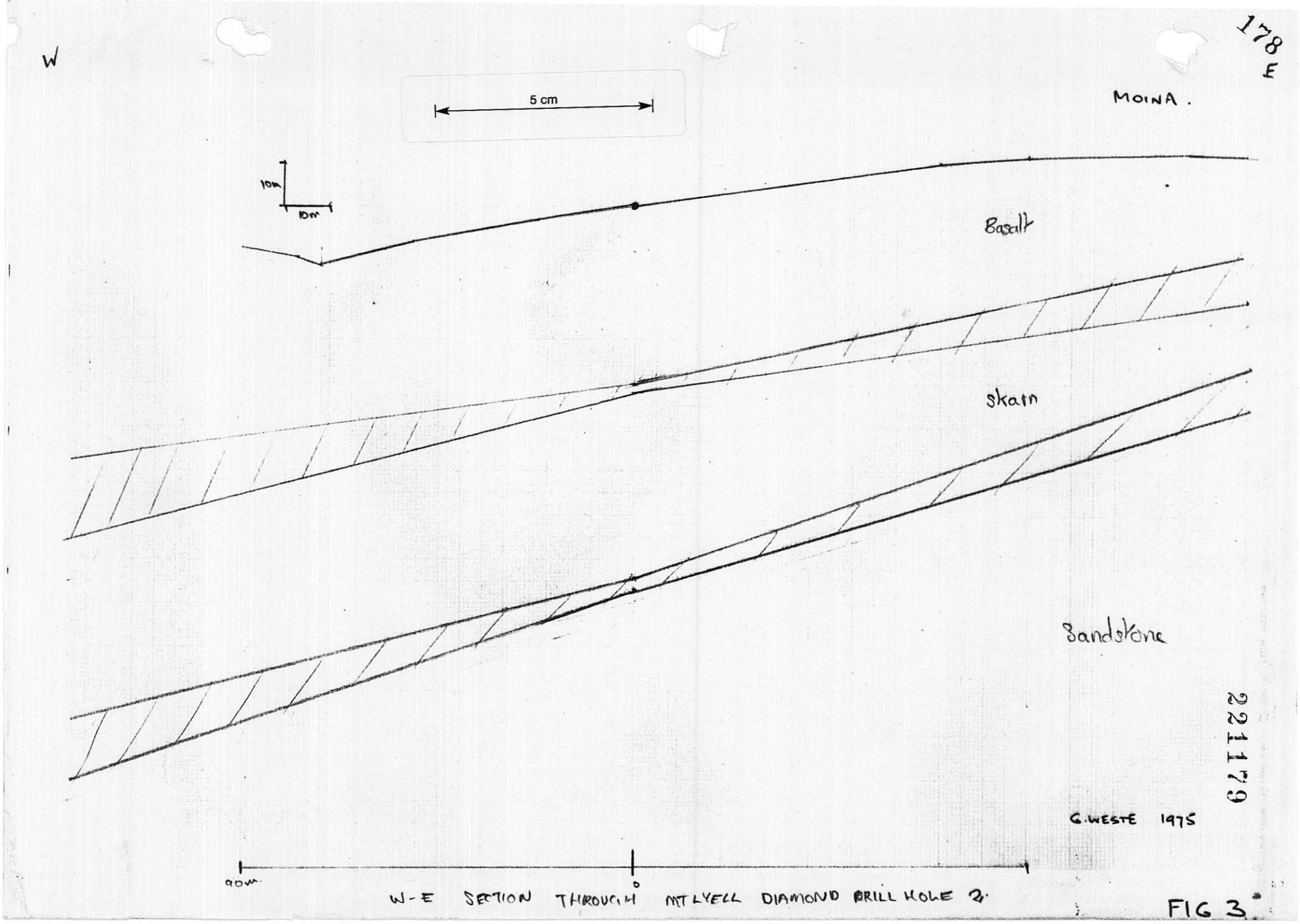
38 Ω.M.

160 Ω.M.

JOB N°: TAS-030
DATE : 2/12/75
AREA : MONA
EXPANDER
a=(3m-90m)
b=1m

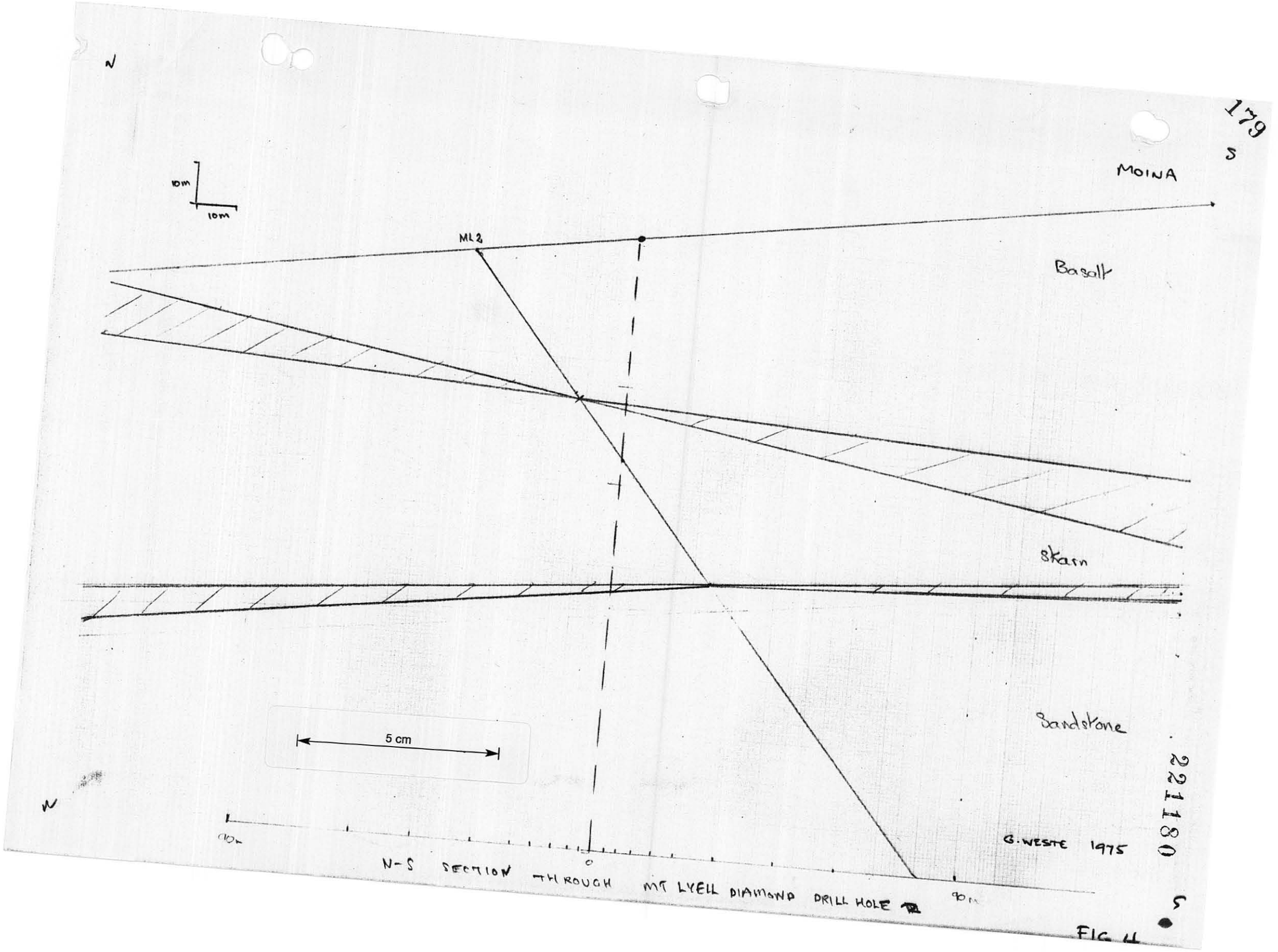
5 cm

FIG 2



W-E SECTION THROUGH MT LYELL DIAMOND DRILL HOLE 2.

FIG 3.



N-S SECTION THROUGH MT LYELL DIAMOND DRILL HOLE

FIG 4

179
2211800

APPENDIX 18

INVESTIGATION OF AN INDUCED POLARISATION ANOMALY - SHEPHERD AND MURPHY GRID - G. WESTE

The time domain electrical induced polarisation gradient array survey carried out over most of the Shepherd and Murphy grid by Scintrex Pty. Ltd. located a chargeability anomaly with a peak extending from 1150 m E/138 m N to 1350 m E/80 m N (drng. No. 78-145).

The rocks outcropping in the area are strongly fractured "Moina Sandstone" adjacent to the Bismuth Creek fault. To help determine the nature of this anomaly a pole - dipole survey was carried out along grid lines 1100 E, 1150 E, 1200 E, 1250 E, and 1300 E in February 1975 by Comalco personnel. A three array spacing was maintained and potential electrode spacings of 25 m and 50 m were used to measure variations in the anomaly's character with depth. The equipment used was a 2.5kw transmitter with a Scintrex IPR7 receiver .

GRADIENT ARRAY DATA - (See drg Nos. 78-147 & 148)

Peak Location	Chargeability (ms)	Approx. Depth (m)	Resistivity (Ωm)
1150 E/138 N	31	50	2000
1250 E/100 N	30	35	2000
1350 E/ 80 N	29	50	1800

POLE-DIPOLE DATA - (See attached profiles)

Line 1100 E.

The chargeability and shape of the source changes little with depth. The source may dip steeply south and the top is probably shallower than 25 m as there are indications of a secondary peak developing.

Line 1150 E.

The profile data indicates that the maximum depth to the chargeable zone is about 25 m shown by a twin peak at 50 m and a single peak at 25 m. The chargeability appears to be greater at 25 m than at 50 m. The exact relationship of chargeability to resistivity is not very clear but if the peaks do correspond then the source is relatively resistive and may dip steeply to the south. Overall the resistivities are lower than expected.

Line 1200 E.

The profiles at 25 m and 50 m are very similar indicating that the body is essentially the same in shape and chargeability at both depths.

Line 1250 E.

The anomaly is complex. The profiles at 25 m and 50 m are again very similar.

Line 1300 E.

The source is still not simple but appears to be close to the surface.

COMPARISON PROFILE

For comparison purposes a gradient array profile was conducted over similar pyritic "Moina Sandstone" exposed by excavations at the Wilmot damsite 2 km to the north west. The profile (attached) is very similar to that under investigation. The relatively low resistivities encountered were lower than expected.

CONCLUSIONS

1. The source of the chargeability anomaly between 1100 E/180 N and 1350 E/ 80 N is steeply dipping (south?) and probably relatively resistive.

2. On line 1150 E the maximum depth to the source is about 25 m, but on the other lines little change occurs with depth.and the depth to source may be less than 25 m.

3. The anomaly is close to, but not quite parallel to, the Bismuth Creek Fault. Pyritic sandstones exposed at the Wilmot damsite (also beside the same fault) show similar chargeabilities. The source is either (a) a steeply dipping pyrite rich sandstone (Moina Sandstone near Cethana is rich in framboidal pyrite) or (b) pyrite introduced or remobilised along fractures associated with the Bismuth Creek Fault.

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221183

F1 7/74 TASMANIA

MOINA: SHEPHERD & MURPHY GRID

Time Domain Electrical Induced Polarisation

Pole - Dipole Chargeability survey

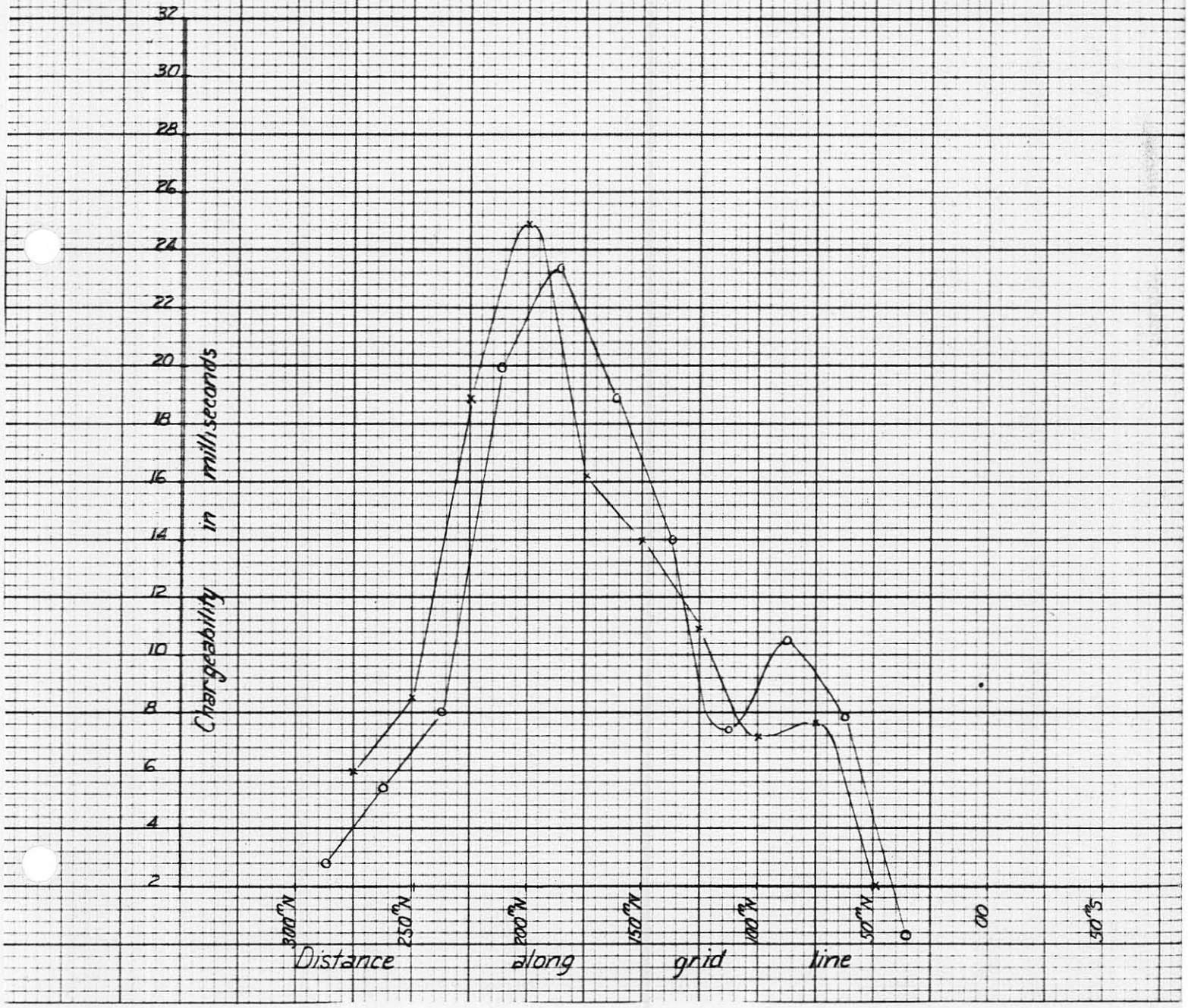
o——o a = 25m } n = 1

x——x a = 50m }

∞ ← C¹ C² P² P¹

LINE 1100mE

5 cm



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221184

EL 7/74 TASMANIA

MOINA: SHEPHERD & MURPHY GRID

Time Domain Electrical Induced Polarisation

Pole - Dipole Chargeability survey

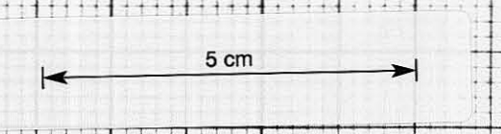
a = 25m } n = 1

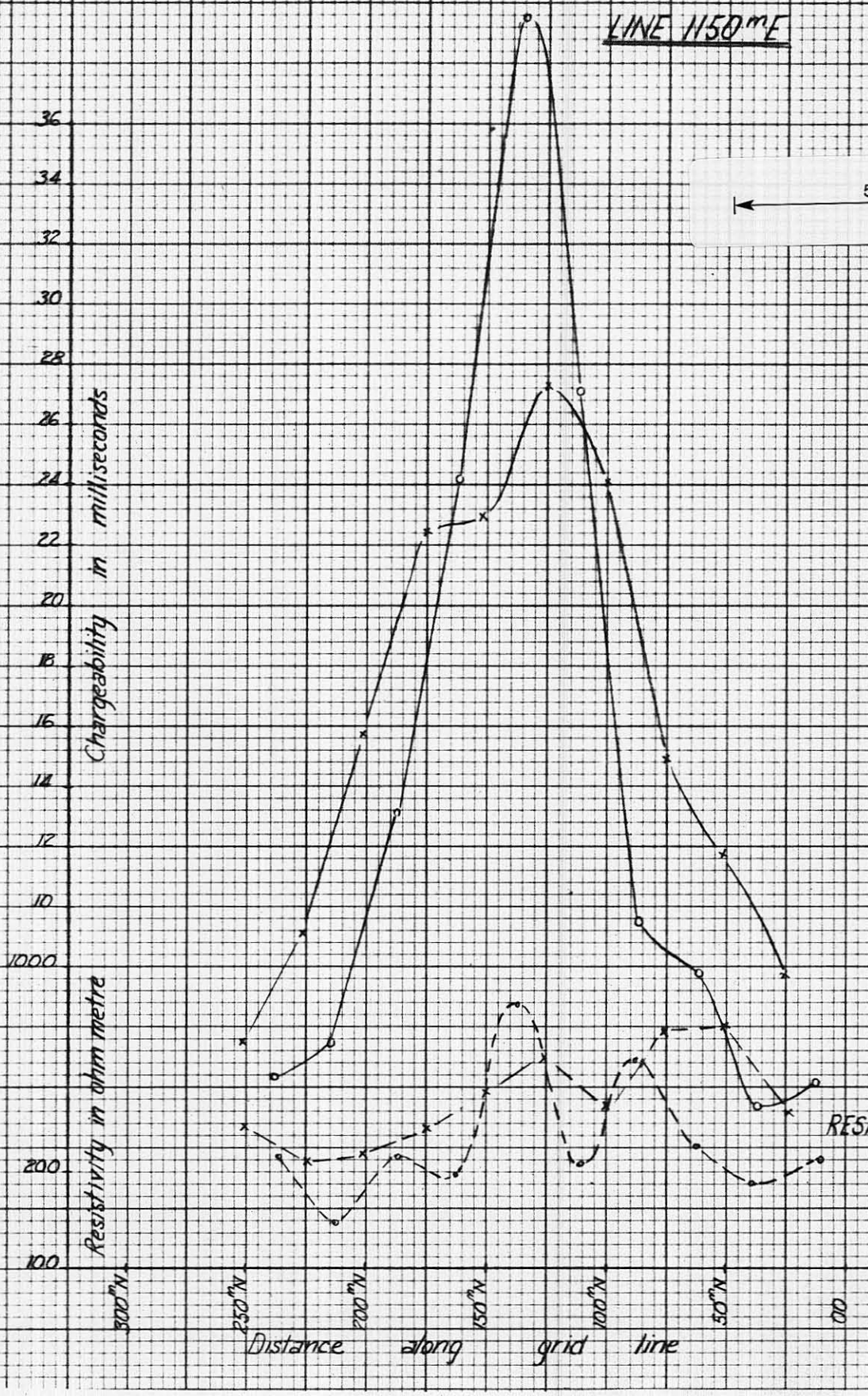
a = 50m

C² P² P¹

C¹

LINE 1150 m E





184

221185

EL 7/74 TASMANIA

MOINA: SHEPHERD & MURPHY GRID

Time Domain Electrical Induced Polarisation

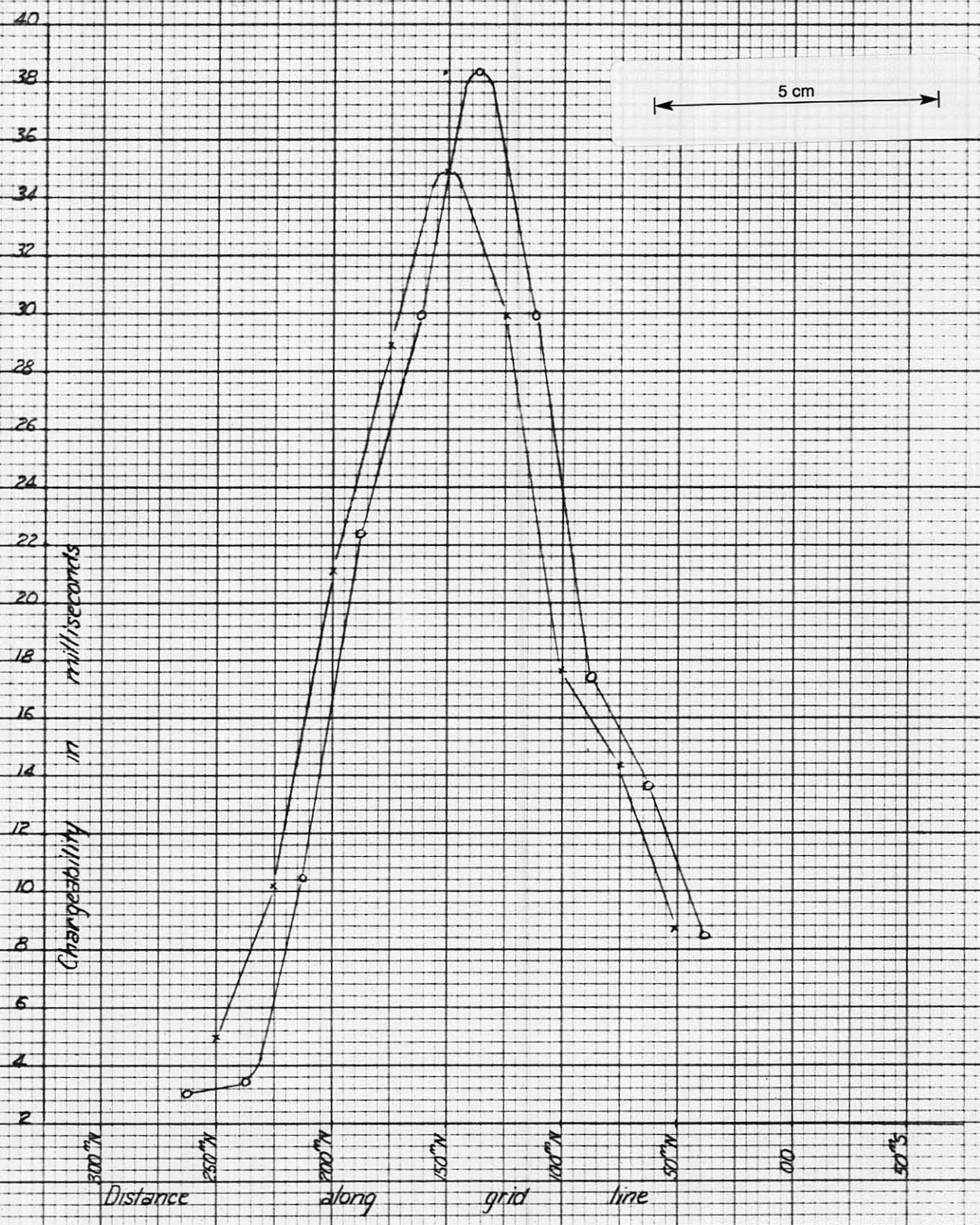
Pole - Dipole Chargeability survey

o—o a = 25m } n = 1

x—x a = 50m

∞ ← C¹ C² P² P¹

LINE 1200m E



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221186

EL 7/74 TASMANIA

MOINA: SHEPHERD & MURPHY GRID

Time Domain Electrical Induced Polarisation

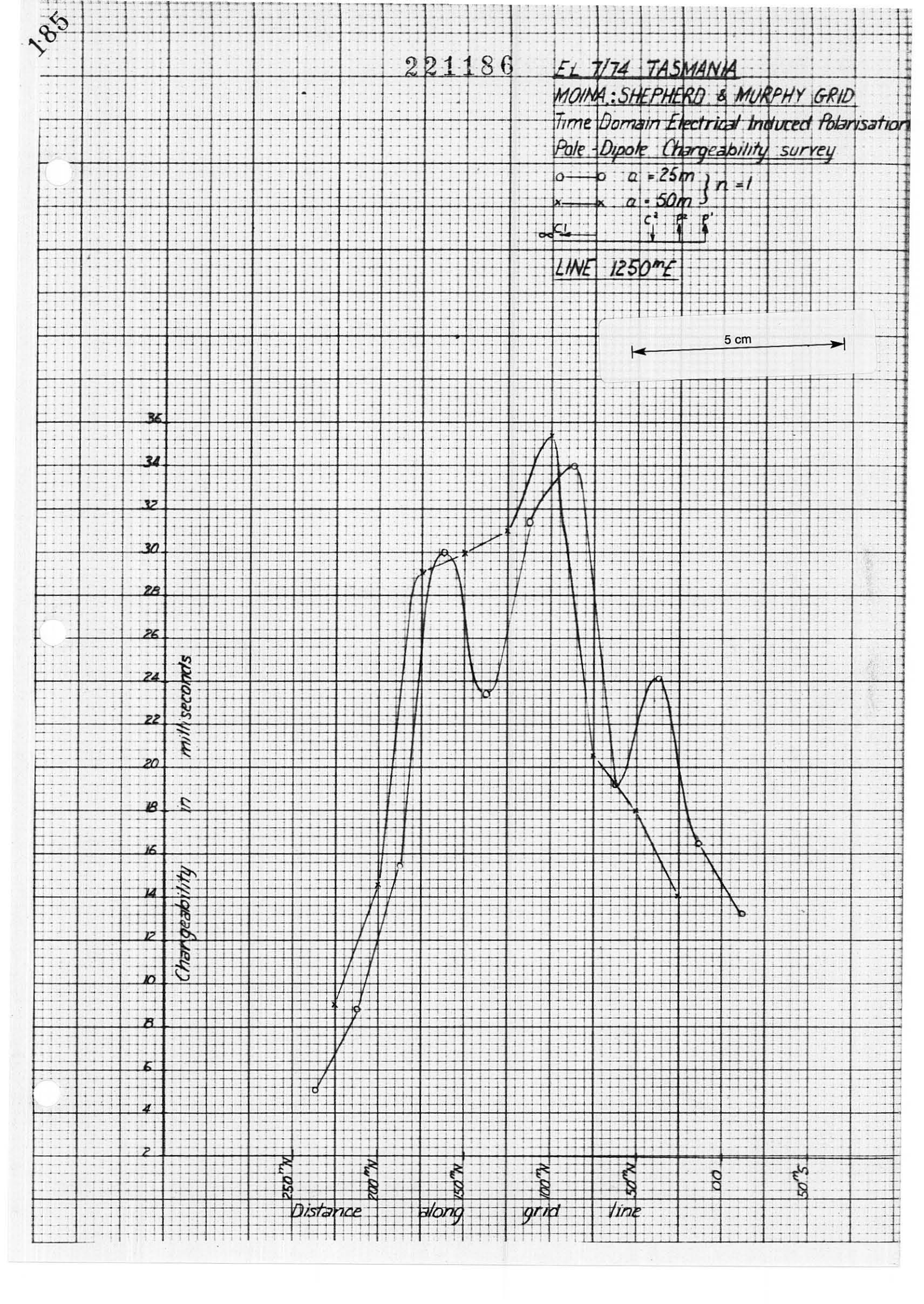
Pole - Dipole Chargeability survey

o——o a = 25m } n = 1

x——x a = 50m }

C1 C2 P2 P1

LINE 1250mE



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221187

EL 7/74 TASMANIA

MOINA: SHEPHERD & MURPHY GRID

Time Domain Electrical Induced Polarisation

Pole - Dipole Chargeability survey

o—o a = 25m } n = 1

×—× a = 50m }

C¹ C² P² P¹

∞

LINE 1300mE

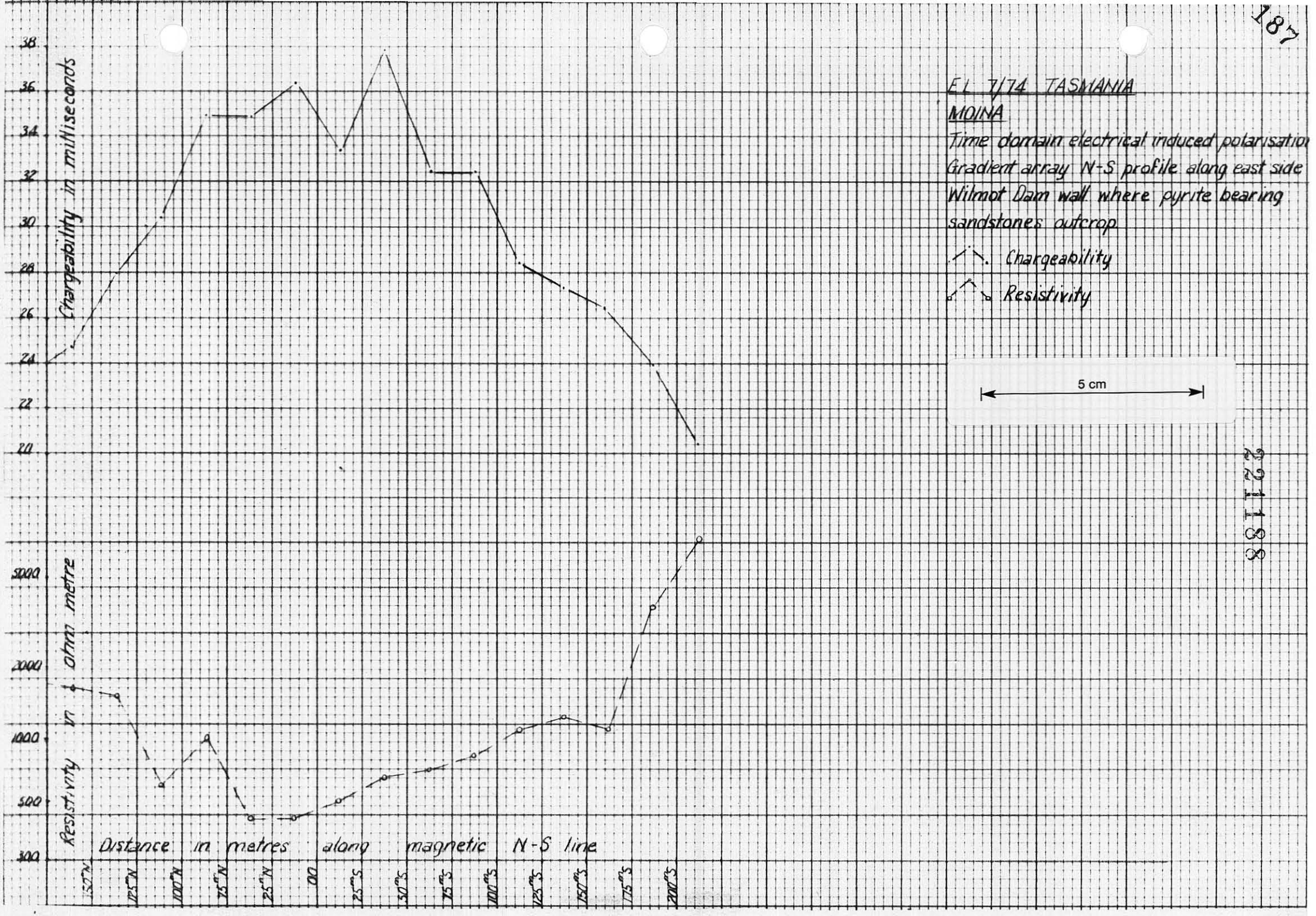
5 cm

Chargeability in milliseconds

50 48 46 44 42 40 38 36 34 32 30 28 26 24 22 20 18 16 14 12 10 8 6 4 2

250mN 200mN 150mN 100mN 50mN 00 50mS 100mS

Distance along grid line


EL 7/74 TASMANIA
MOINA
Time domain electrical induced polarisation
Gradient array N-S profile along east side
Wilmot Dam wall where pyrite bearing
sandstones outcrop
Chargeability
Resistivity
5 cm
Chargeability in milliseconds
38
36
34
32
30
28
26
24
22
20
Resistivity in ohm metre
5000
2000
1000
500
300
Distance in metres along magnetic N-S line
150mN
125mN
100mN
75mN
25mN
00
25mS
50mS
75mS
100mS
125mS
150mS
175mS
200mS

221188

APPENDIX 19

INVESTIGATION OF "POST OFFICE" INDUCED POLARISATION ANOMALY - G. WESTE

INTRODUCTION.

Time domain electrical induced polarisation surveys were carried out in January 1975 and December 1976 by Comalco personnel over an area near the old Moina Post Office immediately to the east of the "Shepherd and Murphy" sheet on the adjoining "Iris River" sheet to follow up open-ended chargeability anomalies located during a Scintrex Pty. Ltd. gradient array survey: (drg Nos. 78-145 & 148)

THE SURVEYS.

A gradient array survey was carried out along lines 50 m apart from line 50 E to 350 E to cover the chargeability anomaly. Readings were taken every 25 m using a 25 m dipole and a Scintrex IPR7 receiver and a 2.5 kw transmitter. Profiles are shown on drawing Nos. 78-150 and 151.

The chargeability anomaly has a somewhat erratic west trend from approximately 50 E/210 N to 350E/150 N. The anomaly is not closed off to the west but has dropped to 24 ms. The highest chargeability is 30 ms on line 100 W. The shape of the anomaly is quite variable and appears to generally consist of multiple sources. The width of the anomaly or anomalies varies from 50 m to about 150 m. A broader anomaly of similar amplitude occurs to the north along lines 50 E to 150 E but was not followed to the east or west due to the close proximity of Lake Gairdner. The high chargeabilities do not correspond to any particular resistivity feature. Resistivity values vary from 500 to 1200 ohm metres. The apparent width of the anomalies indicates that they may be stratiform.

A pole-dipole survey was attempted over what appeared to be the simplest singlepeak of the anomaly on line 100 W at 180 m . The potential electrode spacings used were 25 m and 50 m with a 3 array spacing being maintained. The main purpose of the survey was to gain information on the depth to the chargeable material. This survey was abandonned because cattle kept chewing the electrode wires.

A pole-dipole survey over the broader anomaly to the north along line 50 E was successfully completed (profiles attached) and indicates that chargeabilities increase with depth and appear to dip to the north. The pole-dipole chargeabilities are much lower (about half) than expected. The resistivity increases with depth as expected.

The chargeability peak at 525 N is relatively resistive and the chargeability peak at 325 N relatively conductive.

DRILLING.

A vertical diamond drill hole (SMD 14) was collared at the 30 millisecond peak of the anomaly at 100 W/180 S and intersected limestone with mainly fine disseminated pyrite (see log of pyrite in core - SMD 14).

Thin section studies by H.W. Fander (attached) identified bands of 5-10% ultrafine pyrrhotite and pyrite associated with dolomite which in combination with the medium to fine pyrite observed in the hand specimen and the resistive host rocks would account for the observed chargeabilities. SMD 14 collapsed when casing was being withdrawn and was not eletrically logged.

CONCLUSION:

The chargeability anomalies in the "Post Office" area are due to fine grained disseminated pyrrhotite and pyrite associated with dolomites and appear to be stratiform.

The resistivity data does not indicate the presence of any massive pyrrhotite in the area covered or of any large amounts of disseminated pyrrhotite that could possibly be immediately adjacent to massive pyrrhotite.

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221191

CENTRAL MINERALOGICAL SERVICES PTY. LTD.

Date 23rd May 1977

SAMPLE REPORT (Mineralogy, Petrology, Ore Microscopy)

IDENTIFICATION
SMD 14/10.4
OPAQUES IN CALCAREOUS ROCK

Job No. CMS 77/5/23 Date Received: 17.5.77

Reference PA/T/MOINA 2

Sample No. SMD 14/10.4

Nature of Sample: D.D. Core

DESCRIPTION **SECTION No.** 21202

a. Hand Specimen:

Dark grey fine-grained rock. Disseminated sulphides.

b. Microscopic:

The following opaque minerals were detected in a polished section: carbonaceous matter, pyrite, pyrrhotite, and graphite.

Carbonaceous Matter Estimates are difficult, but this substance could well comprise 2 -3% of the rock; more importantly however, is its distribution as semi-continuous streaks, layers and ptygmatic veins. This is particularly well shown on a surface etched with HCl. The carbonaceous matter is probably sub-graphitic and can be expected to be a good conductor. Traces of graphite have formed sporadically, as thin, isolated flakes.

Sulphides These total about 1% or less, and occur as small, scattered, isolated grains 5μ to 100μ in size, occasionally up to 1mm as porous patches. It seems that most or all the original sulphide was pyrrhotite, but is now pyritised except for occasional small grains, judging from relict textures.

Unetched areas on the acid-treated surface are probably areas of dolomitisation, and seem to contain most of the sulphide (see next description - 13.6m).

H.W. Fander, M.Sc.

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221192

CENTRAL MINERALOGICAL SERVICES PTY. LTD.

Date 23rd May 1977

SAMPLE REPORT (Mineralogy, Petrology, Ore Microscopy)

IDENTIFICATION
SMD 14/13.6
CARBONACEOUS DOLOMITIC/ CALCAREOUS ROCK

Job No. CMS 77/5/23 Date Received: 17.5.77

Reference PA/T/MOINA 2

Sample No. SMD 14/13.6

Nature of Sample: D.D. Core

DESCRIPTION **SECTION No.** 21203

a. Hand Specimen:

Dark, fine-grained rock, with sulphides - conspicuous in some bands or zones.

b. Microscopic:

The polished section indicates that the same opaque minerals are present as at 10.4m. The distribution is different, however, as shown by an etched surface.

There are pyrite-rich bands in the rock, in which quite large pyrite crystals and aggregates have formed; some of these are lenses up to several millimetres long and 0.5mm wide, and others are up to 1-2mm across. In other parts of the rock the sulphides are generally much finer (2-100μ, mostly < 30μ) and scattered, as isolated grains and tabular crystals of pyrrhotite and pyritised pyrrhotite. Very rare grains of chalcopyrite were also detected.

The etched surface shows unattacked areas; immersion-oil determinations indicate that these are dolomite. Sulphides, particularly the coarse pyrite, are virtually confined to these areas of dolomitisation. Moreover, the distribution of sub-graphitic/carbonaceous matter is closely related to dolomitisation, often forming semi-continuous streaks alongside these areas.

The overall percentage of sulphides is perhaps 1 - 2% but rises to 5 - 10% in the zones described. Carbonaceous matter comprises perhaps 2%.

Study of a thin section would be necessary to arrive at a complete understanding of the rock.

H.W. Fander, M.Sc.

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221193

CENTRAL MINERALOGICAL SERVICES PTY. LTD.

Date 23rd May 1977

SAMPLE REPORT (Mineralogy, Petrology, Ore Microscopy)

IDENTIFICATION
SMD 14/38.0
DOLOMITISED CARBONACEOUS ROCK WITH PYRRHOTITE

Job No. CMS 77/5/23 Date Received: 17.5.77

Reference PA/T/MOINA 2

Sample No. SMD 14/38.0

Nature of Sample: D.D. Core

DESCRIPTION **SECTION No.** 21204

a. Hand Specimen:

Dark grey fine-grained rock, with traces of sulphides.

b. Microscopic:

An etched surface indicates that the rock is very largely dolomitised; the etching process brings out subparallel structures, or zones of dolomitisation which may be governed in some way by the distribution of presumably primary carbonaceous matter.

Fine to ultrafine sulphides are relatively abundant throughout; only the coarser grains are visible in hand specimen. Occasional pyrite aggregates up to 2 x 1mm are seen, but the predominant sulphide is pyrrhotite, as individual, irregularly shaped grains ranging from 1μ to 100μ and occasional larger clusters; most grains however, are < 30μ in size. The distribution is irregular though pervasive; some portions contain up to 5%, some 1% or less, but the overall average is around 2% (this includes traces of pyrite).

Carbonaceous matter is absent in some areas but fairly universally dispersed, as small wisps and streaks with more or less subparallel orientation; it comprises 2-3% of the rock.

The pyrrhotite is fresh, ie. very little is pyritised; there seems to be a correlation between its abundance and distribution, and the degree of dolomitisation.

H.W. Fander, M.Sc.

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221194

CENTRAL MINERALOGICAL SERVICES PTY. LTD.

Date 23rd May 1977

SAMPLE REPORT (Mineralogy, Petrology, Ore Microscopy)

IDENTIFICATION
SMD 14/52.0
DOLOMITISED CARBONACEOUS ROCK WITH PYRRHOTITE

Job No. CMS 77/5/23 Date Received: 17.5.77

Reference PA/T/MOINA 2

Sample No. SMD 14/52.0

Nature of Sample: D.D. Core

DESCRIPTION **SECTION No.** 21205

a. Hand Specimen:

Grey, fine-grained carbonate rock with very fine sulphides.

b. Microscopic:

This rock is very extensively dolomitised; the process has had two effects. It has introduced pyrrhotite and has redistributed the carbonaceous matter. This was probably fairly uniformly dispersed in the original limestone perhaps with some bedding, but now conforms to the boundaries of coarser dolomite rhombs. The result is a type of network pattern of carbonaceous matter (ie. the "net" being formed by carbon, the "holes" by dolomite rhombs).

Carbonaceous matter (probably subgraphitic but with poorly-defined optical properties) averages perhaps 2% of the rock; some portions contain very little, others much more, and the distribution is zonal.

Pyrrhotite is conspicuous and is closely related to dolomitisation though not confined to it; whilst it is thought that the dolomitisation probably introduced the pyrrhotite, it is possible that it was responsible for redistribution of already existent (?primary) sulphide. It comprises 2-3% of the rock, occurring as single grains 1-100μ and as clusters (in optical continuity) up to 300μ across.

H.W.Fander, M.Sc.

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221195

CENTRAL MINERALOGICAL SERVICES PTY. LTD.

Date 23rd May 1977

SAMPLE REPORT (Mineralogy, Petrology, Ore Microscopy)

IDENTIFICATION
SMD 14/76.0
CALCAREOUS ROCK, MINOR DOLOMITISATION.

Job No. CMS 77/5/23 Date Received: 17.5.77

Reference PA/T/MOINA 2

Sample No. SMD 14/76.0

Nature of Sample: D.D. Core

DESCRIPTION **SECTION No.** 21206

a. Hand Specimen:

Grey fine-grained rock with pale veins.

b. Microscopic:

Judging from an etched surface, only minor dolomitisation has occurred, in the form of more or less subparallel veinlike bodies.

There is a very close spatial correlation between the distribution of pyrrhotite and dolomite, confirming observations on previous cores; it seems to be more than a matter of redistribution of existing sulphide, and actual introduction of material seems to be involved.

Thus the dolomitic lenses, streaks and irregular veins contain 2-3% of relatively coarse pyrrhotite as individual grains up to 100μ and as loose clusters (but in optical continuity) up to 300μ in size. The remainder (ie. bulk) of the rock contains traces of ultrafine pyrrhotite grains, averaging 5-10μ in size

Carbonaceous matter is sparse (? < 1%) and is very fine-grained; it is evenly dispersed through the rock, with little tendency to form streaks or continuous bodies as in the other intersections.

Probably the semi-continuous pyrrhotite-bearing dolomite "veins" are mainly responsible for geophysical (I.P. anomalies). It may be possible to equate degree of dolomitisation with pyrrhotite development and I.P. effects.

H.W. Fander, M.Sc.

EL 7/74 TASMANIA
MOINA POST OFFICE ANOMALY
Pole - Dipole survey line 50E
a = 25m } n = 1
a = 50m }
Chargeability
Resistivity

5 cm

Resistivity in ohm metre

2000
1000
500
400
300
200
100

300°N
400°N
500°N
600°N
700°N

2211196

PYRITE IN CORE - SMD 14

EXAMINATION AT 40X ON SAWN CORE

by G. Weste

DEPTH METRES	BOX NO	PYRITE (%)	PYRITE GRAIN SIZE	DESCRIPTION
	1	<1	Coarse	
	1	1	Coarse	
13.0	2	1.5	Coarse to fine	Coarse pyrite with stylolites. Fine pyrite in darker limestone.
15.0	2	1-2	Fine	
18.3	2	1-2	Very fine to medium	Larger grains spheroidal in stylolites or alongside them. Fine disseminated pyrite in darker (carbonaceous?) material
19.0	3	1-1.5	Coarse	In stylolitic limestone.
	3	1.5	Coarse	Pyrite in stylolites.
	3	1.5	Fine & coarse	Stylolitic. Fine pyrite throughout.
20.7	3	2	Fine > coarse	The darker material contains most pyrite (fine).
27.0	4	0.5	Coarse > fine	Stylolitic
	4	1	Coarse > fine	Stylolitic
	4	1	Coarse > fine	Stylolitic
36.1	5	2-2½	Fine - medium	Banded limestone py concentration greatest in pale green bands.
38.0	5	2.5-3	Fine > coarse	Fecal pellet mudstone. Fine pyrite in and surrounding pellets.
	5	1-2	Fine - medium	Dark grey limestone.
44.0	6	1.5-2	Coarse = fine	Fecal pellet mudstone.
	6	1	Fine to coarse	Dark grey muddy limestone.
	6	1	Coarse & fine	Fine pyrite in dark grey limestone, coarse in stylolites.
52.0	7	3	Fine to coarse	Fecal pellet mudstone.
	7	2	Medium	Pale grey and dark grey limestone. py in dark limestone.
59.0	8	1.5	Medium	Mottled limestone.
	8	<1	Coarse - medium	Grey limestone with small white blebs.
67.0	9	<0.5	Coarse - medium	Pale grey limestone with variably sized white blebs.
	9	2.5	Coarse - medium	Banded limestone with bands of white calcite with pale green centres.
75.0	10	1	Medium	Dark grey limestone with thin wavy white white calcite bands.
	10	1.5	Fine	Grey limestone.
82.7	11	< 0.5	Fine	Grey limestone with white lensoidal blebs

221198

A REPORT ON

A RECONNAISSANCE MAGNETIC INDUCED POLARIZATION SURVEY

OVER THE MOINA GRID EL 7/74 NEAR WILMOT, TASMANIA

ON BEHALF OF

COMALCO LIMITED

MICROFILMED

OPEN FILE

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221139

PRIVATE AND CONFIDENTIAL

A REPORT ON
A RECONNAISSANCE MAGNETIC INDUCED POLARIZATION SURVEY
OVER THE MOINA GRID EL 7/74 NEAR WILMOT, TASMANIA
ON BEHALF OF
COMALCO LIMITED

BY

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GEOPHYSICIST

SYDNEY, N.S.W.

JULY, 1977
TAS-040

541

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221200

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221201

SCINTREX PTY. LTD.
GEOPHYSICAL CONSULTANTS AND CONTRACTORS

SUMMARY

A forty day magnetic induced polarization survey executed over the Moina Grid (EL 7/74) has defined a substantial number of major internal polarization anomalies. In both the magnitude and areal extent of anomalies, this area is unique.

The predominant strike is east west as gauged from both the internal polarization responses and from the significant conductors (H_N maxima), which have also been located. The form of all anomalies suggests steeply dipping sources, while the interline correlation suggests lensoidal sources having strike lengths of up to 800 metres. As these characteristics cannot be assigned to any componenet of gently dipping limestones which underlie the bulk of the survey area, it is considered that the sources must lie underneath, or have been introduced into, or re-mobilised within the limestones. As such they warrant further investigation.

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221202

A REPORT ON

A RECONNAISSANCE MAGNETIC INDUCED POLARIZATION SURVEY OVER THE MOINA GRID EL 7/74 NEAR WILMOT, TASMANIA ON BEHALF OF COMALCO LIMITED

INTRODUCTION

At the request of Mr. P. Askins Comalco Limited's Senior Geologist, Northern Tasmania, Scintrex Pty. Ltd. executed magnetic induced polarization surveys over the Moina grid near Wilmot Tasmania on about 40 production days between 5th April and 12th May, 1977.

These surveys were undertaken by Mr. B. Ekstrom Senior Scintrex operator with assistants provided by Comalco Limited. On site geological supervision was provided by Mr. G. Weste and Mr. P. Askins, while such additional geophysical direction as was required was provided by A.W. Howland-Rose who visited Comalco for the purpose of discussions on 22nd April and 6th June, 1977.

The method is briefly described in Appendix 'MIP'.

DATA PRESENTATION

The data profiles are appended to this report at the following scales.....

Horizontal scale 1:2500

Vertical scales....

Chargeability, M_1, M_3, M_5	1 centimetre = 1 milligamma/gamma
Decay form, ΔM	1 centimetre = 1 milligamma/gamma
Horizontal magnetic field, H_N	1 centimetre = 10%
Secondary magnetic field, H_S	1 centimetre = 1 milligamma/ampere

The chargeability and normalised horizontal magnetic field have been contoured by current dipole block on Plates 1 and 2 at the horizontal scale of 1:2500. The relative "highs" and "lows" in each parameter have been emphasised by shading. These two contour interpretations have been combined into an interpretation plan on Plate 3, also at the horizontal scale of 1:2500.

DISCUSSION OF THE DATA

GENERAL COMMENTS

Without a doubt the results obtained over the Moina grid are unique. The very material changes in the relative chargeability are some of the largest yet observed, and the substantial number of such anomalies is unique.

A substantial portion of the grid area is covered by a semi-horizontal limestone (with dolomite), while the anomalous chargeability responses observed are clearly lenticular and can be traced over hundreds of metres. Thus they are related either to structures *beneath* this limestone cover, or perhaps

to alteration of, or within, the lower levels of the limestone. The maximum guesstimated depths to source on the whole, infer sources to be either deep within or under the limestones, but as maximum depths are difficult to gauge with precision, the true source depth remains to be proven.

The *form* of the data is more characteristic of a series of vertical or steeply dipping sources rather than horizontal units within the limestone itself. However, since these surveys were commenced, both sulphide and carbonaceous material has been identified within the limestones, but the distribution within them is still unknown. Should either the sulphides and/or carbonaceous material indeed be the source material, then it is suggested that these will tend to be confined to semi-vertical zones rather than horizontal units. Therefore, bearing in mind the horizontal nature of the limestones, it is suggested that the source material may have been either *introduced* or *"remobilised"*.

Each line is separately discussed and each anomaly commented on. The correlation of the anomalies between lines is generally good and is shown on the contour interpretations. Although discussions have been held on the geological implications of the data, the emphasis of this report is on the *physical characteristics* inferred from the geophysical data, as the author does not feel wholly competent to relate these in detail to the geology of the area, which is complex in detail.

DETAILED DISCUSSION OF LINES

It should be emphasised that both the maximum horizontal magnetic field data and the chargeability data display the *relative* chargeability and conductivity (H_N) within each current dipole, and not the *absolute* conductivity or chargeability (see Appendix MIP).

A general comment on each anomaly is given in code form as set out below, followed by specific comments on each response.

In order to facilitate the description of each anomaly, the following shorthand code has been adopted.....

#001 Number of anomaly on profile.

F/T Maximum inferred width of the source. Where this is left blank, the source is considered to be "narrow".

C Centre of response (assumed to be vertically above source centre).

T Anomaly type (see Appendix MIP).

S Size of anomaly in milligamma/gamma *against* the *internal* (negative) current flow in the case of Type 'D' responses.

(R) size of anomaly relative to *background*.

D The estimated depth to source.

DF Decay form; S: slow, N:normal, F:fast

Note: Groups of anomalous responses which are related are shown so(018/019/020) prior to description. Such groups are

interpreted to respresent segregations of chargeable material within a generally chargeable unit.

LINE 900W

(001/002/003/004/005)

#001 F/T:-; C:240S; T:A; S:(R)-6; D:25/40M; DF:S/N

Major contact in chargeability from the immediate south, also the rocks to the north are more resistive.

#002 F/T:-; C:175S; T:B?; S:(R)-6?; D:?; DF:S

Minor change in chargeability within a broad zone, superimposed on a gradual but marked change in resistivity - rock type?

#003 F/T:-; C:125S?: T:A; S:(R)-5?; D:25M+; DF:N

Situated on overlap between arrays, making interpretation difficult.

#004 F/T:-; C:050S; T:D; S:+5(180%); D:25/50M; DF:N

Type 'A' response either side of this conductive axis causes discharge through it. It is not known whether the source is conductive by virtue of interconnected sulphides and/or graphite or wholly electrolytic conduction within say a fault.

#005 F/T:-; C:025N; T:A; S:-4; D:75M; DF:N

Northern edge of the series of chargeable responses recorded above (#001 to #005).

(006/007/008)

#006 F/T:-; C:300N; T:C; S:(R)-7(200%); D:30M; DF:N

Narrow, relatively shallow (30 metres?) chargeable source.

#007 F/T:362N-375N; C:375N; T:C; S:-9; D:50M; DF:S

Major Type 'C' response associated with slightly increased conductivity (140%).

#008 F/T:-; C:425N; T:D??; S:+2; D:25M?; DF:N

Possible Type 'D' anomaly *along strike* but may only represent discharge of the Type 'C' to the immediate south.

#009 F/T:-; C:575N; T:C; S:(R)-13; D:50M; DF:N

This type 'C' response is associated with H_N of about 105% as against 60% and 80% background.

#010 F/T:675N-760N; C:725N?; T:B/C; S:(R)-4; D:50M?; DF:N

This response straddles an array overlap and therefore may be wider than normal.

(111/112)

#011 F/T:-; C:850N; T:C; S:-4; D:50M; DF:N

This chargeability response is close to, or on a rock type change.

#012 F/T:-; C:900N; T:C; S:-4; D:60M; DF:N/F

The source material is different in that it has a fast decay form, and thus an inferred finer than average (for the area) grain size.

#013 F/T:-; C:962N; T:D; S:+6; D:50/60M; DF:N

This substantial external polarization response is associated with a marked increase (to 140%) in H_N with respect to the north (70%). It is considered that the conductor *may* increase in significance along strike.

#014 F/T:-; C:1062N; T:A; S:(R)-15; D:60M; DF:N

This major Type 'A' response must, in part, decay through the conductor located to the south (Type 'D', #013).

LINE 800W

A series of relatively minor responses (#015-#020) were recorded between 300S and 100N

(021/022/023)

#021 F/T:-; C:300N; T:C/B; S:-5; D:50M; DF:S

#022 F/T:-; C:375N; T:C; S:-4; D:?; DF:N

#023 F/T:-; C:430N; T:B; S:(R)-4; D:40M; DF:N

#024 F/T:-; C:525N; T;B; S:(R)-5; D:50M; DF:N

(025/026)

#025 F/T:-; C:625N; T:A; S:-6; D:50M; DF:S/N

#026 F/T:-; C:675N; T:A; S:-5; D:50M?; DF:N

Anomalies #025 and #026 are related and are in close proximity. They may either represent two separate sources, or edges of a broader source.

#027 F/T:837N-888N; C:850N; T:C; S:(R)-8; D:60M; DF:F/N

This substantial Type 'C' response is most likely to emanate from a broad chargeable zone rather than a single conductor.

#028 F/T:-; C:975N; T:D; S:+4(150%); D:70M; DF:N

This very strong zone of external polarization current within a relatively conductive zone at 975N represents the path of discharge of the substantial Type 'C' response to the south and Type 'A' reponse to the north (probably related to #013).

#029 F/T:-; C:1050N; T:A; S:(R)-9; D:60M; DF:N/S

This response, although classified as a Type 'A', could be interpreted as a slightly conductive chargeable source on a contact between a conductive (to the south) and a resistive (to the north) unit. This is inferred by a slight shoulder in the H_N profile.

LINE 700W SOUTHERN SECTION(500S-075N)

While this section of the line relates well to those surveyed to the east, there is in fact very little similarity to lines 800W and 900W. Thus, a discontinuity is considered to extend between lines 700W and 800W over the section surveyed.

#030 F/T:-; C:425S; T:D; S:+5(170%); D:50M; DF:N

This first significant feature is a strong external current flow situated over a strong H_N response. The source is a strong conductor of sulphides, graphite or perhaps electrolyte.

(031/032)

#031 F/T:350S-400S; C:375S; T:C; S:(R)-8; D:50M; DF:N

#032 F/T:-; C:325S; T:B/C; S:(R)-15; D:60M; DF:N/S

Anomalies #031 and #032 are related and form a strong zone of internal polarization responses. This discharges through the differentially more conductive zones either side (#030 and #033) to form strong Type 'D' responses.

#033 F/T:-; C:260S; T:D; S:+7+; D:50M?; DF:N

This strong Type 'D' response is the zone through which the Type 'B/C' response to the south decays. The H_N recorded here is 160% of normal as against 85% to the south.

#034 F/T:-; C:175S; T:D; S:+4+(150%); D:?; DF:N

This very strong conductor of 150% of normal (as against less than 50% to the north) coincides with a strong external polarization current.

#035 F/T:-; C:070S; T:B; S:-3; D:45M; DF:N

#036 F/T:-; C:012N; T:B; S:-3½; D:40M; DF:N

The above two responses are definite and arise from chargeable material contained within a host showing little to no contrast with the enclosing material.

LINE 700W NORTHERN SECTION

#037 F/T:888N-950N; C:920N; T:C/B; S:(R)-5; D:60M; DF:N

This substantial internal polarization response rises over 5 milligammas/gamma above strong external polarization current flows, which reach +5 milligammas/gamma at 850N. The H_N reaches a peak value of just under 110% at 900N, as against less than 85% to north and south of this anomaly. Thus, the host to the chargeable material shows a variation of conduction within it.

#038 F/T:1000N-1050N; C:1037N; T:C?; S:-2; D:50M?; DF:S

Unfortunately, this internal polarization anomaly showing a slow decay form (rare in this area) occurs on the edge of the array.

#039 F/T:-; C:1100N; T:A; S:(R)-4; D:50M; DF:N

This source has a more resistive host to the chargeable source material than the enclosing material, and may in fact be related to #038, and probably represents a variation within it.

#040 F/T:-; C:1175N; T:B; S:(R)-3; D:25M; DF:S

This very minor response may be worthy of consideration. The source is coarse grained within a host having no contrast with the enclosing material.

LINE 600W SOUTHERN SECTION

#041 F/T:-; C:475S; T:D; S:-8; D:75M; DF:N

This very strong external polarization response is associated with H_N values of 120% as against 70% to both the north and south. The centre of the H_N response appears to be 462S. This response is clearly associated with #030 on line 700W.

#042 F/T:325S-425S; C:375S; T:A; S:-17; D:75M; DF:S

This substantial Type 'A' response is probably a broad response and is either due to a series of *separate* sources *or* a *broad source*. This response is clearly related to #031 (Type 'C') and #032 on line 700W. The source is disseminated chargeable material of probably normal grain size within a host which is resistive with respect to the enclosing material.

#043 F/T:-; C:275S; T:D; S:+4(200%); D:50M?; DF:N

This Type 'D' response represents the external polarization discharge of the Type 'A' response seen to the south (#042) through the highly conductive 200% H_N anomaly at 275S. This anomaly very clearly correlates with the Type 'D' #033 on line 700W. The source of this *conduction* may be either electrical (interconnected sulphides or graphite) or *perhaps* electrolytic (salts).

#044 F/T:-; C:175S; T:D; S:+5(150%); D:50M; DF:N/S

This is a similar response to #043, and clearly correlates with $034 on line 700W.

Between #043 and #044 a *relatively* external polarization response was recorded at 225S coincident with much less relative conductivity (120% versus 200% south and 150% north). This feature coincides with a "shoulder" effect at about 212S (#033A) on line 700W.

#045 F/T:-; C:100S; T:B; S:(R)-7; D:100M; DF:S

Two substantial *relative* internal responses show a *slow* decay form which is rare in the area as a whole. This indicates the source material to be coarse grained. The anomaly correlates with #034A (at 150S) on line 700W. A shoulder (#045A) at 050S on this line is equivalent to anomaly #035 on line 700W.

#046 F/T:037N-100N; C:050N; T:B; S:-7; D:60M; DF:N/S

This substantial internal polarization response can be correlated

with a much lesser response (#036) on line 700W. On this line it has a somewhat slower than normal decay form. The source is disseminated chargeable material within a host which shows little to no contrast with the enclosing rocks.

LINE 600W NORTHERN SECTION

#047 F/T:-; C:750N; T:C/B; S:-3; D:25/50M; DF:N/S

This sharp Type 'C/B' response occurs on a *change* in H_N and shows a slower than normal decay form. This anomaly cannot be traced to the west on line 700W.

#048 & #049 F/T:937N-1050N; C:962N & 1025N?; T:B; S:(R)-7; D:50/60M; DF:S/F

These two Type 'B' responses indicate disseminated sources and correlate with anomalies #037 and #038 on line 700W. On both lines the array overlap makes direct comparison difficult.

#050 F/T:-; C:1120N; T:A; S:-6(RELATIVE); D:50M; DF:S

This sharp induced polarization response comes from chargeable material within a resistive host which has two conductive sources at 1075N and 1175N, which emphasises this response. This anomaly correlates with #039 on line 700W.

LINE 500W

#051(#051A) F/T:550S-(662S); C:600S(650S); T:A; S:(R)-6; D:50M+(?); DF:S

This major Type 'A' response is open to the west. The source is disseminated chargeable material within a host more resistive than the enclosing rocks. A relatively minor shoulder was noted at 650S.

#052 F/T:-; C:512S; T:D/C?; S:+9; D:50-75M?; DF:S
The substantial external polarization anomaly is due to the decay of the internal chargeability of the very substantantial Type 'A' anomaly (#053) to the north, and the less substantial Type 'A' anomaly (#051) to the south. This response almost certainly correlates with the strong 'D' type anomaly on lines 600W at 475S.

#053 F/T:425S-325S; C:425S, 375S, 337S; T:A; S:-8;
D:50/60M?; DF:S, S, F
A broad substantial internal response from either a single broad source or a number of individual sources which cannot be resolved. This zone is the correlative of the major response #042 on line 600W.

#054 F/T:-; C:290S; T:D; S:+4; D:60M; DF:F/N
This Type 'D' response is not as well formed as that seen on line 600W at 290S to which it correlates.

#054A C:225S
A relative negative (internal polarization) section *between* the Type 'D' responses #054 and #055 centred about 225S correlates with

a more pronounced feature of similar form (#043A) on line 600W.

#055 F/T:-; C:175S; T:D; S:+5; D:60M?; DF:S

The Type 'D' response correlates with that seen at 175S on line 600W (#044).

#056 F/T:-; C:125S?; T:A?; S:(R)-4; D:?; DF:F/N

This anomaly *may* represent an aberation between arrays, however, it may correlate with the substantial 'B' type anomaly on line 600W (#045). Therefore it may represent the *"lensing out"* of this significant feature.

#057 F/T:012S-150N; C:00, 050N, 112N; T:A; S:-5, -7, -7; D:50-70M; DF:N/S

This broad significant internal polarization response has at least three distinct centres and this may represent multiple sources rather than a broad source. The most southerly of the responses correlates well with #046 on line 600W, but the lake precluded further surveying to the north on that line.

LINE 400W

#058 F/T:-; C:700S; T:D?; S:+3; D:50-60M?; DF:N

This external polarization maximum *may* be a Type 'D' response as it is associated with H_N values of 135% as against background values to both north and south of 90%.

#059 F/T:-; C:625S; T:B; S: RELATIVE -5; D:50M+; DF:N

This source is considered to be disseminated in nature and to correlate with #051 on line 500W.

#060 F/T:-; C:538S; T:C/D; S:+7; D:?; DF:S

This anomaly is a major external polarization axis which in part owes its existence to the Type 'B' response to the south and the substantial Type 'A' to the north. The H_N on this line and on the correlative portion of line 500W cannot *of themselves* explain the size of the external polarization on this line as they, normally if chargeable, would be expected to produce Type 'C' responses. It is suggested that this significant feature is wholly due to discharge of the major disseminated type source on either side, and is not due to "massive sulphide or graphite conduction" as the H_N shows it only to be *more conductive, NOT* very conductive with respect to the enclosing rocks.

#061/#062 F/T:450S-325S; C:350S & 425S; T:A & A; S:-6 & -6; D:50-60M; DF:N/S

These substantial internal polarization responses are due to disseminated chargeable material within a host which is more resistive than the enclosing material. Two distinct sources (350S and 425S) were recorded. This zone correlates with #053 on line 500W.

#063 F/T:-; C:288S; T:D; S:+6; D:50M?; DF:N

A substantial rise in H_N at 288S enhances a strong external polarization flow from #062. This response correlates with a Type 'D' response of similar magnitude, #054, on line 500W.

#064

A relative increase in H_N at about 125S probably correlates with the Type 'D' response recorded on line 500W at 175S (#056), but this is by no means sure.

#065 F/T:050S-050N; C:00; T:A; S:-6; D:60-70M?; DF:N

This broad internal (negative) polarization response is considered to be due to a series of disseminated chargeable material sources within a relatively resistive host. This anomaly correlates well with the southern section of #057 on line 500W.

#066 F/T:-; C:175N; T:A; S:-10; D:50M?; DF:S

This substantial chargeable source within a relatively resistive host may represent a continuation of the unit seen to the south (#065) as it has the same characteristics.

LINE 300W

#067 F/T:OPEN-1100S; C:1150S; T:C; S:(R)-8; D:75M?; DF:N/S

This substantial internal polarization response is associated with an increase in H_N to 115% against 90% to the south and north. The source is thus contained within a medium which is relatively less resistive than the enclosing rocks. The most *conductive*

section also shows the *slowest* decay forms which infers a coarse grain size to the causative material, or perhaps some interconnection between grains.

#068 F/T:-; C:975S; T:D; S:+5; D:25M; DF:S

This sharp increase in H_N to 140% of background is accompanied by a strong external polarization current flow. The maximum depth appears shallow at 25 metres.

#069 F/T:900S-775S; C:875S (CHARGEABILITY) 825S (CONDUCTIVITY) 800S (CHARGEABILITY); T:C; S:-7 & -3; D:50M; DF:N/S

The most interesting *series* of features between 900S and 775S consist of a H_N peak of 170% at 825S with a major internal polarization maximum of 7½ milligamma/gamma at 875S to the south and a relatively minor peak of -2 milligamma/gamma at 800S

#070 F/T:700S-650S; C:662S & 688S; T:A; S:-3; D:30M; DF:S/N

Two *relatively* minor anomalies from a coarse grained sulphide source within a relatively resistive host. The anomalies *probably* correlate with #059 on line 400W.

#071 F/T:-; C:600S; T:D?; S:?; D:?; DF:S

An H_N maximum of 120% at 600S is considered to represent the end effect or easterly continuation of #060 seen on line 400W.

#072 & #073 F/T:475S-312S; C:450S, 325S; T:A,A/B; S:-3, -2
D:50M & 30M; DF:N

These two associated chargeable sources correlate with similar anomalies, #061 and #062, on line 400W, but are of lesser magnitude.

#074 F/T:-; C:ABOUT 275S; T:D; S:+7(150%); D:50/70M; DF:N

This strong external polarization axis is coincident with an increase in H_N to 150% against backgrounds of 80% or less to the north and south. The response correlates with #063 on line 400W.

#075 F/T:-; C:175S; T:A; S:-6(RELATIVE -12); D:75M; DF:N/F

The substantial -12 milligamma/gamma internal polarization anomaly has a source of "normal" grain size hosted within a medium which is *more resistive* than the enclosing rocks. This feature is contained between two conductive axes on this line (#074 and #076) and while it can be traced west between the westerly extensions of these conductors, this is the first line where it forms a major internal polarization feature on its own merit.

#076 F/T:-; C:ABOUT 112S; S:?; T:D; D:?; DF:S

The overlap in arrays obscures this feature. However, a strong external polarization feature is inferred centred at about 112S(±). This correlates with a similar feature at about 125S on line 400W.

#077 & #078 F/T:075S-050N; C:050S & 020N; T:B,B; S:-4,-6;
D:50/60M(BOTH); DF:N, S

Two major and distinct anomalies from disseminated material within a host showing little or no contrast with the enclosing material were located. These anomalies correlate with zone #065 on line 400W.

#079 & #080 F/T:100N-212N; C:125N & 188N; T:B, A; S:-7, -6
D:50-65M; DF:S/N, N/S

These two anomalies are similar in form and magnitude to #077 and #078 described above, and are the equivalent to #066 on line 400W. The decay form on #079 infers a coarser than normal grain size, while both are inferred to have a disseminated source.

LINE 200W

#081 F/T:OPEN-1100S; C:?; T:A; S:-4; D:?; DF:S/N

A coarse grained source from within a resistive host is inferred from sources at or south of 1175S and at 1125S.

#082 F/T:925S-812S; C:850 (H_N); T:C; S:-8(RELATIVE)
D:50-65M; DF:N/S

This series of responses showing an H_N peak at 850S and an internal (negative) polarization peak of 8 milligamma/gamma (at 900S) and of 2-3 milligamma/gamma (at 825S) either side of it, is identical to that seen on line 300W centred at 825S (#069). Thus this unusual feature occurs across both lines. The interpretation for #069 and #082 is that the greatest dissemination of chargeable material occurs *either side of* and particularly *to the south of* the conductor

axis. The uniformity of the decay form suggests that the chargeable material itself *does not* cause the conduction, but rather resides either side of a conductive geologic unit.

#083 F/T:-; C:688S; T:A; S:(R)-6; D:65M; DF:S/N

This substantial internal polarization anomaly is associated with lower than background H_N values. Thus the disseminated material which causes the anomaly is contained within a relatively resistive host. The zone is equivalent to #070 observed on line 300W.

#084 F/T:-; C:588S; T:D; S:+4; D:50M; DF:N/S

A substantial increase in H_N to 140% from 80% background is accompanied by a substantial increase in external polarization also. A much smaller H_N and an external polarization feature was noted on line 300W at 600S.

#085 F/T:-; C:512S; T:A; S:-12; D:60M; DF:F

This major -12 milligamma/gamma internal polarization response reaches some 16 milligamma/gamma above the external polarization response. This response has no correlative on line 300W, although #072 may represent an enechelon response.

#086 F/T:-; C:275S; T:D; S:+6; D:60-70M; DF:N

A sharp increase in H_N to 140% from 80% (to the north) has resulted in a strong external polarization response. This zone is considered to be the correlative of #074 on line 300W.

#087 & #088 F/T:210S-100S; C:175S, 125S; T:A, A; S:-15, -9; D:50M, 35M; DF:N/S, N/S

These two sharp and substantial internal polarization responses come from discrete sources contained within a host resistive to the enclosing material. *Together* they correlate with zone #075 on line 300W.

#089 F/T:-; C:088S; T:D; S:+7; D:40M; DF:N/S

A substantial rise in the H_N from a normal background (100%) to 175% of normal has resulted in a strong external current axis at this point also. #076 is the correlative on line 300W.

#090 F/T:-; C:020S; T:A; S:-5; D:50-60M; DF:N

This significant Type 'A' response is considered to be due to disseminated chargeable material contained within a resistive host. To the immediate north and south, 'D' type responses over relatively conductive zones (160%+) at 100S and 037S (#089 and #091), are due, in part at least, to the decay of the internal polarization response through them. The correlative on line 300W may be *either* #077 *or* #078.

#091 F/T:-; C:037N; T:D?; S:+2; D:?; DF:N

This weak external polarization response is coincident with a strong 160% H_N response. This *external* flow must in part be due to the decay of the strong internal Type 'A' anomalies #090 and #092 either side.

#092 F/T:-; C:100N; T:A; S:-16; D:60M; DF:S/N

From within a *relatively* resistive host (100%), an extremely strong internal polarization response was recorded. This is considered to correlate to a similar 'B' type response on line 300W centred at 125S (#079). Both show slower than normal decay forms.

#093 F/T:162N-225N; C:?; T:C?; S:+3; D:?; DF:N

This relatively small feature appears to correlate with a much more substantial Type 'A' response (#080) on line 300W.

LINE 100W

COMMENT: A significant reduction in the amplitudes and number of anomalies was recorded on this line south of about 300S compared with lines 200W and to the west of that line. A common feature on this line and lines 00 and 200W was the broad zone of external polarization recorded from about 100 metres either side of 300S on all three lines.

#094 & #095 F/T:SOUTH OF 1275S-1170S; C:1275S?, 1188S;
T:A, A; S:-5, -5; D:?, 35M; DF:N/S, N/S

These two disseminated chargeable sources from within a relatively resistive host correlate with similar but smaller anomalies (#081) on line 200W.

It is significant that the substantial 'C' type anomalies at 900S and 825S are not seen on this line.

#096 CENTRE BETWEEN 675S AND 725S; T:C; S:-3, D:50M; DF:N

This moderate sized Type 'C' response occurs on the overlap of two arrays and so the centre is difficult to fix precisely. The source is considered to be a "broad" source or two single sources some 25 metres apart. The southern section appears relatively resistive with respect to the enclosing rocks, while the northern section appears relatively conductive. The correlative on line 200W is considered to be the more substantial 'A' type anomaly (#083) at 688S.

#097 F/T:525S-475S; C:?; T:A; S:-3; D:40M; DF:N/S

Again on an overlap between two arrays and therefore difficult to precisely fix, this internal response of -3 milligamma/gamma correlates with a very substantial -17 milligamma/gamma response (#085) on line 200W.

#098 F/T:?; C:275S; T:D; S:-6(160%); D:?; DF:N/F

Within a broad zone of external polarization between about 200S and 450S, a distinct H_N maximum of 160% was observed at 275S, against a 40% level at 150S. This conductor is certainly of some geological interest. The strong coincidental external polarization is certainly in part due to the strong internal polarization anomaly, #099, described below.

#099 F/T:175S-088S; C:150S; S:RELATIVE -8/-10; D:70M?; DF:N

This strong 'A' type response of about -4 milligamma/gamma

rises some 6 milligamma/gamma above strong *external* flows to both north and south. This response *may* have two sources, one at 150S and a second at about 112S. These two *possible* sources may correlate with anomalies #087 and #088 on line 200W.

#100 F/T:- C:075S; T:D; S:+5; D:?; DF:S/N

A small narrow increase in H_N to 130% above sub-normal levels of 50% and 80% to south and north is responsible for a coincidental *external* current flow axis.

#101 & #102 F/T:050S-050N; C:025S, 025N; T:A, B; S:-8, -5 D:30M; DF:N/S

Two narrow disseminated chargeable sources within a relatively resistive host are considered to correlate with a more significant single 'A' type anomaly at 025S on line 200W (#090).

#103 F/T:088N-250N; C:?; T:A; S:-10; D:50M?; DF:-

A broad zone of high internal polarization was noted from within a relatively resistive host, with *possible* maxima at 125N, 175N and perhaps 225N also. This source is considered to be disseminated chargeable material within a resistive host. This zone is considered to correlate with two distinct anomalies #092 and #093 on line 200W which have a similar form.

#104 F/T:- C:350N; T:D?; S:+2; D:?; DF:S/N

An external polarization axis coincident with a 140% H_N response

at 325N which correlates with a similar external polarization axis *but without* coincidental H_N on line 200W at 275N. However, a north-west/south-east dislocation *may* be present between the two.

LINE 00

#105 F/T:OPEN-1190S; C:1200S, 1250S, 1300S; T:A; S:-5
D:30M?; DF:N

A broad Type 'A' response was recorded, which has a series of individual peaks within the zone. The host to the chargeable material is very much more resistive (70% of normal) as against over 140% to the immediate north. This zone is equivalent to #094 and #095 on line 100W.

#106 F/T:700S-775S; C:775S; T:A; S:(R)-5; D:30M?; DF:S/N

This Type 'A' response of chargeable material within a resistive host, may correlate with a Type 'C' response (#096) on line 100W.

#107 F/T:625S-500S; C:612S, 550S, 500S; T:A; S:(R)-3;
D:?; DF:N

A broad zone of disseminated chargeability coincides with lower than normal (to 70%-80%) H_N values, indicating a resistive host. The northern section (500S and 550S?) correlates with a similar response (#097) on line 100W.

#108 F/T:200S-480S; C:?; S:-; D:-; DF:-

This broad zone of external polarization correlates with a similar

feature on the lines to the west (#098 and #086). An H_N maxima is *inferred* between 200S and 300S which also correlates with a more definite feature on the previous line at 275S.

#109 F/T:200S-100S; C:100S, 150S; T:A; S:(R) -8; D:40-60M?; DF:F/N, S/N

A substantial external polarization response having two distinct peaks, was recorded from a relatively highly resistive host. This feature clearly correlates with the two peaks of an almost identical response (#099) on line 100W.

#110 F/T:-; C:055S; T:D?; S:+7(140%); D:50M; DF:N

A sharp increase in H_N indicating a relatively conductive, narrow source, coincides with a very strong external polarization axis, which clearly correlates with #100 on line 100W.

#111 F/T:050S-075N; C:?; T:A; S:-14; D:50M; DF:N/S

This anomaly is interpreted as having a disseminated source either over a broad zone or disseminated sections within the broad zone. This zone apparently correlates with two minor zones (#101 and #102) on line 100W, which may represent the "fish tail" extensions of this zone.

#112 F/T:100N-275N; C:?; S:-12; T:B; D:60-75M; DF:N/S

This anomaly is similar in form and magnitude to that observed on line 100W for anomaly #103 to which it correlates. However,

there is a possible north-west/south-east dislocation between the two anomalies. The source is again considered to be disseminated chargeable material within a host which shows no conductive contrast with the enclosing rocks.

#113 F/T:-; C:337N; T:D?; S:+3½(130%); D:40M; DF:N/F

An increase in H_N to 130% of normal against a 100% background has caused an external positive polarization axis. The source to a large degree, is due to the substantial internal polarization anomaly logged to the immediate south. The correlative on line 100W is #104.

#114 F/T:-; C:375N; T:?; S:?; D:?; DF:S/N

A small *relative* internal polarization response at 375N correlates with a smaller response at 375N on line 100W.

#115: F/T:430N-550N; C:465N(525N); T:B; S:-7,(-4); D:50M; DF:N/S

The substantial internal polarization high is consdered to be due to a major disseminated source centred at 462N, with a *possible* secondary centre at about 525N. There is a slight depression in H_N in both cases, indicating the source to be slightly more resistive than the enclosing rock units.

#116 F/T:-; C:612N; T:B; S:-5½; D:50M; DF:N

A strong internal polarization response of 5½ milligamma/gamma

was recorded which is interpreted as emanating from material somewhat more resistive than the enclosing rocks. This anomaly decays via a moderate increase in H_N (#117) to the north.

#117 F/T:-; C:725N; T:D?; S:+7; D:?; DF:N

A strong external polarization response of +7 milligamma/gamma was recorded over a 30% above normal H_N. This is primarily caused by the strong decay of the Type 'B/A' response recorded to the immediate south (#116).

LINE 100E

#118 F/T:-; C:450S; T:B/C; S:-4; D:?; DF:N/S

This Type 'B/C' response centred at 450S probably correlates with #129 on line 200E.

#119 F/T:200S-350S; C:-; T:-; S:-; D:-; DF:-

This zone of strong external polarization is considered to be the stratigraphic equivalent of the wider zone (200S-475S) on line 00. This could perhaps imply "shortening" in the profile width over this section, perhaps by faulting.

#120 F/T:088S-200S; C:-; T:A/C; S:(R)-3½) D:60M; DF:S

This most interesting feature is a *relatively* internal response of 3½ milligamma/gamma superimposed on a strong external polarization. This feature is considered to be the equivalent on this line of a substantial Type 'A' anomaly recorded on line 00 between 200S and 088S.

#121 F/T:-; C:062S; T:D; S:+8(140%); D:?; DF:N

A strong H_N anomaly of 140% (against background of less than 50%) has given rise to a coincident strong external polarization response. This feature is almost certainly the extension of #110 at 075S on line 00.

#122 F/T:-; C:00; T:A?; S:-16; D:50M; DF:N

This response is situated on an array overlap so its relative merits are difficult to assess. It appears to represent a narrow chargeable source situated on, or in close proximity to a rock type change as seen by the rapidly changing H_N. The maximum depth appears to be of the order of 50 metres. It appears to correlate with anomaly #111 on line 00, which, on that line, is a broad zone possibly composed of multiple sources. Thus the source on this line *may* be one of those individual sources.

#123 F/T:200N-300N; C:225N, 375N?; T:B?/C; S:?; D:?; DF:N/S

Again, array overlap appears to grossly distort the response from a source situated between 200N and 300N. However, it is considered a southerly extension of the more substantial (in both width and magnitude) anomaly #112 recorded on line 00.

#124 F/T:350N-525N; C:375N, 425N; T:B/A; S:-6; D:50M; DF:N

This substantial internal chargeability anomaly appears to have two main sources at 375N and at 425N which are considered to be due to disseminated chargeable material within a relatively resistive

host. These two maxima are considered the correlatives of two similar responses at 460N and 525N on line 00 (#115).

#125 F/T:-; C:600N; T:A; S:-4; D:50-75M; DF:N

This disseminated chargeable source within a resistive host is considered the equivalent of a similar response (#116) at 612N on line 00.

#126 F/T:-; C:700N; T:D?; S:+4; D:?; DF:N

A strong external polarization axis of +4 milligamma/gamma situated at 700N was not seen to coincide with *any* material increase in H_N. This feature, however, is clearly the equivalent of #117 on line 00.

LINE 150E

#127 F/T:012S-088N; C:012N, 075N; T:A; S:(R)-7; D:?; DF:N

This anomaly occurs on the overlap between two arrays and infers two chargeable maxima at 012N and 075N. This response is the eastern extension of anomaly #122 on line 100E.

#128 F/T:225N-312N; C:288N; T:B; S:-12; D:50M?; DF:S

This strong internal polarization response is considered the eastern correlative of #123 on line 100E.

LINE 200E

#129 F/T:OPEN-400S; C:450S; T:B; S:-4; D:75M; DF:N

This strong broad internal induced polarization anomaly is interpreted as emanating from a disseminated source at a maximum depth of 50 to 75 metres. This anomaly is the equivalent of anomaly #118 on line 100E.

#130 F/T:00-400S; C:-; T:-; S:-; D:-; DF:-

A broad zone of strong external (positive) polarization was again noted over a wide zone. This is the southern correlative of the broad zone of external polarization over the same co-ordinates on line 100E, but without the anomalies #120 and #121.

#131 F/T:-; C:100N; T:A/B; S:(R)-3; D:25M; DF:N/S

This small single station *relative* internal polarization response *may* be the southern correlative of a small Type 'A/B' response on line 100E at 100N and #127 on line 150E.

#132 F/T:-; C:175N; T:B; S:(R)-3½; D:30M; DF:N/F

This is a similar anomaly to #131 but may correlate with a "shoulder" observed on line 150E at 175N.

#133 F/T:-; C:300N; S:-6; T:B/A; D:40M; DF:N

This sharp apparently narrow source is considered to be disseminated, and to correlate with the much more substantial anomaly located on line 100E at 275N.

#134 F/T:362N-475N; C:425N; T:A?; S:-18; D:75M?; DF:N/S

This substantial internal polarization response is interpreted as

as originating in disseminated chargeable material within a host which is more resistve than the rocks to the south, *but* more conductive than the rocks to the north. This zone is clearly the eastern correlative of #124 on line 100E.

#135 F/T:512N-588N; C:525N, 575N; T:A; S:-4½, -5½; D:30M? DF:N

An increase in internal polarization response was recorded with two distinct centres. As the normalised horizontal field shows a marked decline, the source is considered chargeable material within a relatively resistive unit. This zone correlates with a similar single maxima response on line 100E at 600N (#125)

#136 F/T:625N-OPEN; C:-; T:-; S:-; D:-; DF:-

North of #135 strong external polarization was recorded with little change *from normal in* background H_N. This external polarization is most likely due to the chargeable material to the south.

LINE 300E

This line is of very similar form to that observed on 200E and lines to the west, but markedly different to lines observed to the east. A major change in geological units is suggested between 300E and 400E.

#137 F/T:OPEN/425S; C:500S?, 437S; T:C/B; S:-3; D:40M?; DF:N/S

This significant internal polarization response correlates with similar responses to the west including #129 on line 200E, but has no clear correlative to the east.

#138

A broad zone of external polarization was noted between 400S and 00 which correlates with similar responses to the west, but again is not seen to the south.

#139, #140, #141 F/T:050N-275N; C:050N(±), 175N, 250N; T:B, B, C; S:-2; D:30-50M?; DF:S/N

These three small responses *may* correlate with more substantial anomalies observed on line 200E (#131, #132 and #133).

#142 F/T:325N-450N; C:400N; T:A TO C; D:75M; DF:S

This response is interpreted to come from disseminated chargeable material within a host more resistive than the rocks to the south but more conductive than the rocks to the north. These are precisely the same characteristics as noted on #134 on line 200E. There is absolutely no sign of this response on line 400E to the east.

#143: F/T:-; C:550N; T:A; S:-6: D:50M; DF:S

This source is interpreted as coming from a "narrow" source 50 metres deep. The host is resistive relative to #142 to the south. The zone may correlate with one of the peaks on #135.

#144

As with lines to the west (but not to the east) strong external polarization was recorded south of 625N.

LINE 400E

This line shows a profile form materially different to that seen to the west, however this line *is* similar in form to lines to the east. Thus it is considered that a major stratigraphic change takes place *between* lines 300E and 400E.

#145 F/T:-; C:500S; T:B; S:-3; D:?; DF:N/S

A relatively low amplitude internal polarization response centred at or south of 500S may correlate with a similar, more substantial, feature on line 500E.

#146 F/T:-; C:350S; T:B; S:-2; D:50M; DF:N

A small -2 milligamma/gamma internal polarization response which shows little change in H_N is correlated with a similar feature on line 500E at 375S, but has no northern correlative.

#147 F/T:100N-550N; C:-; T:-; S:-; D:-; DF:-

A strong broad zone of external polarization was noted between 100N and 550N which can be traced east but not west. This is similar in size and form to that recorded between 00 and 400S on line 300E, and to the west of that line.

#148 F/T:-; C:600N; T:A/B; S:-3; D:50M; DF:N

A small type 'A' anomaly infers a disseminated chargeable source at about 600N which does not correlate with any feature seen to the west.

LINE 500E

#149 F/T:OPEN TO SOUTH; C:AT OR SOUTH OF 500S; T:B; S:-4; D:?; DF:S/N

This end-of-line anomaly correlates with a similar response (#145) on line 400E.

#150 F/T:-; C:375S; T:A; S:-3; D:50M; DF:S

This anomaly is interpreted as coming from a narrow source whose coarse grain sized chargeable material is contained within a resistive medium with respect to the enclosing rocks. It *may* correlate to #146 on line 400E.

#151 F/T:050N-350N; C:-; T:-; S:-; D:-; DF:-

A broad zone of external polarization on this line is equivalent to that seen on line 400E (#147).

#152 F/T:400N-525N; C:450N (512N); T:A(B); S:-7½; D:60-70M; DF:N, S

A major internal polarization response of over 8 milligamma/gamma is associated with a major depression in H_N. The disseminated source is interpreted to be set within a relatively resistive

medium. A more coarsely disseminated source is possible at 512N also. This second source *may?* be the eastern correlative of #148 on line 400E.

LINE 600E

#153 F/T:-; C:412S; T:B; S:-1½; D:40M; DF:N

This minor response may correlate with #150 on line 500E and with a very minor internal high at 450S on line 700E.

#154 F/T:-; C:250S; T:-; S:(R)-2; D:50M; DF:S

A minor response only.

#155 F/T:025N-325N; C:-; T:-; S:-; D:-; DF:-

This broad zone of external polarization corresponds to #151 to the west.

#156 F/T:325N-462N; C:350N, 400N, 450N?; T:A; S:-3 TO -5; D:50-60M; DF:N

This series of internal polarization maxima may be related to #152 on line 500E, but shows only minor possible anomalism to the east.

LINE 700E

#157 F/T:-; C:325N; T:A; S:(R)-3; D:30M; DF:N

This narrow source is considered to correlate with a more substantial source at about 400N on line 500E, which, however, is observed on an array overlap.

INTERPRETATION DIAGRAM

The H_N and chargeability data has been summarised on Plate 3 at the same scales as the contour interpretation of the individual parameters, namely, 1:2500.

The internal polarization responses are shown in thick (for substantial) and fine (for relatively minor) anomalies. Continuous solid lines mark the *probable* extent of each source while broken lines represent the *possible* extent of each individual source.

Major conductor axes are marked by open circles, and while these are often coincident with external polarization axes, they have not been marked as such.

Areas of extensive external polarization have been shown by fine dots.

Each significant internal polarization response has been numbered using the system adopted in the body of this report and on the profiles.

Strike.....The predominant strike noted, varies about grid east west. Individual events can be traced for significant distances east-west over 500 metres for both internal polarization anomalies and narrow relatively conductive zones. The latter are much more clearly correlated between lines, while the former can show very

significant interline variation between lines.

In the extreme north-east of the surveyed area, the strike is distinctly north-east/south-west.

Discontinuities..... Two distinct sets of discontinuities have been inferred from both the H_N and chargeability data. The difficulty in recognising such differences where the whole area is composed of a large number of *individual* current dipoles *within which* the H_N and chargeability data is *relative*, should be recognised. Nevertheless it is the author's opinion that these two sets of discontinuities represent real features.

The *first* and most prominent series, are two approximately grid north-south zones between lines 300E and 400E (X), and between lines 700W and 800W (Y). Now *both* also coincide with current dipole boundaries, which of themselves would be expected to produce variations between lines, however, the differences logged are so marked, that they are not considered to be primarily due to changes caused by array position, This may, however, have assisted in the exaggeration of these changes.

The causes could be flexures, monoclines or faults.

The *second* series are four north-west/south-east trending dislocations which are marked on Plate 3 as Weste's, Askins, R and S. The relative merit appears to be in that order also.

The effect of these dislocations is to break apparent dislocations up into a series of distinct blocks whose strike is grid east-west. It is understood that the apparent approximate grid north-north-west/south-south-east strike in the most north-easterly portion of the grid is likely due to the extension of the Bismuth Creek Fault which also has this trend.

Extensive Areas of External (Positive) Polarization.....These areas have been shown by fine dots on Plate 3. On the mainland in regions such as Cobar, such areas have been noted over areas of *deep* and *intense* oxidation overlying a bedrock which is fresh. The characteristics of the oxidised layer in these regions is *low resistivity* approaching 10 ohm-metres and intrinsic chargeability *approaching zero*, the explanation being that the more chargeable freshrock below decayed *above* in the conductive near surface cover. In this case this analogy *cannot* apply. However, as in both the H_N and M parameters *contrast* in these properties is important, this is the most likely explanation here also. This would imply that the material *under* the nearer surface rocks is *more chargeable* than the overlying units, and also *more resistive*. Now, as limestones are themselves very resistive, this infers the underlying rocks to be very significantly resistive. This is a feature of the area which bears additional consideration.

The Conductor Axes..... These conductor axes form one of the most interesting features observed in the area and extend from

300 metres to over 800 metres. Those two major axes contained between 00 and 300S on lines 100E to 700W, together with the arcuate to the immediate south thereof between line 700W and 400W, with perhaps extensions to lines 200W and 300W at 600S, are the most prominent. To state the obvious, all three represent zones of marked increase of conduction. Now, this means that these zones are much more conductive *relative* to the enclosing rocks, but as the enclosing rocks are resistive limestones(?), it may not necessarily mean that the conduction has to be high in absolute terms. The depths to source, due to their strike length, are difficult to gauge, but maximum depths to source are considered to be of the order of 40 to 80 metres. It is difficult to envisage such conductors being a feature of horizontal limestones *as such*. Therefore, the sources must either lie *beneath* the limestones, be *introduced into*, or *re-mobilised within* the limestones. They are not considered to be purely superficial features.

The strong *external* polarization axes (Type 'D') which often correlate with the H_N peaks are due to the fact that invariably the high *internal* polarization anomalies lie in close proximity to them and therefore the preferential decay route for the energy stored within these disseminated sources is via the conductors. This type of anomaly has been noted for massive pyrrhotite surrounded by disseminated sulphides at more than a dozen separate locations. However, similar anomalies have also been defined over zones which are significantly more conductive within chargeable rock units. A recent example was chlorite schists within resistive

gneiss carrying disseminated sulphides. It is therefore important to realise that the strong external polarization current axes are there *as a result of* the increased *relative* conductivity of the axes which themselves *may or may not* be due to electronic (as opposed to electrolytic) sources.

These features do warrant the most careful further investigation, particularly as pyrrhotite has been identified within these limestones.

The Major Internal Polarization Anomalies..... This survey has produced some of the largest chargeability anomalies recorded to date. Also, the areal extent of these anomalies is the largest observed anywhere to date. For the most part, the source is interpreted as being due to disseminated chargeable material (sulphide or graphite) within a host which either has *no contrast* (Type 'B'), or is *slightly* more or less conductive than the enclosing rocks.

The strike of these anomalies appears to be grid east-west and rather than individual zones being traced from line to line, the form of the anomalies strongly suggests a series of lenses of *limited* strike length within given boundaries. Thus, while the zones *as a whole* have significant strike length, individual members *within it* do not.

The *form* of the profile suggests vertical to steeply dipping sources, while the interline continuity suggests strike lengths often in excess of 500 metres. Both these features are in conflict with the shallow dipping limestones which for the most part underly these anomalies. Thus it is concluded that the source of the chargeability anomalies is either introduced into, remobilised within, or is wholly beneath the limestones.

CONCLUSIONS

1. The nature of the internal polarization anomalies defined in this area is unique, both in size and number.

2. The strike appears to be grid east-west while the *form* suggests narrow sources of limited strike length, but contained *as a zone* over many hundreds of metres.

3. The H_N peaks can be traced for up to 800 metres and as with the chargeability responses, cannot be easily explained as being part of the shallow dipping limestones.

4. It is considered important that the cause of both the internal polarization maxima and more particularly, the H_N maxima, be unambiguously ascertained. It is therefore strongly recommended that should either feature be investigated by diamond or percussion drilling, 2½ metre three array logs for chargeability and resistivity should be run, followed by 10 metre three array

logs to investigate the extent of such zones.

As the data obtained in this survey is unique, this report has necessarily concentrated on the physical properties and their possible geological meaning.

Respectfully submitted on behalf of:

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A.W. HOWLAND-ROSE, MSc, DIC, AMAusIMM, FGS.

GEOPHYSICIST

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THE MAGNETIC INDUCED POLARIZATION METHOD

It is essential to grasp the very basic differences between the magnetic mode of acquiring induced polarization data *(MIP)* and the more conventional electrical mode *(EIP)*. As even geophysicists of some experience have difficulty in appreciating the full significance of the data obtained in the magnetic mode, I have incorporated into this report some remarks and comments on the main anomaly forms obtained from MIP.

The very brief description which follows is also designed to give a visual picture of the relationship between EIP and MIP, and demonstrate how the various MIP parameters of normalised horizontal magnetic field (H_N), chargeability *(M)*, secondary magnetic field (H_S) and decay rate *(ΔM)*, represent the electrical properties of the material over which the tests are made.

Comparison of the Electrical and Magnetic Modes of Acquiring Induced Polarization Data:- By far the most meaningful manner in which to visualise the nature of MIP (and indeed EIP) data, is to consider the *energy storage concept* and to look at the initial current flow patterns and the resultant equipotential field caused by the energising current, and then the consequent secondary current flow pattern and its associated secondary potential field caused by the decay of energy stored in the induced polarization phenomenon.

Energisation Process:- Normally current is applied to the volume to be sampled by means of two electrodes placed semi-parallel to the expected strike of the target mineralisation. In the diagram shown in Figure 1, the fine solid lines represent the current flow pattern so generated. The dashed faint lines represent the equipotential surfaces (lines in the section).

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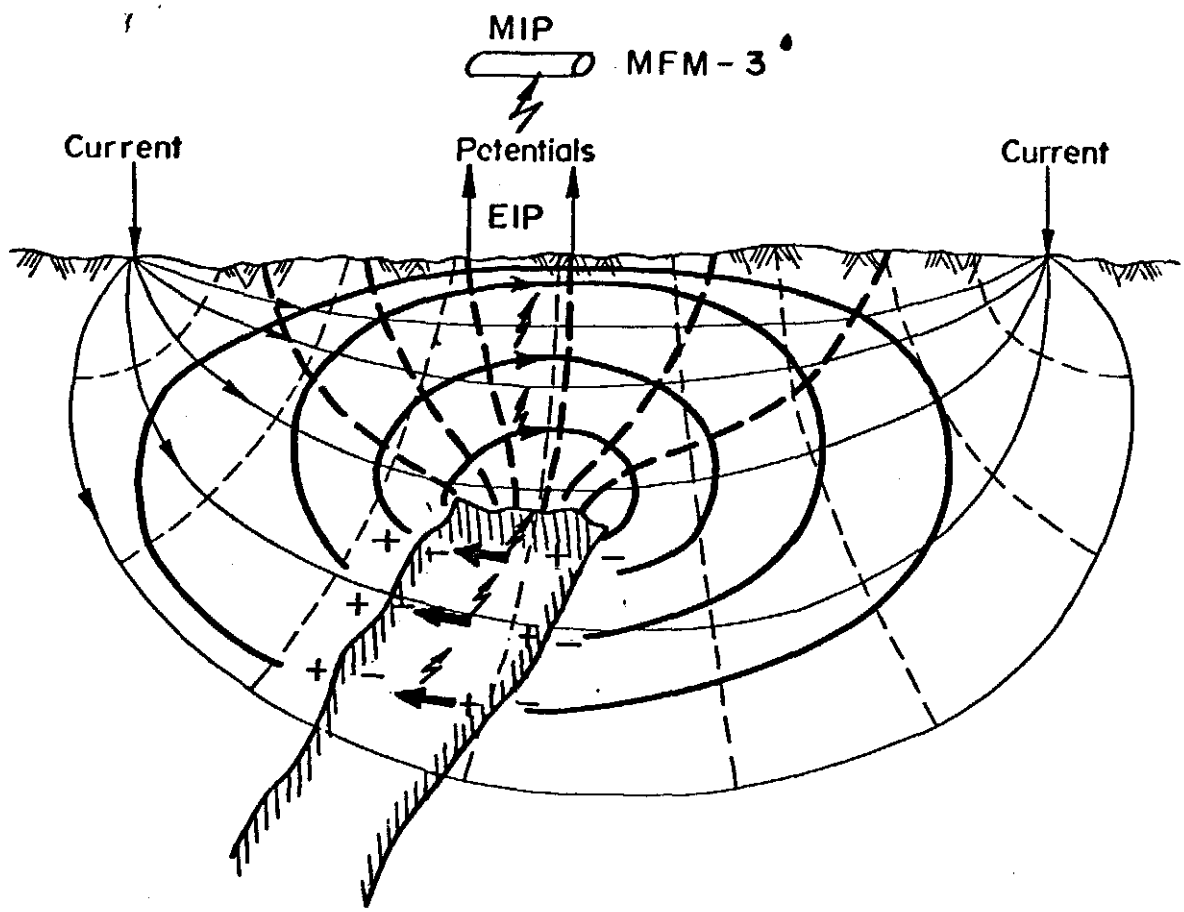
EIP & MIP

ENERGISATION & DISCHARGE PATTERN



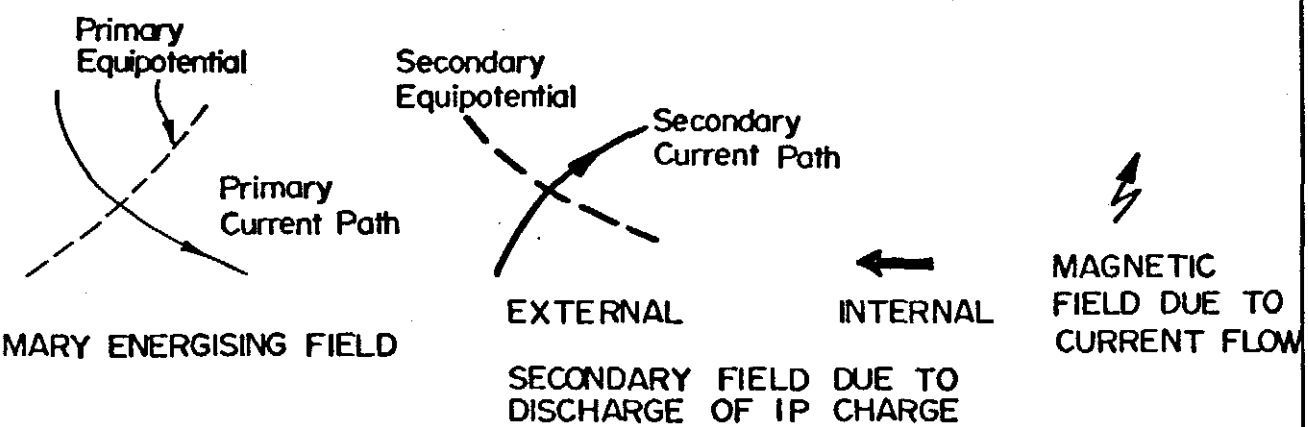


Fig. 1

In the *electrical mode*, the two potential electrodes (See Figure 1) will measure the *resistivity* of a volume of material defined by the equipotential surfaces which are always at right angles to the current flow.

Energy Storage Process:- The material through which the current passes will store some portion of the energy in a way determined by the properties of the storage material. The amount of energy stored will depend on the total area of the sulphides (or graphite etc.) presented to the current, and thus, the greater this surface area with respect to the volume of material, the greater will be the energy stored. Finely disseminated material will store substantially more energy than coarse grained material.

The Discharge of Stored Energy:- On cessation of the energising current flow, the energy stored by the *chargeable source* will discharge *internally* within the source as shown by the solid arrows in Figure 1, and *externally* around the body in the medium surrounding the source as shown by the solid heavy lines in Figure 1. These currents are respectively known as *external* and *internal* current flow. The former is of *positive sign* as it is in the *same direction* as the original energising current, and the latter is *negative* in sign because it is in the *opposite direction* to the energising current.

In the electrical mode the discharge outside the body *ONLY* is investigated. In Figure 1 the thick solid lines show this discharge together with the *equipotential surfaces* (thick broken lines) which this current imposes. As with the charging process these surfaces must be at right angles to the current lines which impose them. The potential electrodes will therefore measure the stored energy (chargeability) as seen via the secondary

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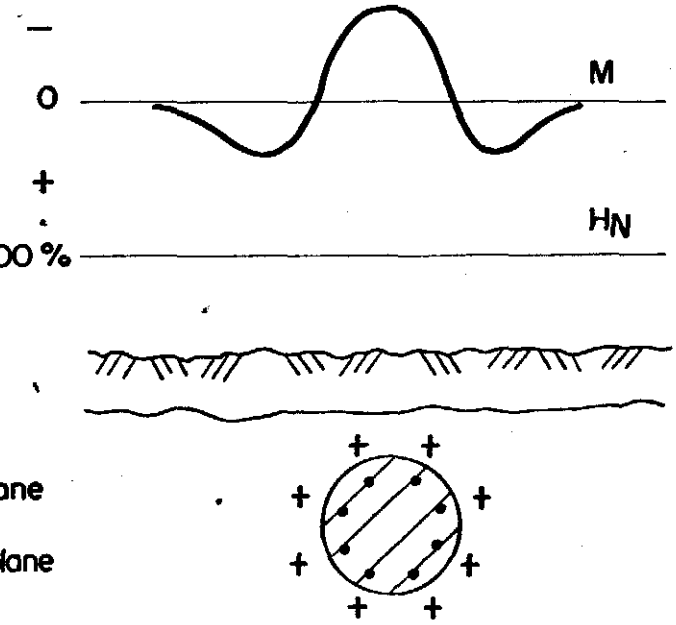
TYPICAL M.I.P. ANOMALY FORMS

THEORETICAL MODEL

CHARGEABLE SOURCE
NO RESISTIVITY CONTRAST



+ External current flow into plane of paper
• Internal current flow out of plane of paper

TYPE A

CHARGEABLE SOURCE
RESISTIVE SOURCE

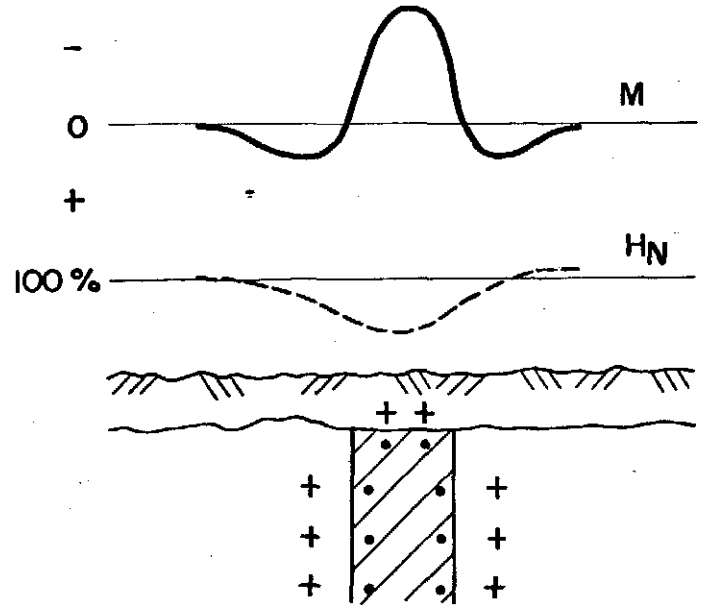


Fig. 2

equipotential field. It is important to note that *(i)* this is *NOT* the same volume as the resistivity measurements and *(ii)* it is *NOT* the original IP signal as stored by the body, but a measurement distorted and processed by the environment through which it passes.

In the *magnetic mode* a very sensitive magnetometer (Scintrex MFM-3) is used to "sense" the current flow both *inside* and *outside* of the *source material*. This occurs because each electron which flows in the ground carries with it an associated magnetic field. This magnetic field will pass *unhindered* through the environment and thus both the discharge *internally* and *externally* to the source can be monitored on the surface.

The Form of MIP Anomalies:- The enclosed Figure 2 demonstrates the theoretical form of an MIP anomaly from a source which has no electrical contrast with the enclosing material, but has the property of retaining charge. (In nature such anomalies are in fact observed over heavy mineral deposits in beach sands)

Energisation is along strike, out of the plane of the paper, by a gradient array. All diagrams represent the current flow into (crosses) and out of (dots) the plane of the paper due to a storage of electrical charge in the source. In Figure 2 over the source, the magnetometer will "see" a surplus of internal (negative) current flow, while on the flanks of the body, the external (positive) current flow will become predominant. The *"head and shoulders"* MIP anomaly shown is *always* seen over all sources. It is the distortions in shape, form and zero level that yield vital information as to conductivity of the source, conductivity of the environment above and about the source, the depth to the source and the nature of the mineralisation in and around the source.

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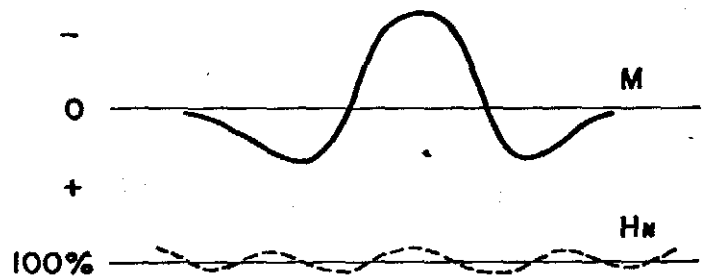
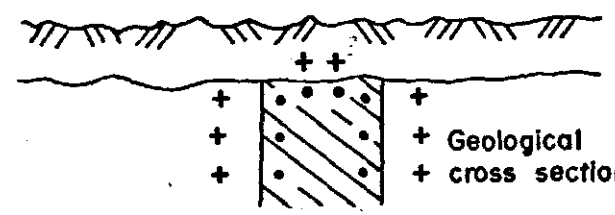
TYPICAL M.I.P. ANOMALY FORMS

TYPE B

CHARGEABLE SOURCE
HOMOGENEOUS



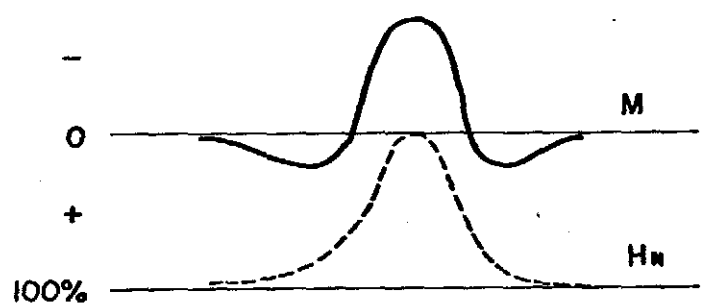


+ External current flow into plane of paper.

• Internal current flow out of plane of paper.

TYPE C

CHARGEABLE SOURCE
CONDUCTIVE



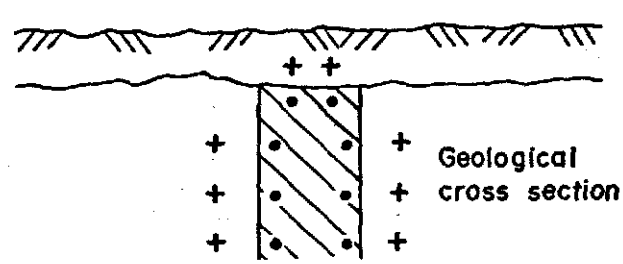


Fig. 3

In MIP the energising field is normalised, so that should the medium through which it passes be homogeneous, the resultant normalised horizontal field (H_N) will be 100% everywhere. Therefore, H_N in this "theoretical model" is everywhere 100%.

TYPE A (FIGURE 2) shows the typical anomaly form over a chargeable source which is more resistive than the surrounding medium. In such cases the normal *"head and shoulders"* anomalies coincident with a depression in the H_N are observed. An example of such an anomaly form is chalcopyrite/pyrite in quartz veins itself within a more resistive conductive rock unit.

TYPE B (FIGURE 3) In this case the chargeable source has no resistive contact with the enclosing material. This example is very similar to the theoretical model. An example of such an anomaly form would be over disseminated sulphides within a homogeneous rock unit.

TYPE C (FIGURE 3) In this case the source of the chargeable material is itself more conductive than the enclosing rock type. When the observed H_N values are *less than* 180% - 200%, a normal *"head and shoulders"* anomaly is observed over the source. In practice, observed H_N values rarely exceed 150% of normal.

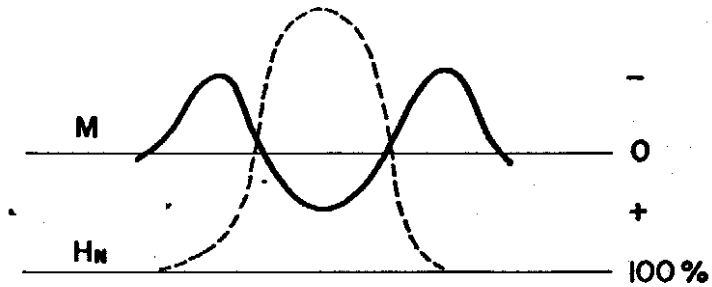
TYPE D (FIGURE 4) In this most important anomaly form which invariably is associated with massive sulphides which are both conductive and electrically continuous, a massive sulphide *must* be surrounded by a disseminated halo within more resistive host rocks. In this case the disseminated sulphides will naturally store the induced polarization charge *far more efficiently* than the massive electrically continuous core. Thus, on completion of the energisation process, the charge stored within the disseminated halo will preferentially discharge through the conductive massive sulphide core. This effect has *NEVER* been

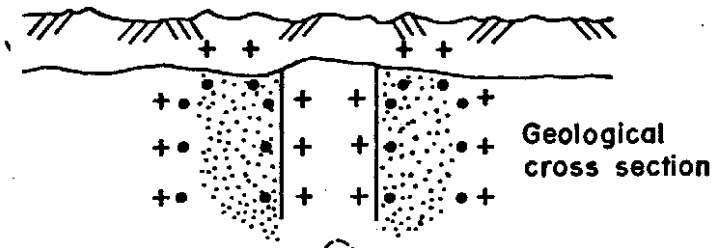
TYPICAL M.I.P. ANOMALY FORMS

TYPE D

CHARGEABLE SOURCE VERY CONDUCTIVE WITH DISSEMINATED HALO

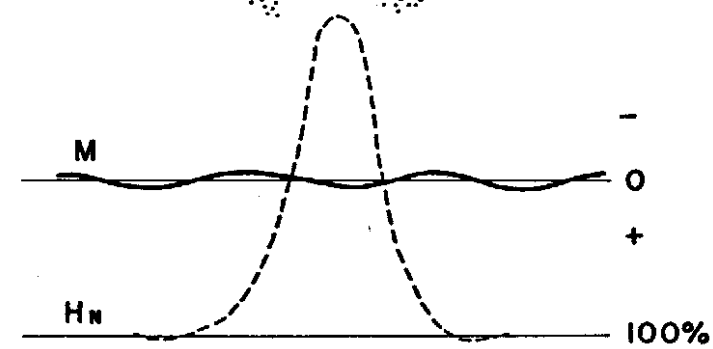
+ External current flow into plane of paper.
• Internal current flow out of plane of paper.





TYPE E

SOURCE VERY CONDUCTIVE BUT WITH LITTLE OR NO APPRECIABLE DISSEMINATED HALO



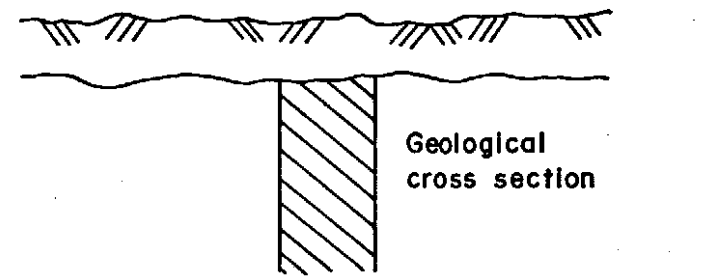


Fig. 4

observed where H_N values have been less than 180% of normal. This anomaly form due to its high H_N and coincident predominantly external (positive) current flow, is diagnostic when observed. An example of such a response is the Mt. Windarra pyrrhotite nickel, copper deposits in Western Australia.

TYPE E (FIGURE 4) Theory has always predicted that massive sulphides should show no IP effect, but invariably the disseminated material surrounding massive zones are sufficient to show the body up clearly. Over a number of Western Australian pyrrhotite nickel deposits, the massive, conductive, electrically continuous sulphides *are not* surrounded by any significant disseminated halo. Thus, little or no material is available for storing the induced polarization charge. However, the substantial normalised magnetic field which is invariably very high indeed (certainly in excess of 200% and as high as 400%) is the diagnostic feature. To date responses of such magnitude have never been observed over electrolytic conductors, even in the world's most conductive environments in Western Australia's salt lakes belt. However, such cases will inevitably be recorded therefore caution is needed in evaluating this rare anomaly type.

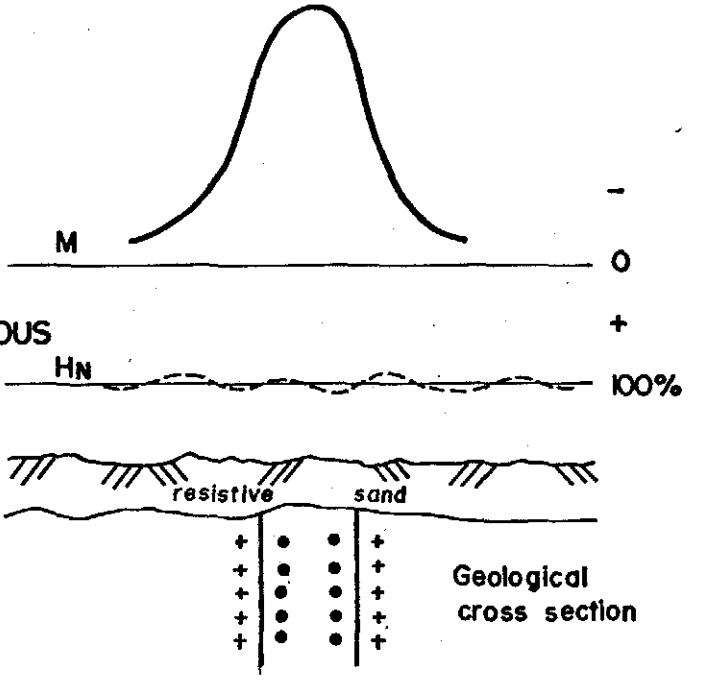
Modification of Anomaly Form by the Environment:- It is important to realise that each of the above anomaly forms is modified by the environment occurring immediately above the chargeable source. Two such cases are shown to illustrate the modification in the form of the chargeability anomaly which occurs. Both the sources show no contrast with the enclosing host material as per Type B.

In Type F (Figure 5) the source is covered by dessicated sands. Thus the return external current *cannot* flow above the body. Thus the external current flows *under* the source and/or around the sides of the source. The observed induced polarization

TYPICAL M.I.P. ANOMALY FORMS

TYPE F

CHARGEABLE SOURCE HOMOGENEOUS
(AS B) UNDER DESSICATED
RESISTIVE SAND COVER



\+ External current flow into plane of paper.

• Internal current flow out of plane of paper.

TYPE G

CHARGEABLE SOURCE HOMOGENEOUS
(AS C.) UNDER HIGHLY
CONDUCTIVE COVER

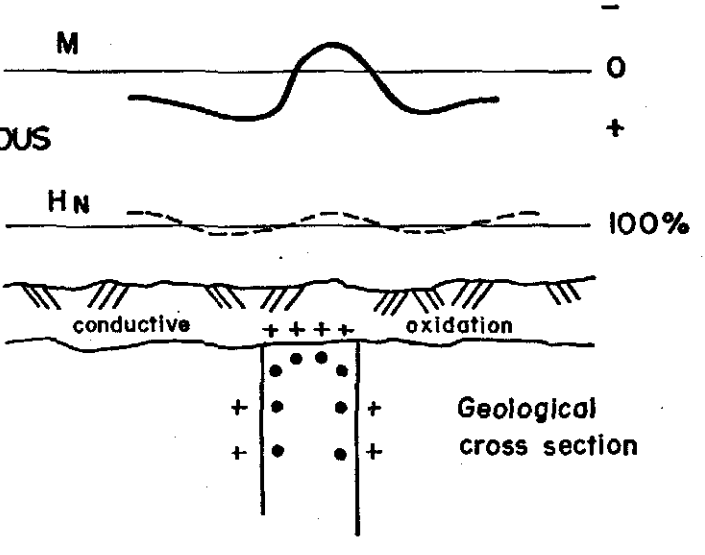


Fig. 5

anomaly observed via the magnetic sensor is invariably larger than normal as it is not reduced by external current flow above the body, and lacks the typical *"head and shoulders" form*. In addition the chargeability is usually entirely *internal* (i.e. negative).

In the case of *Type G* (Figure 5) the material between the source and surface is highly conductive. Some observed examples include relatively shallow, highly conductive salt lakes in Western Australia and, in the deep (100 metres) less conductive (5 to 10 ohm-metres) oxidation occurring in the Cobar area of New South Wales. In this case the characteristic features of the anomaly form are a much reduced *"shoulder"* level and a much reduced anomaly amplitude. The former occurs because of the preferential external current flow in the conductive overburden/ oxidation as a whole, and the latter due to the more intensive external current flow over the body itself.

Secondary Horizontal Magnetic Field (H_S) This parameter is also displayed, and is the magnetic field due to the discharge of the stored energy. Under certain conditions a study of H_S is more meaningful than M and H_N. However, in most cases it mirrors them.

The Significance of Decay Form (ΔM) Since the induced polariztion data was first studied in the laboratory, the variable nature of the decay of the stored IP charge with mineral composition and grain size has been known. However, rarely if ever, is this phenomenon observed in the field. There is in fact a very simple explanation as to why this is in fact so.

In *Figure 1* the energising current is shown in faint dashed lines. The resultant discharge (external and positive), is shown in

bold lines, while the internal negative discharge is seen in similarly bold arrows. The direction of the arrows indicates the direction of current flow.

Now, on the passage of the energising current, the material making up the chargeable source will store the energy in a way and a mode determined by that mineralisation. As the induced polarization effect is proportional to the *total surface area* presented to the passage of the energising current, very finely electrically continuous sulphides will store significantly more charge than will an equivalent value of coarsely disseminated material. Also should the grains be *finely* divided and/or *sharp grain boundaries*, or *platey* they will acquire the energy *rapidly* and on cessation of the energising current will discharge *rapidly*. Coarser grained material will take longer to charge and discharge than will fine grained material. Thus the speed at which the stored energy discharges can be a diagnostic feature. This is particularly true when mineralisation of markedly different form is required to be differentiated where other properties such as conductivity show no diagnostic difference.

As has been seen in Figures 2 to 5 the external (positive) chargeability above the body is characteristic of the internal current flow *within the body*. Thus the observed decay form is characteristic of the mode of the decay *within the body*.

In the case of electrical induced polarization, the energy stored in the body is observed on the surface *after it is discharged through the environment*. Thus as the discharge passes through the medium between the source and the surface the medium will, under normal circumstances, modify not only the amplitude of the signal but also the decay form thereof. In "normal" circumstances where the medium is both resistive and chargeable, the decay form observed via the electric mode

will be characteristic of the *medium* and *not the source*. However, as the intrinsic chargeability and resistivity of the medium approaches zero, the observed decay *will approach that of the source*. The decay form observed in the electric mode in highly conductive areas of intense oxidation such as Cobar New South Wales and Kalgoorlie, Western Australia approaches that of the source mineralisation, while over normal resistive cover the observed EIP decay form is always "normal". Therefore, only when induced polarization measurements are taken in the magnetic mode can the nature of the causative mineralisation's decay form be identified.

How Decay Information is Obtained:- The Scintrex IPR-8 induced polarization receiver is used in the time domain. This instrument records the chargeability at various times *after* the energising current is switched off and in this way records the rate at which the stored energy is discharged. The resultant recorded slices M_1, M_2, M_3, M_4, M_5, M_6 are then "normalised" by the receiver in such a way as to read $M_1 = M_2 = M_3 = M_4 = M_5 = M_6$ *if* the decay form is normal. Chargeable responses from finely grained mineral assemblages will be indirectly shown up in the field as $M_1 > M_2 > M_3 > M_4 > M_5 > M_6$, while coarser grained mineral assemblages will be shown up as $M_1 < M_2 < M_3 < M_4 < M_5 < M_6$.

In the display of the data ΔM is plotted where $\Delta M = |M_6| - |M_1|$. In this shortened presentation ΔM will be positive and large when "coarse grained" material is the source, 0 for "normal grain" size sources and large and negative for "fine grained" mineral assemblages.

The Importance of Frequency:- When induced polarization is carried out over a range of frequencies in the magnetic mode whether in terms of chargeability (time domain) or phase angle or frequency effect (in the frequency domain), a band of frequencies

is observed at which the maximum IP effect occurs. Frequencies above and below this band show a *reduced* effect. This *"peak frequency"* can be an important factor in whether the source material is seen clearly or not. The wholly electrically discontinuous disseminated deposits such as porphyry coppers have peak frequencies of 6Hz - 40Hz while massive electrically continuous coarse grained deposits occur low in the spectrum (less than 0.1Hz to 1Hz).

Much has to be learnt about this phenomenon (which is of course equivalent to decay form information), but it does indicate that the effective frequencies at which any IP (EIP or MIP) is carried out is important.

Other Significant Points:-

1 - The above remarks refer to time domain. However, measurements made in terms of frequency effect and phase angle in the frequency domain, show precisely the same phenomenon. There are distinct advantages in the application of MIP in the frequency domain, in particular the signal to noise ratio for a given transmitter is significantly greater with frequency. A disadvantage is that the effective "multi-frequency" approach of observing the decay form cannot be obtained without repeating the work at various frequencies.

2 - In several unique circumstances the method has vast superiority over other electrical methods. Firstly, in areas where the external (positive) current flow from the IP source cannot reach the surface either because of dessicated desert sands and/or due to a complete shorting out of the return signal at the base of a salt lake. Other cases where MIP is a unique solution is where chargeable material (even when weakly so) such as heavy minerals occur in barren quartz sands. In such circumstances EIP techniques

cannot be applied. One further application is when materially different mineral assemblages having markedly different decay forms are required to be differentiated.

3 - In applying the MIP method to exploration problems there are two characteristics problems to solve. The first is to clearly ascertain what are the range of characteristics of the target sources, and secondly to apply the method in an effective *and* efficient manner. This may mean commencing test work in the time domain and carrying out production surveys in whichever domain gives the greatest efficiency.

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GEOPHYSICAL CONSULTANTS AND CONTRACTORS

SCINTREX IPR-8 INDUCED POLARIZATION

TIME DOMAIN RECEIVER



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I INTRODUCTION

The basic equipment required for an Induced Polarization survey consists of a transmitter, a receiver, wire and electrodes.

Most time domain induced polarization transmitters transmit square waves with equal "on" and "off" times. Polarity is automatically changed between the pulses. The waveform shown below indicates how the current is usually transmitted. The pulse times range from 1 to 8 seconds.

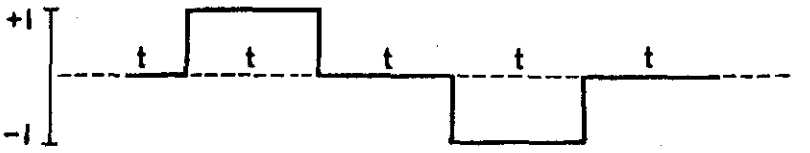


FIGURE 1A

The transmitter is powered by batteries (portable type units) or a motor driven generator. Scintrex manufactures various time domain induced polarization transmitters ranging in power from 25 watts to 15 kW. The choice of a transmitter depends on various factors such as: the electrode spacings to be employed, contact resistance and the resistivity of the subsurface. The IPR-8 receiver is designed for use with any time domain induced polarization transmitter.

The IPR-8 time domain induced polarization receiver is of the state-of-the-art design, packaged in a rugged and portable manner. Using integration and automatic normalization, it measures the characteristics of an induced polarization decay curve set up by overvoltage and other effects occurring in rocks. When induced polarization effects (such as due to metallic-non metallic interfaces in rocks) occur, the waveform received at the receiver is not the same square wave as transmitted by the transmitter. The waveform shown below indicates the sort of wave distortion which is caused by the induced polarization phenomena.

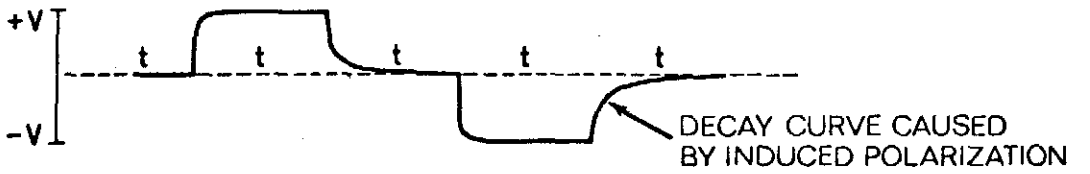


FIGURE 1B

II SPECIFICATIONS

The IPR-8 has the following specifications:

Input Impedance	3 megohms
Primary Voltage (Vp) Range	300 microvolts full scale to 40 volts full scale in 10 ranges
Accuracy of Vp Measurement	±3% of full scale
Vs/Vp Ranges	20 and 100 mV/V full scale
Vs/Vp Accuracy	±3% of full scale
Primary SP Buckout Range	±1 volt
Accuracy of SP Measurement	±3%, ±5 mV
Automatic SP Tracking Range	6 x Vp, maximum ±1 volt
Continuity Meter Reading	0 - 500 k ohms
50 or 60 Hz Powerline Rejection	-50 db (300x)*
Low Pass Filter	6 db/octave with fc = 20 Hz and 12 db/octave with fc = 36 Hz
Required Stability of Transmitter Timing	Need only exceed measuring program selected (1 or 2 seconds)
Operating Temperature Range	-30°C to +60°C
Dimensions	320 mm x 135 mm x 160 mm
Weight, Complete with Lid and Batteries	3.6 kg
Power Supply	4 D cells - Eveready No. 1050 or equivalent; estimated battery life 2 months intermittent duty at 25°C 1 Alkaline cell Eveready No. E91 or equivalent; estimated life 1 year

* 50 or 60 Hz depending on power system.

III QUANTITIES MEASURED BY THE IPR-8

Figure 2 shows the different parameters measured by the IPR-8. The usual measurements are Vp, the received primary voltage and "M", a parameter related to the transient curve. The Vp measurement is used in resistivity calculations while M is the chargeability (induced polarization) parameter. In addition, absolute values of the self-potential (SP) can be measured.

In all cases, the M quantity measured by the IPR-8 is the mean value of the transient voltage over a selected time interval to which the following normalizations have been applied:

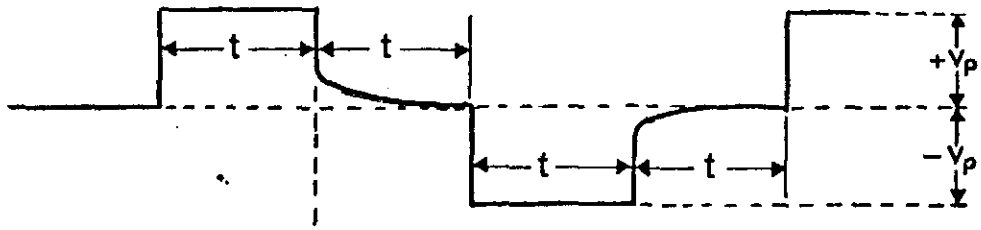
- normalization for the length of the integration interval
- normalization for the primary steady state voltage (Vp)
- normalization for curve shape
- normalization for number of pulses

The units of the quantities measured are, therefore, dimensionless and are normally expressed in "millivolts per volt".

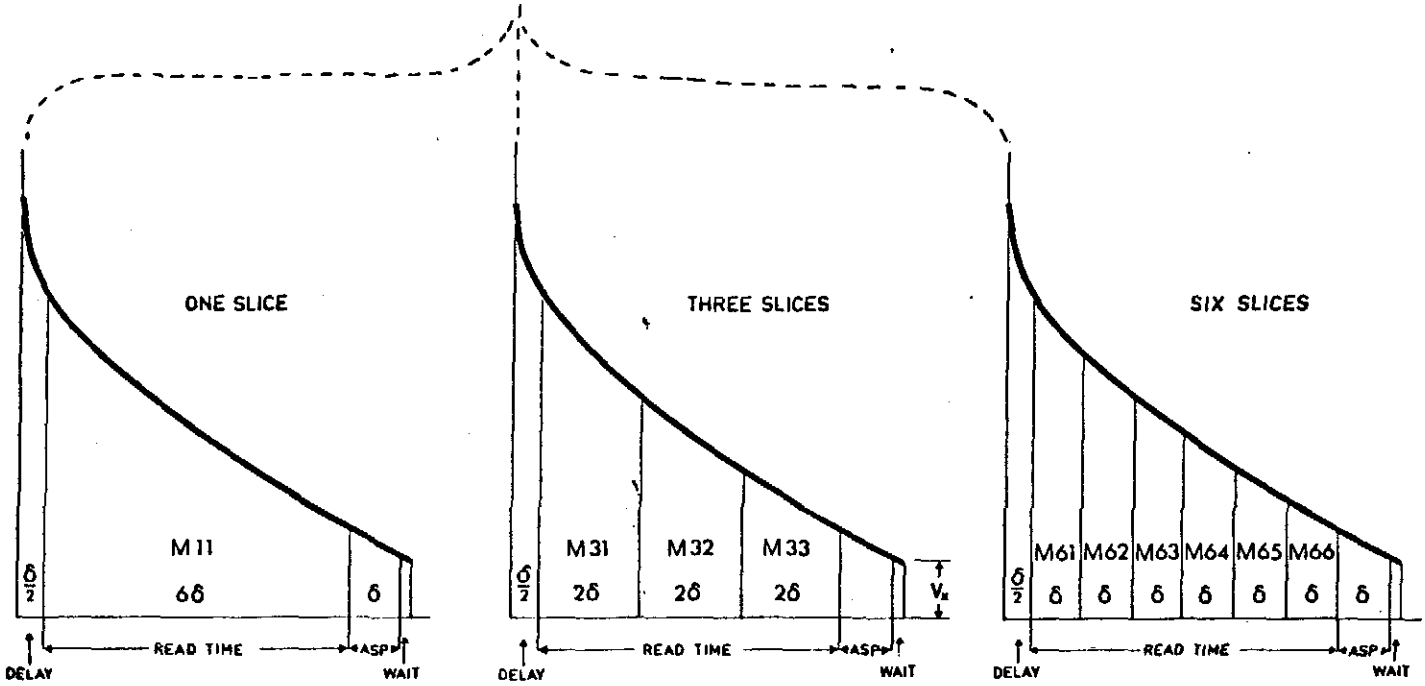
In the various modes of operation the transient voltage following the interruption of the primary current pulse is either integrated over one long period of time or sliced into either 3 or 6 slices. By using 6 slices, a good record of the decay curve shape can be obtained. The 3 slice mode gives some curve shape information and provides an economical standard mode in which to operate. The centre slice of this mode is reasonably close to the measurement made by the Scintrex IPR-7 and other receivers of the "Newmont Type", while the first and last slices can be used for a rapid check of curve shape. A more precise relationship is, however, presented later in this section.

Figure 2 shows the actual times used. For the receiver to operate, the transmitter timing may be any time period of one second or greater (i.e. $t \geq 1$ second) although transmitter and receiver timings of 2 seconds are considered normal for most surveys. Equal on and off timing assures the best noise rejection as the signal is averaged over the longest possible time, and the automatic self-potential adjustment is made closest to the reading time.

With the receiver set at $t = 1$ second, the decay ($\delta/2$) from the current-off time to the commencement of the measurement is 65 milliseconds and the slice width (δ) is 130 milliseconds. With the receiver set at $t = 2$ seconds the delay is 130 milliseconds and the slice width is 260 milliseconds. Fuller information on the programs is available from the tables in Figure 2.



WAVE FORM AT INPUT TERMINALS



SECONDARY DECAY CURVE SHAPES AS APPLIED TO THE INTEGRATORS

t sec.	δ	delay time	waiting time	M 11				M 31			M 32			M 33			length
				from	to	mean	length	from	to	mean	from	to	mean	from	to	mean	
1	130	65	25	65	845	455	780	65	325	195	325	585	455	585	845	715	260
2	260	130	50	130	1690	910	1560	130	650	390	650	1170	910	1170	1690	1430	520

t sec.	M 61			M 62			M 63			M 64			M 65			M 66			length
	from	to	mean	from	to	mean	from	to	mean	from	to	mean	from	to	mean	from	to	mean	
1	65	195	130	195	325	260	325	455	390	455	585	520	585	715	650	715	845	780	130
2	130	390	260	390	650	520	650	910	780	910	1170	1040	1170	1430	1300	1430	1690	1560	260

FIGURE 2

PARAMETERS MEASURED WITH TIMES OF RECEIVER PROGRAM IN MILLISECONDS.

607

221266

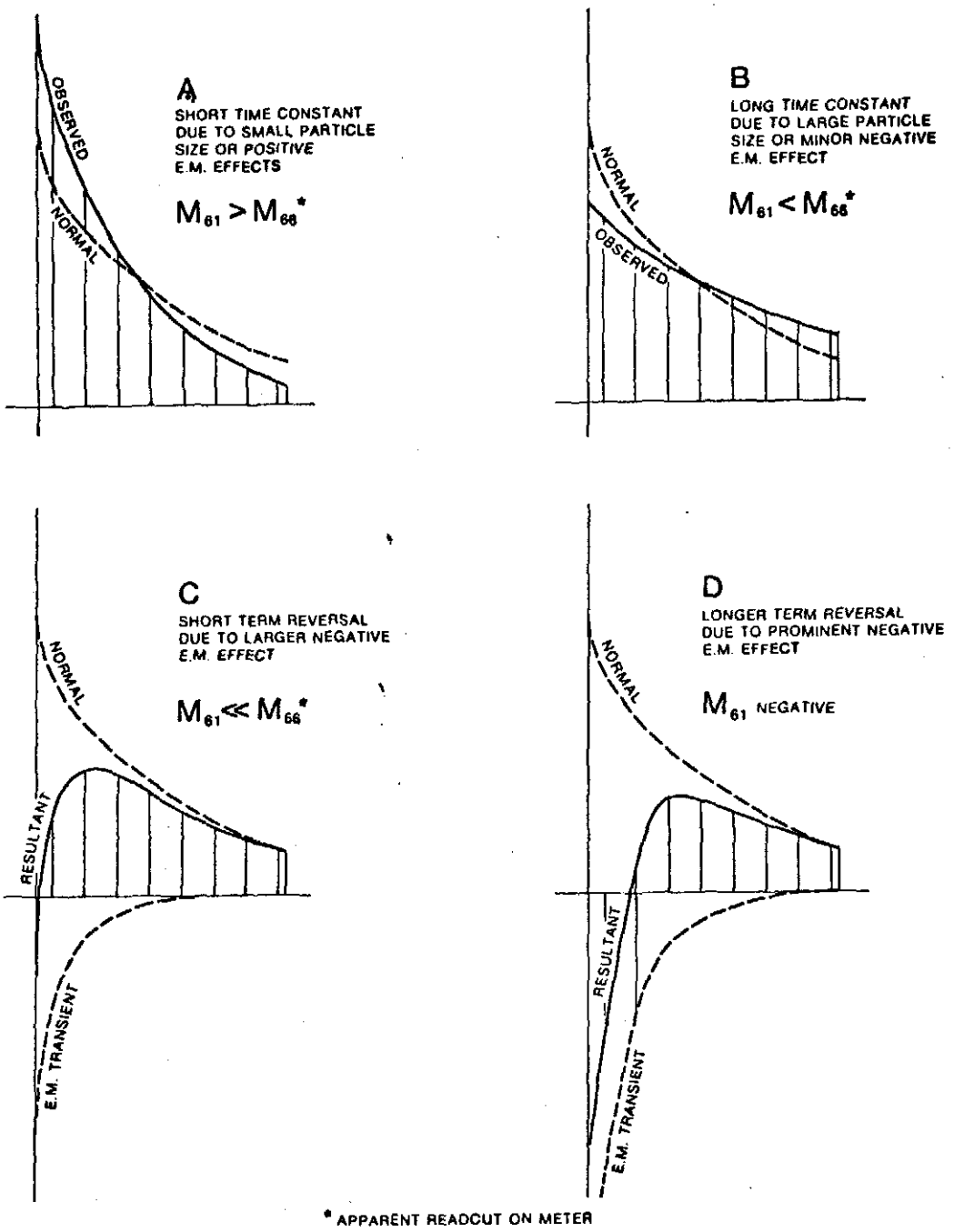


FIGURE 3

THE SIGNIFICANCE OF CURVE SHAPE INFORMATION GAINED USING 6 SLICE READINGS.

Each integration is normalized with respect to the Standard Induced Polarization Decay Curve which has been established by Newmont Exploration Limited. (ref. Dolan and McLaughlin in bibliography) This is achieved by choosing the sensitivities of the integrators so, that if the curve shape is normal, all slices within a given mode show the same amplitude of measurement. A further normalization is built in for the slice width, be it full, one-third or one-sixth of the total integration period. The net effect is that the reading will be the same regardless of the slice measured, providing that a standard transient decay curve form is present and that the same measuring cycle is used for transmitter and receiver (1 second or 2 seconds). Any departure from this standard curve form will be immediately obvious to the operator, without performing any calculations. For instance, a steeper decay will give a higher reading on earlier slices than on later slices. Reconstruction of the actual decay curve is easily effected by using the correction factors given in Table 1.

The shape of a time domain induced polarization decay curve can be altered by electromagnetic or interline coupling, by variations in the average size or degree of interconnection of the metallic particles in the bedrock or by other I.P. sources. Figure 3 illustrates the advantage of breaking the decay curve into slices. Utilizing only one wide slice, there is no indication of the shape of the decay curve. Positive electromagnetic coupling effects or small particle size may give rise to an abnormally short time constant (Case A) which, for multi-slice modes will be indicated by higher normalized readings of the earlier slices with respect to the later slices. An increase in the later slices over the earlier ones (Case B) may imply a longer time constant due to a minor negative EM transient or I.P. responses from large metallic particles, etc. Cases C and D, where the values of the initial slices are considerably reduced or are even negative, show the effect of negative EM transients of increasing amplitude.

A system of symbols has been created to indicate each of the measurable slices.

The general symbol is M_{txy} where:

- t is the timing chosen (i.e. 1 or 2 seconds)
- x is the number of slices in the mode chosen (i.e. 1, 3 or 6)
- y is the number of the slice referred to (i.e. 1, 2, 3, 4, 5 or 6)

Wherever two subscripts only are given, eg. M_{32}, it is understood to apply equally for t = 1 sec. or t = 2 sec.

A chargeability reading is defined by the following formula:

$$M = \frac{V_s \cdot 1000}{V_p} \quad \text{in mV/V}$$

where $$V_s = \frac{t_1 \int^{t2} V_s \, dt}{t_r} + V_x$$

and t_1 = time at beginning of slice

t_2 = time at end of slice

V_x = residual transient voltage at the end of the automatic self potential correction

$t_r = t_2 - t_1$, i.e. the integrating period

Chargeability values, uncorrected for curve shape, can be easily calculated if required. Normalizations for all slices are made using the M_{232} value as reference. In other words, there is no curve shape normalization applied to this slice; the M_{232} readout is, therefore, directly as measured. The same statement holds for the M_{132} slice, however, its value is one-half the value for M_{232} provided that the transmitter timing matches the receiver timing.

To restore the true transient curve shape (M true), the observed M readings (M read) are multiplied by the factors in Table 1.

TABLE 1

$$M_{true} = M_{read} \bullet k_1$$

Slice	k_1	
M_{11}	1.09	
M_{31}	1.47	
M_{32}	1.00	← NORMAL
M_{33}	0.81	
M_{61}	1.68	
M_{62}	1.27	
M_{63}	1.06	
M_{64}	0.94	
M_{65}	0.85	
M_{66}	0.78	

For the ideal "normal" I.P. transient curve form $M_{2xy} = 2M_{1xy}$ where M_{2xy} is for a 2-second on-off transmitter cycle and M_{1xy} is for a 1-second on-off cycle. The relationship between readings taken with differing transmitter and receiver timings is more complicated, particularly if the curve shapes are not normal.

Table 1 still applies for the case where the transmitting times are longer than the receiving times in order to reconstruct the relative curve shape.

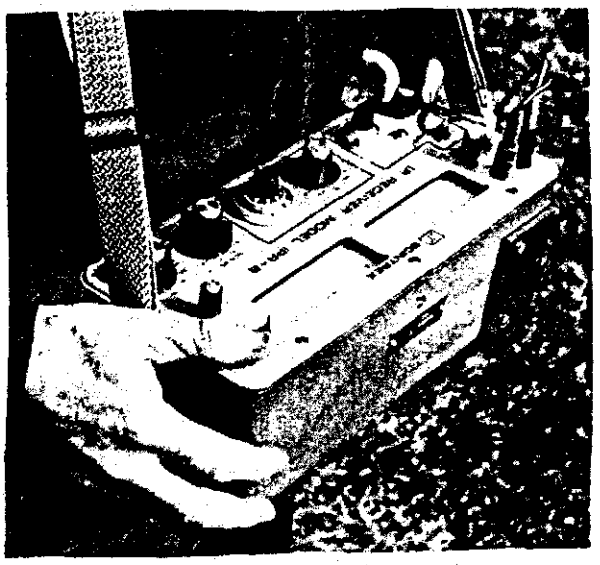
611

221270

Relationship between IPR-8 and
"Newmont Type" Receiver Measurements

The "Newmont Type" receivers (eg. Scintrex IPR-7) integrate the area under the transient curve from 0.45 seconds to 1.1 seconds. This is then multiplied internally be an instrumental factor to obtain the chargeability M in milliseconds.

For a normal decay curve form, the approximate relationship between the IPR-8 measurements and the Newmont Type chargeability is given by M_{232} (in mV/V) = M_N (in milliseconds) • 0.7.



613

221271

A REPORT ON

A RECONNAISSANCE MAGNETIC INDUCED POLARIZATION SURVEY

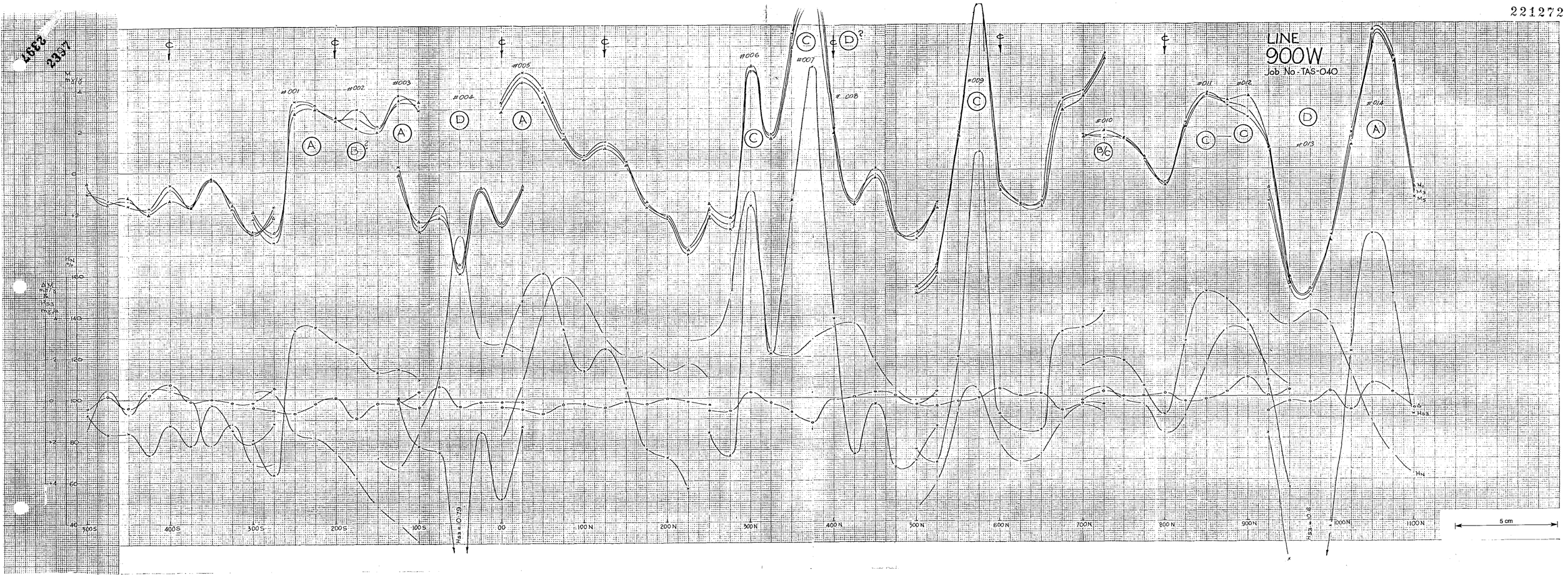
OVER THE MOINA GRID EL 7/74 NEAR WILMOT, TASMANIA

ON BEHALF OF

COMALCO LIMITED

MICROFILMED

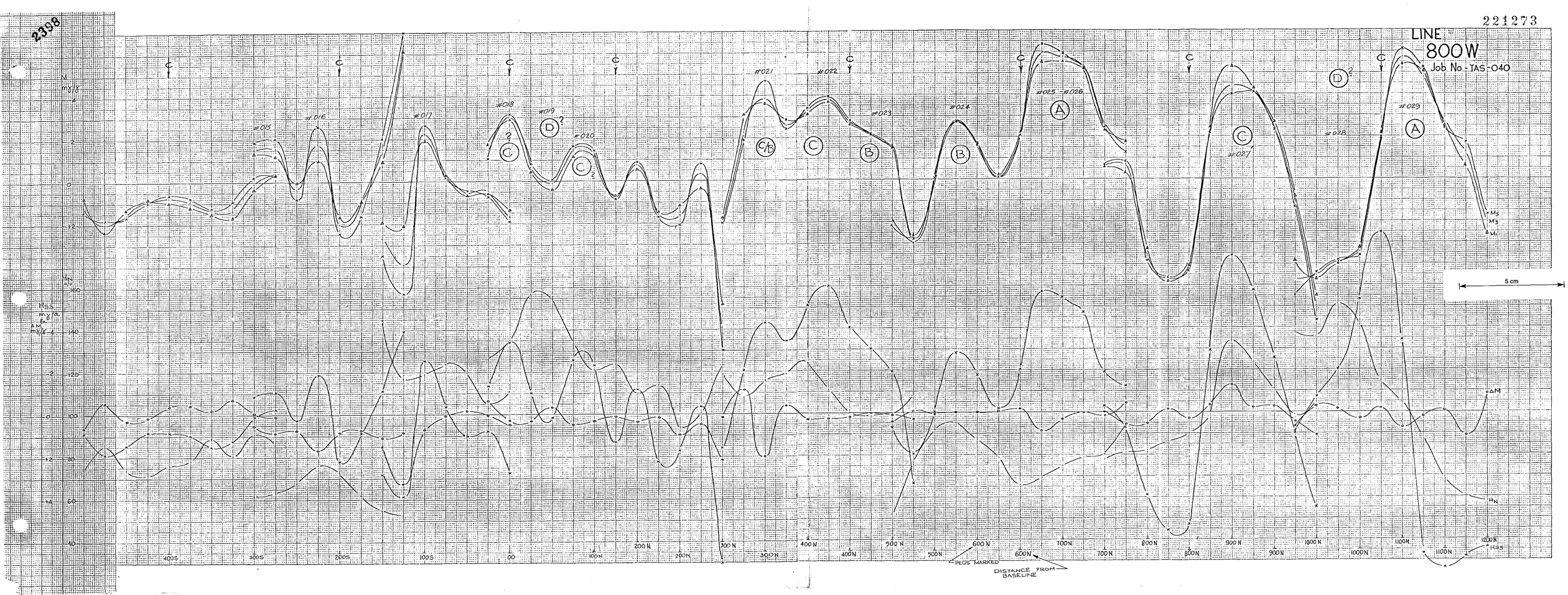
OPEN FILE


LINE
900W
Job No - TAS-040
#001
#002
#003
#004
#005
#006
#007
#008
#009
#010
#011
#012
#013
#014
A
B
C
D
B/C
400 S
300 S
200 S
100 S
00
100 N
200 N
300 N
400 N
500 N
600 N
700 N
800 N
900 N
1000 N
1100 N
Hss = 0.79
Hss = 0.18
5 cm

2398

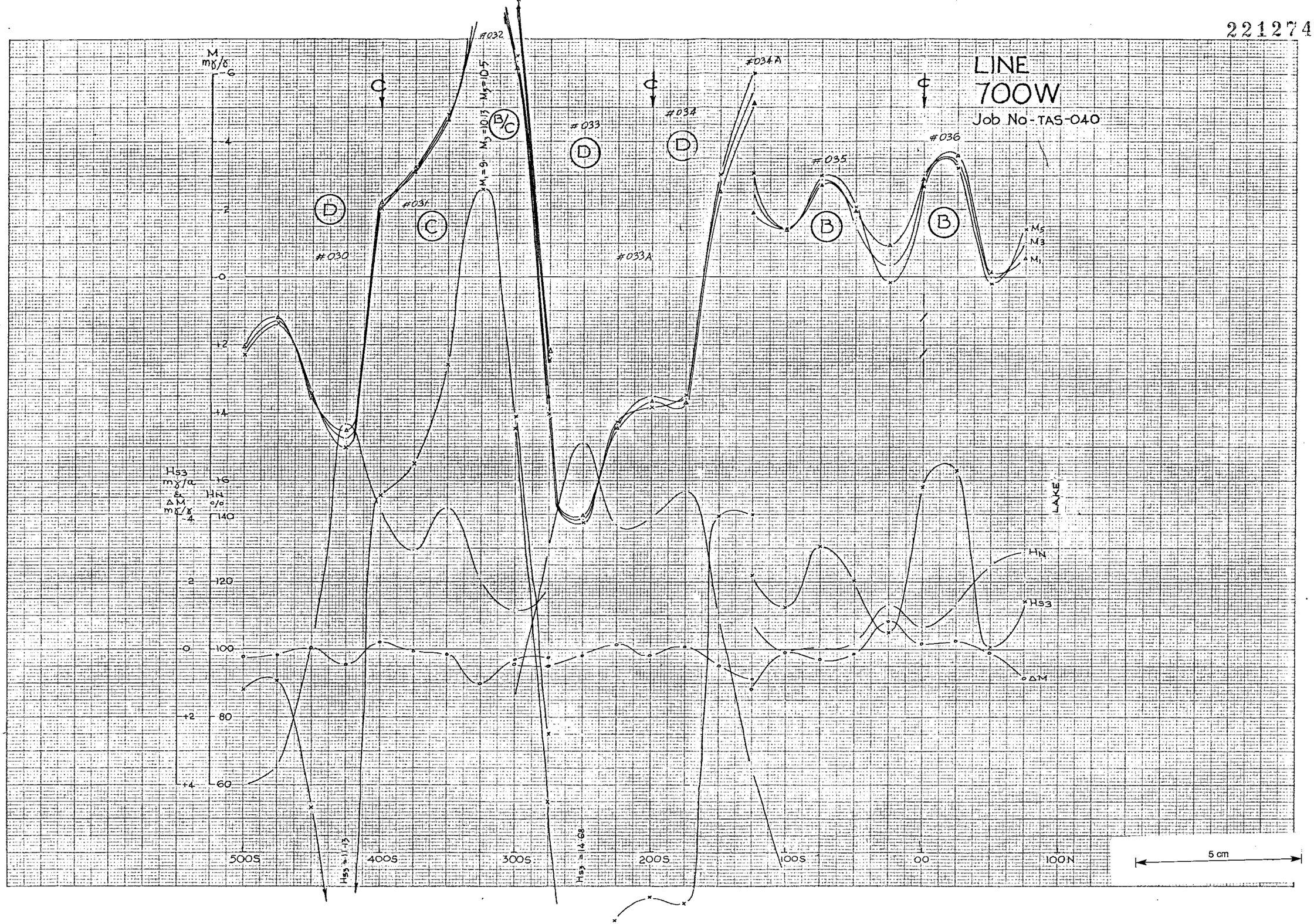
221273

LINE
800W
Job No.-TAS-040



2399

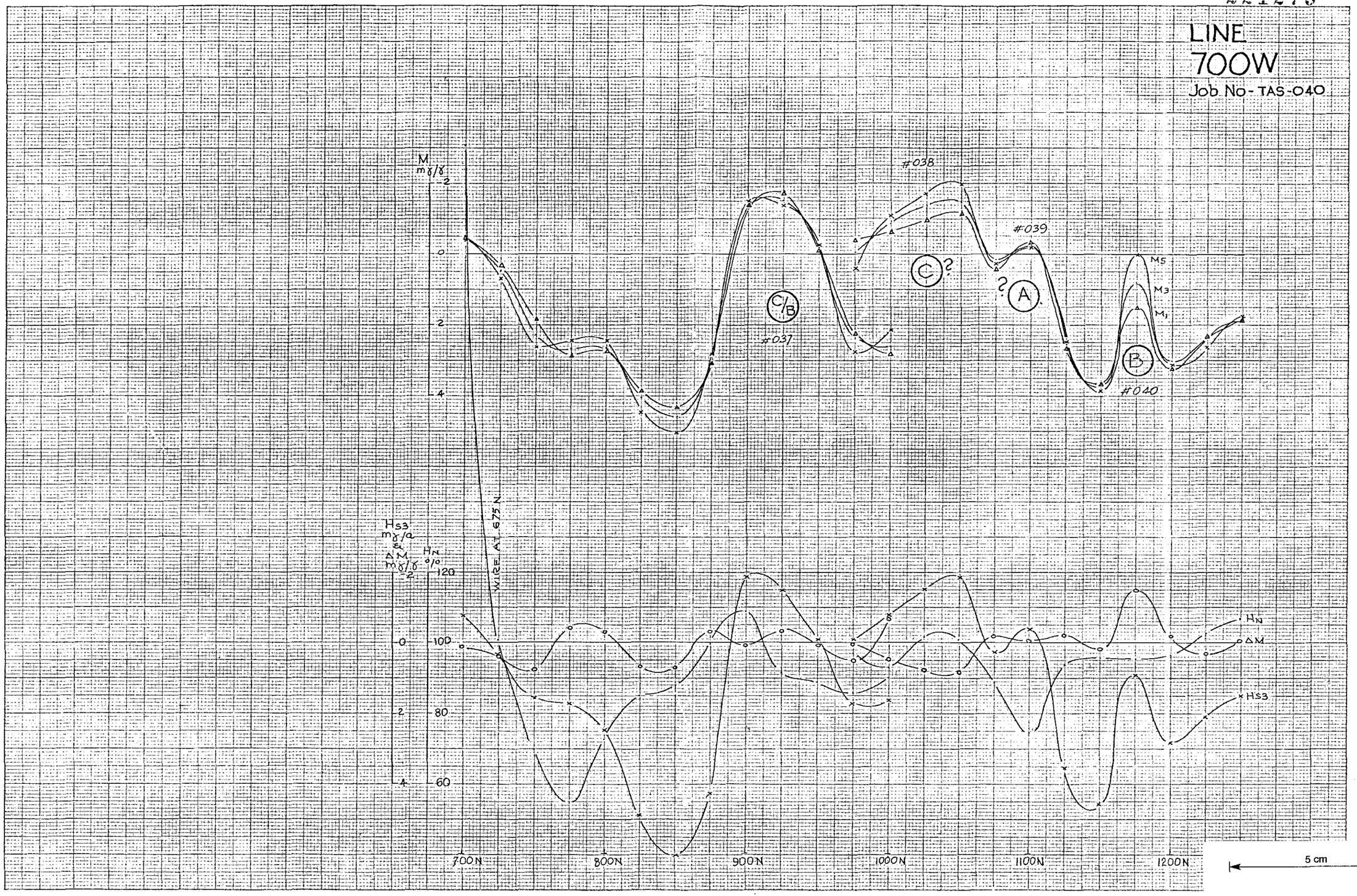
221274



LINE
700W
Job No-TAS-040
M
mγ/γ
Hs3
mγ/a
&
ΔM
mγ/γ
HN
%
#030
#031
#032
#033
#033A
#034
#034A
#035
#036
D
C
B/C
B
M1=9 M3=10.13 M5=10.5
Hs3=11.13
Hs3=14.68
LAKE
HN
Hs3
ΔM
M5
M3
M1
500S
400S
300S
200S
100S
00
100N
5 cm

2400

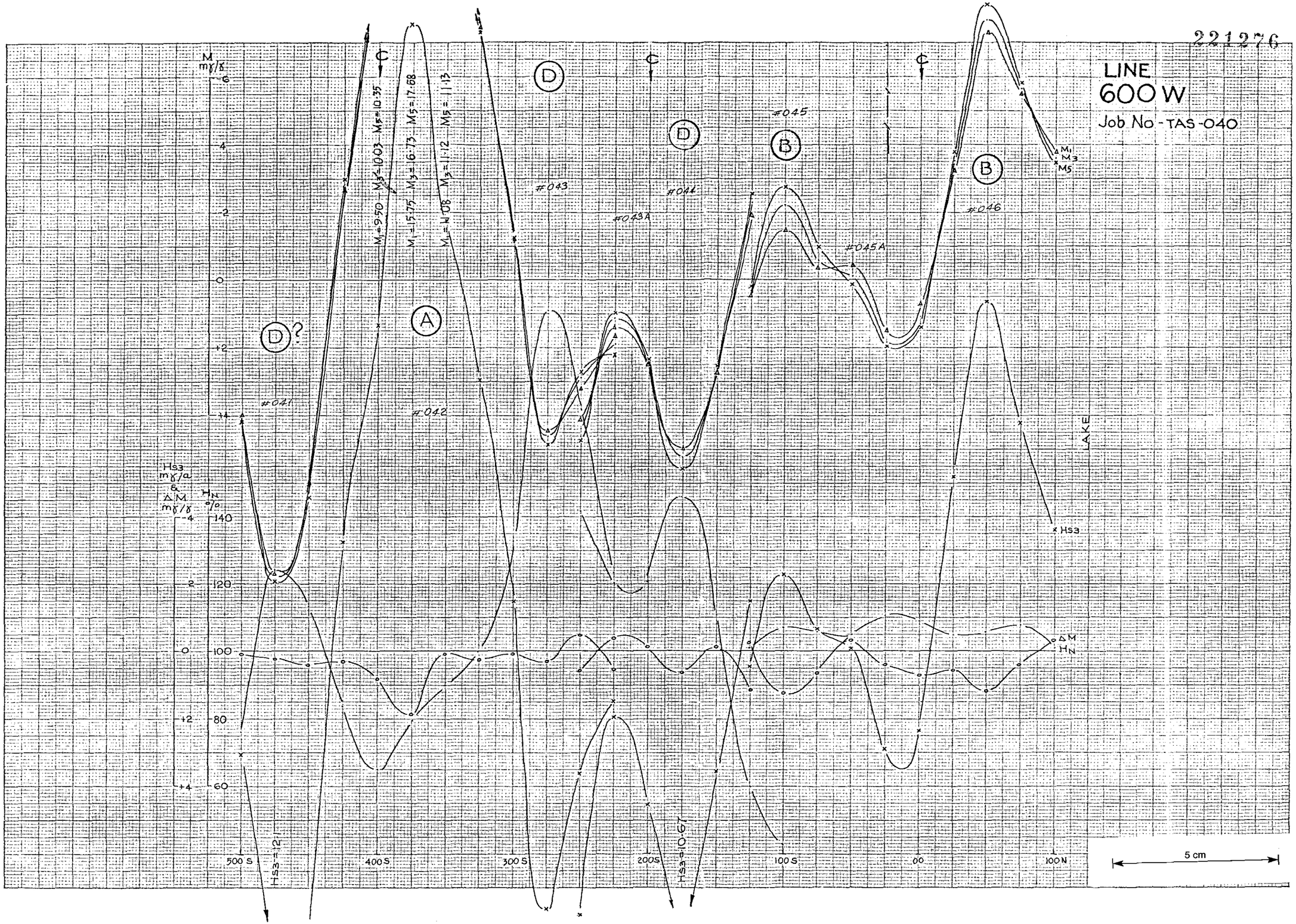
221275



LINE
700W
Job No - TAS-040
M
mγ/γ
2
0
2
4
#038
#039
#037
#040
C/B
C ?
? A
B
M5
M3
M1
WIRE AT 675 N
Hs3
mγ/a
&
ΔM
mγ/γ
HN
%
-2
0
2
4
120
100
80
60
HN
ΔM
Hs3
700N
800N
900N
1000N
1100N
1200N
5 cm

2401

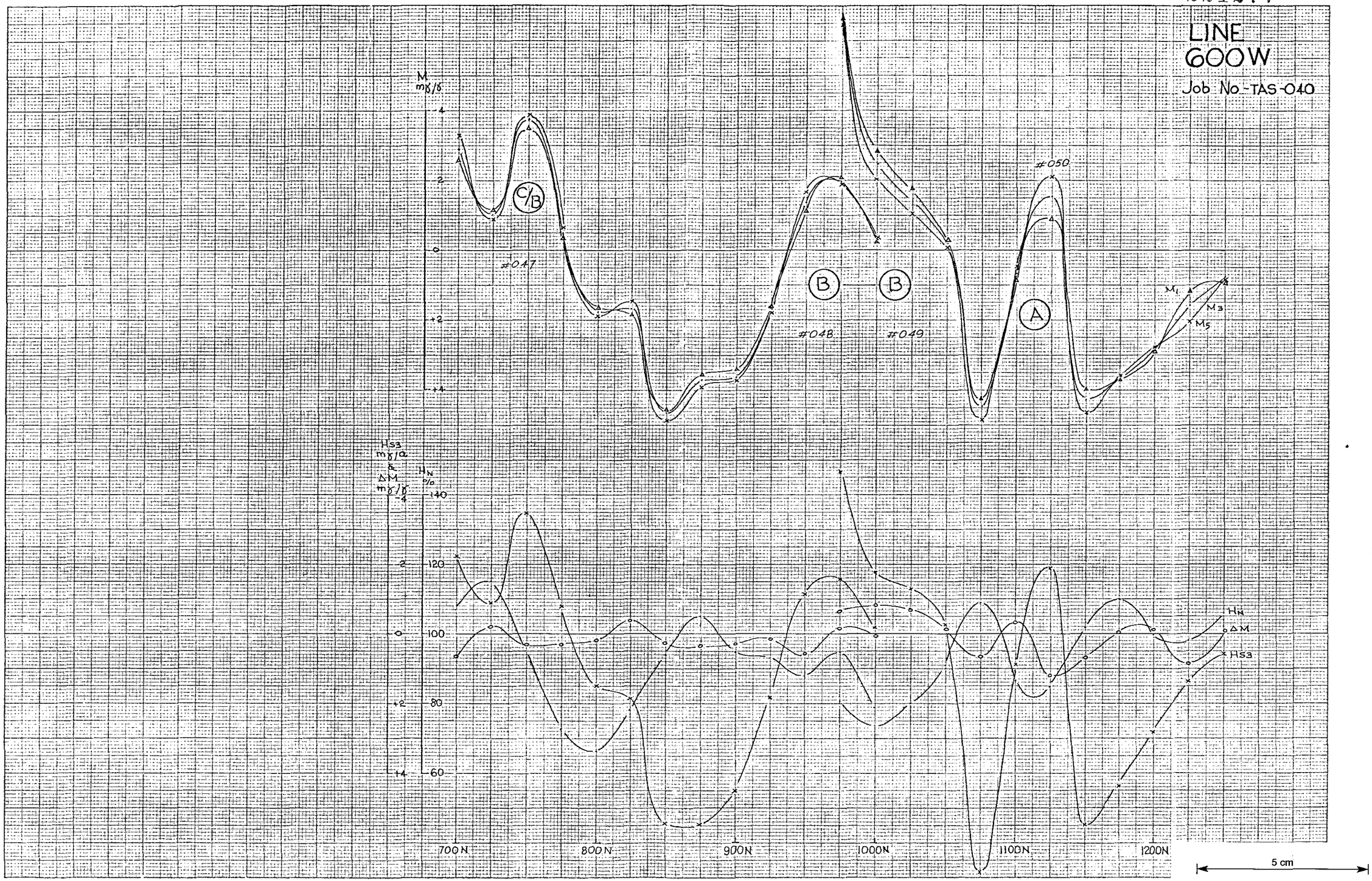
221276



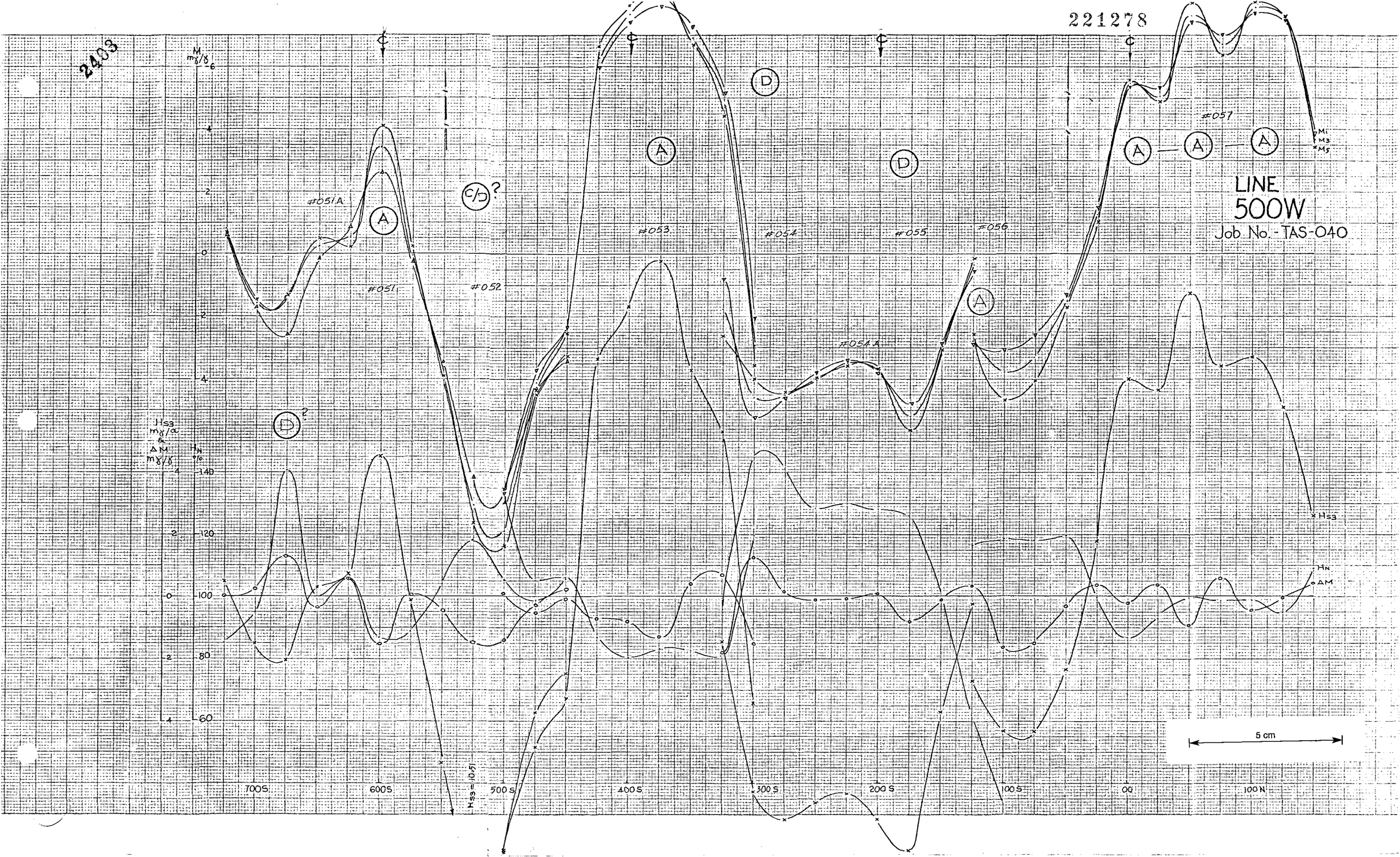
LINE
600 W
Job No - TAS-040
M mγ/γ
Hs3 mγ/a & ΔM mγ/γ
HN %
M1 = 9.50 M3 = 10.03 M5 = 10.35
M1 = 15.75 M3 = 16.73 M5 = 17.68
M1 = 11.08 M3 = 11.12 M5 = 11.13
D?
A
D
B
#041
#042
#043
#043A
#044
#045
#045A
#046
M1
M3
M5
Hs3
ΔM
HN
LAKE
Hs3 = 12.1
Hs3 = 10.67
500 S
400 S
300 S
200 S
100 S
00
100 N
5 cm

2402

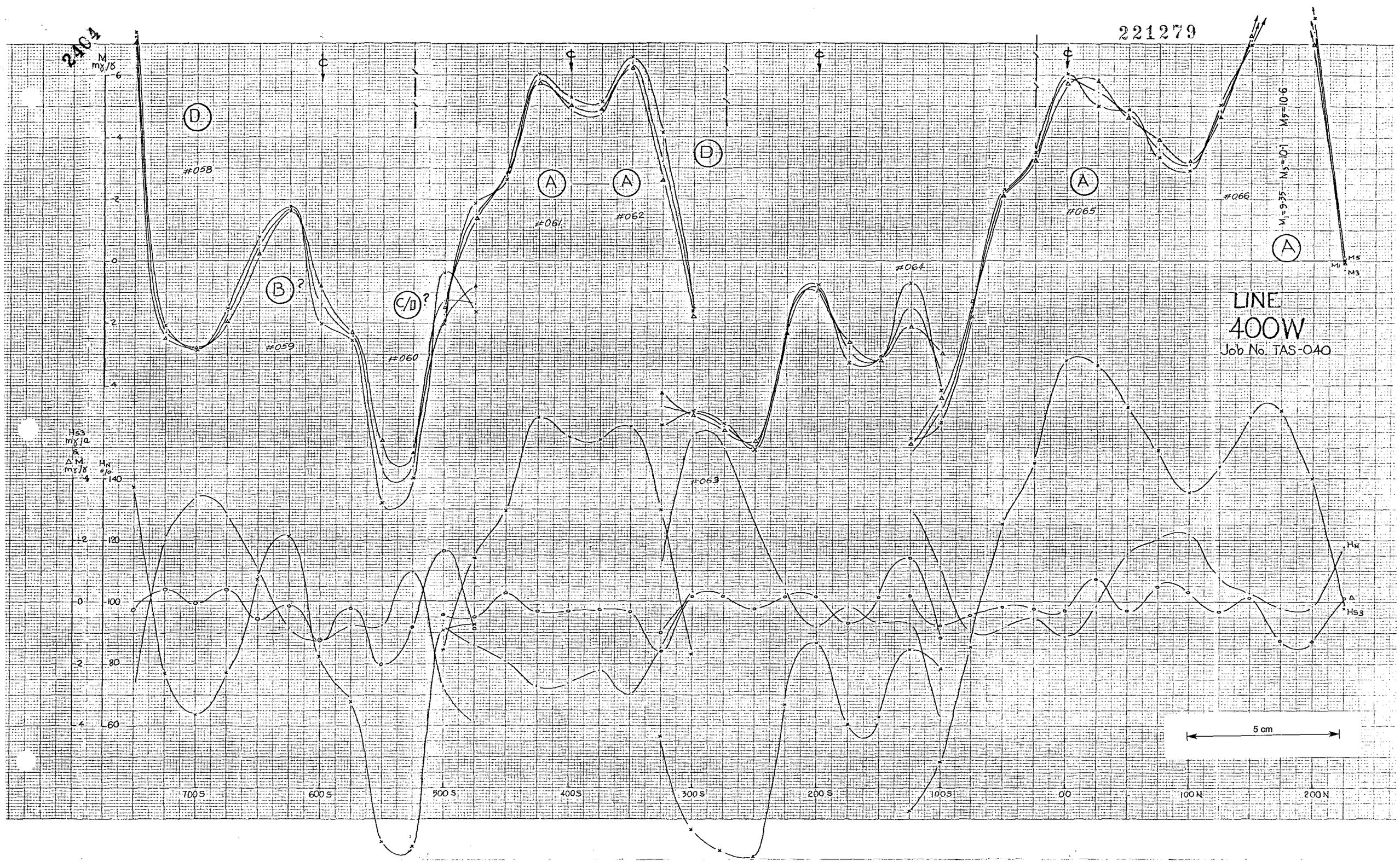
221277



LINE
600W
Job No-TAS-040
M
mγ/γ
-4
2
0
+2
+4
C/B
#047
B
B
#048
#049
#050
A
M1
M3
M5
Hs3
mγ/a
&
ΔM
mγ/γ
HN
%
-4
2
0
+2
+4
140
120
100
80
60
HN
ΔM
Hs3
700N
800N
900N
1000N
1100N
1200N
5 cm



2403
221278
LINE
500W
Job No. - TAS-040
#051A
#051
#052
#053
#054
#054A
#055
#056
#057
A
D
C/D ?
M mγ/γ
Hs3 mγ/a & ΔM mγ/γ
HN %
Hs3 = 10.51
700S
600S
500S
400S
300S
200S
100S
00
100N
5 cm

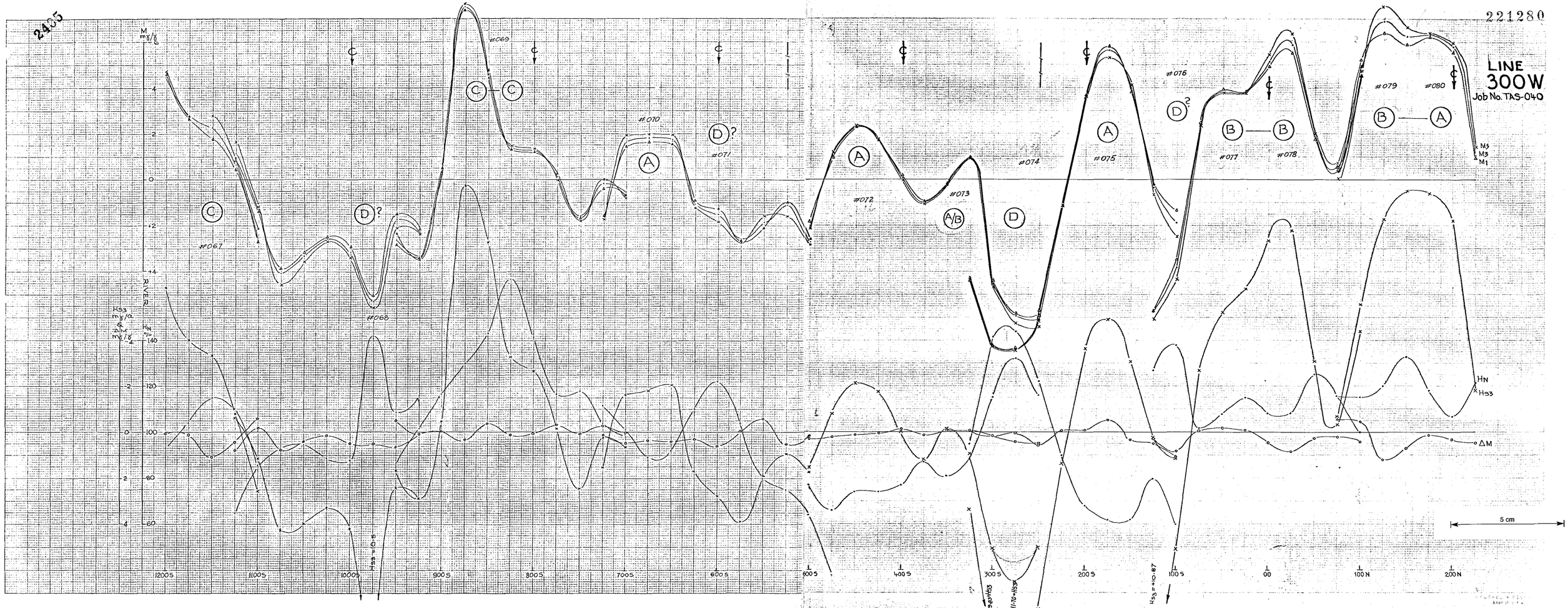


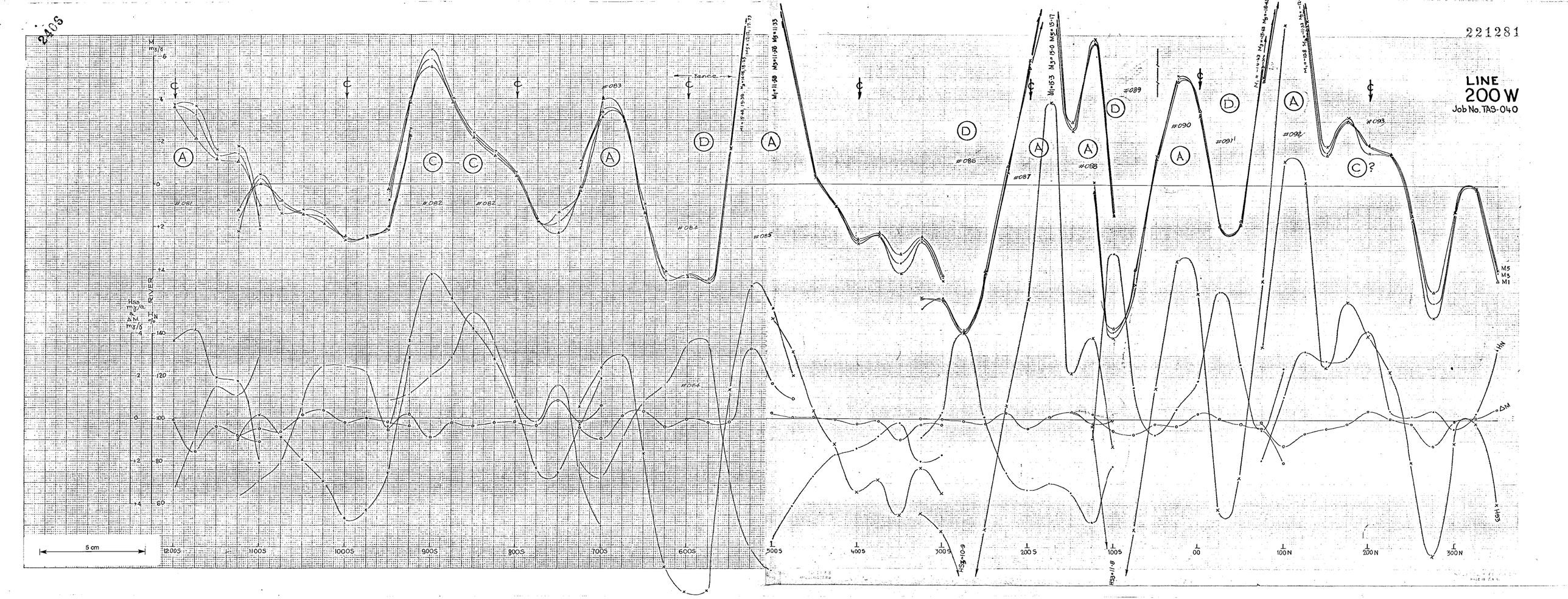
2404
221279
LINE
400W
Job No. TAS-040
M mγ/γ
Hs3 mγ/a & ΔM mγ/γ
HN %
#058
#059
#060
#061
#062
#063
#064
#065
#066
M1=9.35 M3=10.1 M5=10.6
700 S
600 S
500 S
400 S
300 S
200 S
100 S
00
100 N
200 N
5 cm

2435

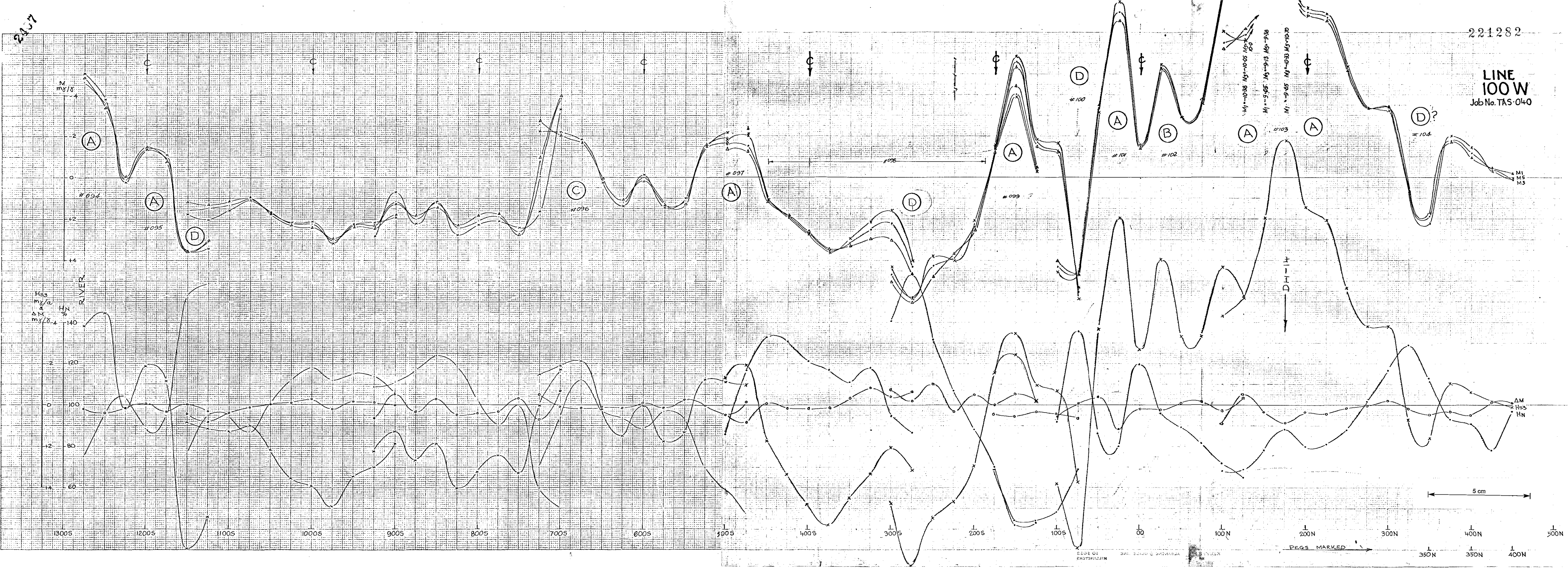
221280

LINE
300W
Job No. TAS-040

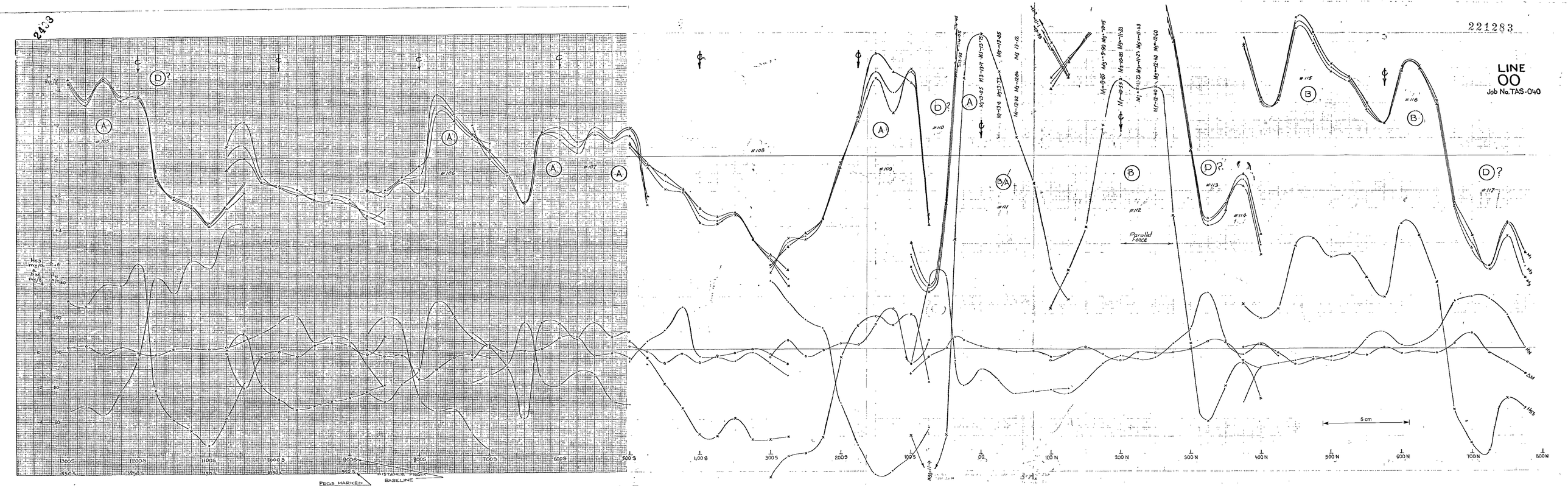




2405
221281
LINE
200 W
Job No. TAS-040
5 cm
1200S
1100S
1000S
900S
800S
700S
600S
500S
400S
300S
200S
100S
00
100N
200N
300N
M5
M3
M1
ΔM
HN
HS3



221282
LINE
100 W
Job No. TAS-040
RIVER
DH-14
5 cm
PEGS MARKED
1300S
1200S
1100S
1000S
900S
800S
700S
600S
500S
400S
300S
200S
100S
00
100N
200N
300N
400N
500N
350N
350N
400N

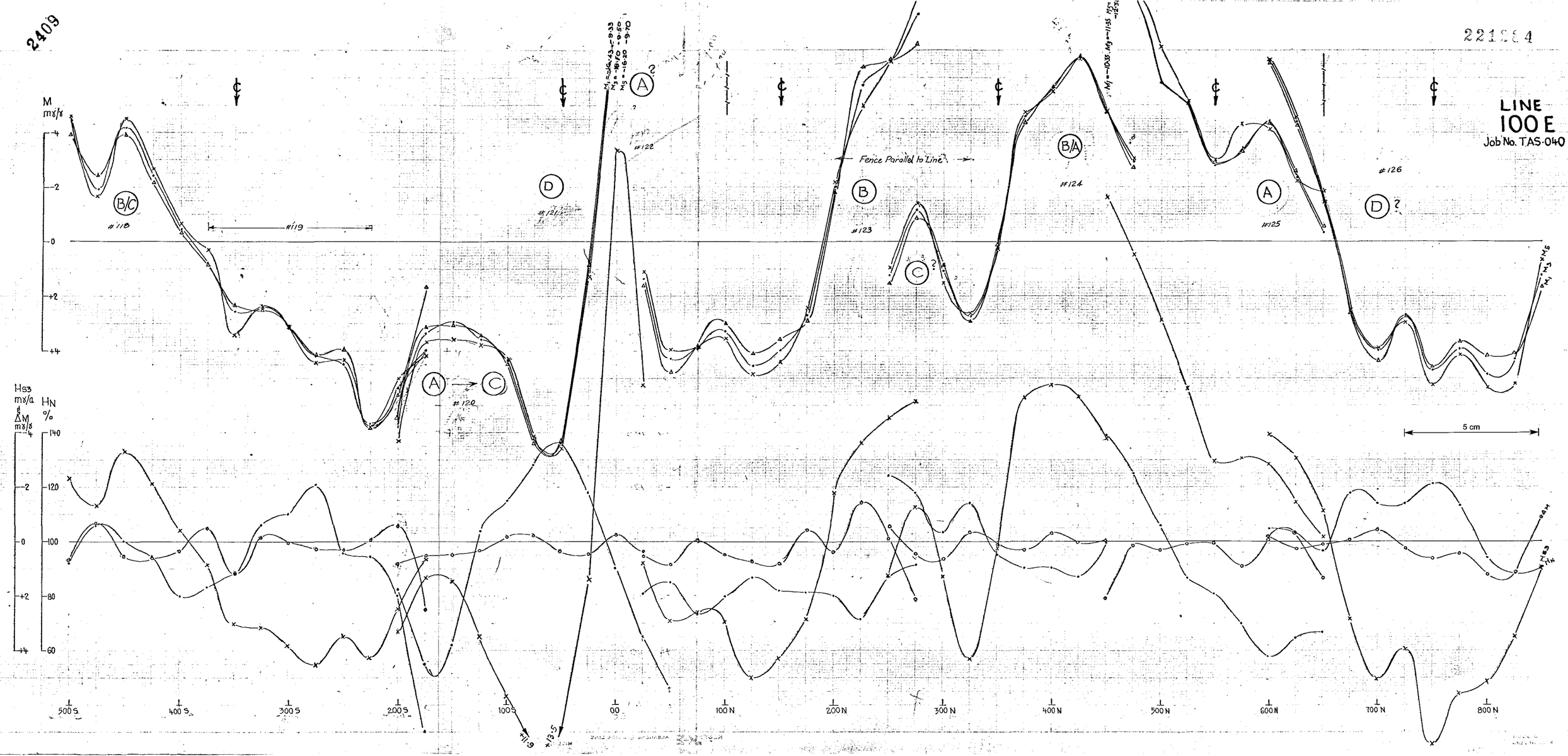


2403
221283
LINE
00
Job No. TAS-040
Parallel Fence
5 cm
PEGS MARKED
DISTANCE FROM BASELINE
1300S
1200S
1100S
1000S
900S
800S
700S
600S
500S
400S
300S
200S
100S
00
100N
200N
300N
400N
500N
600N
700N
800N

2409

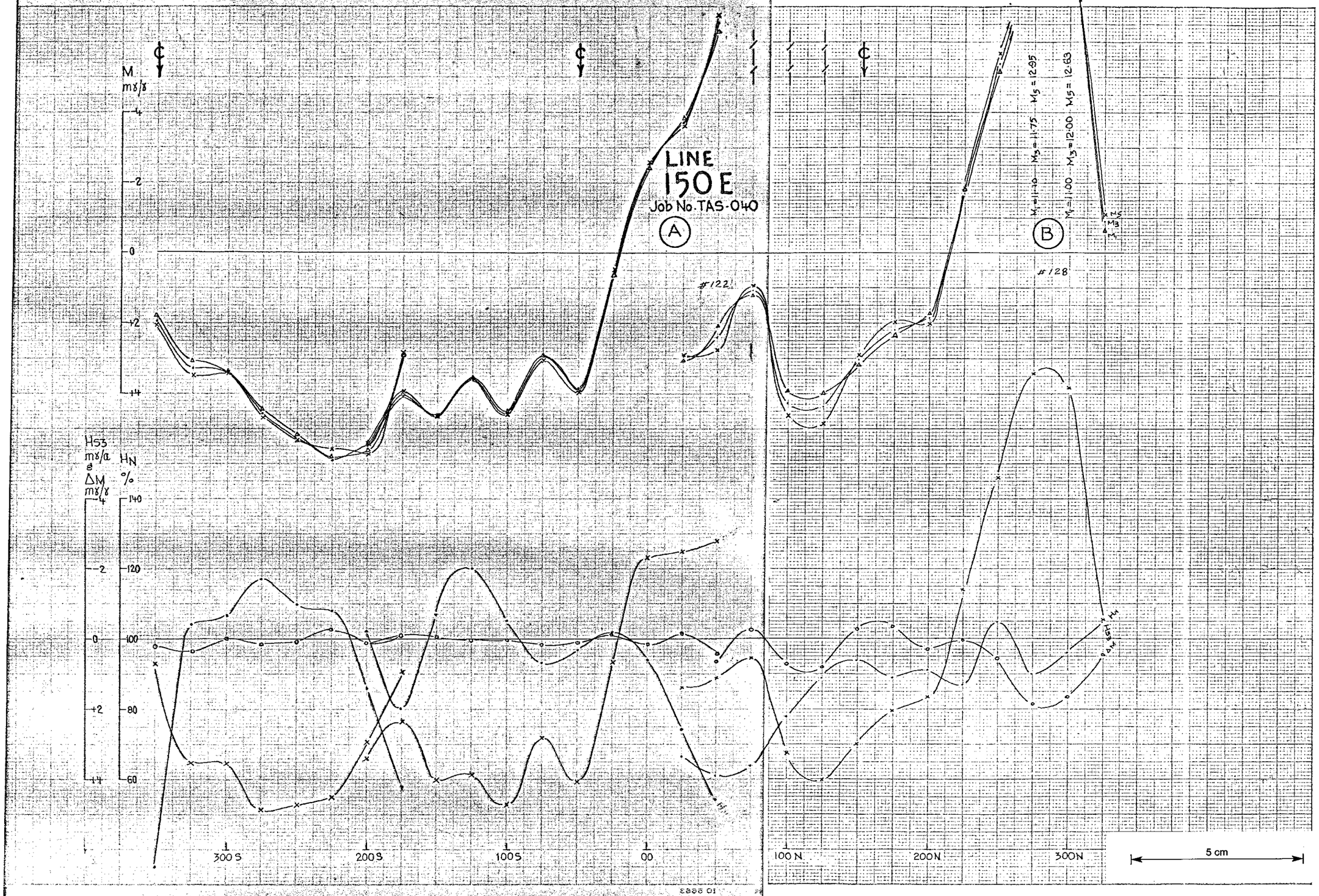
221884

LINE
100 E
Job No. TAS-040



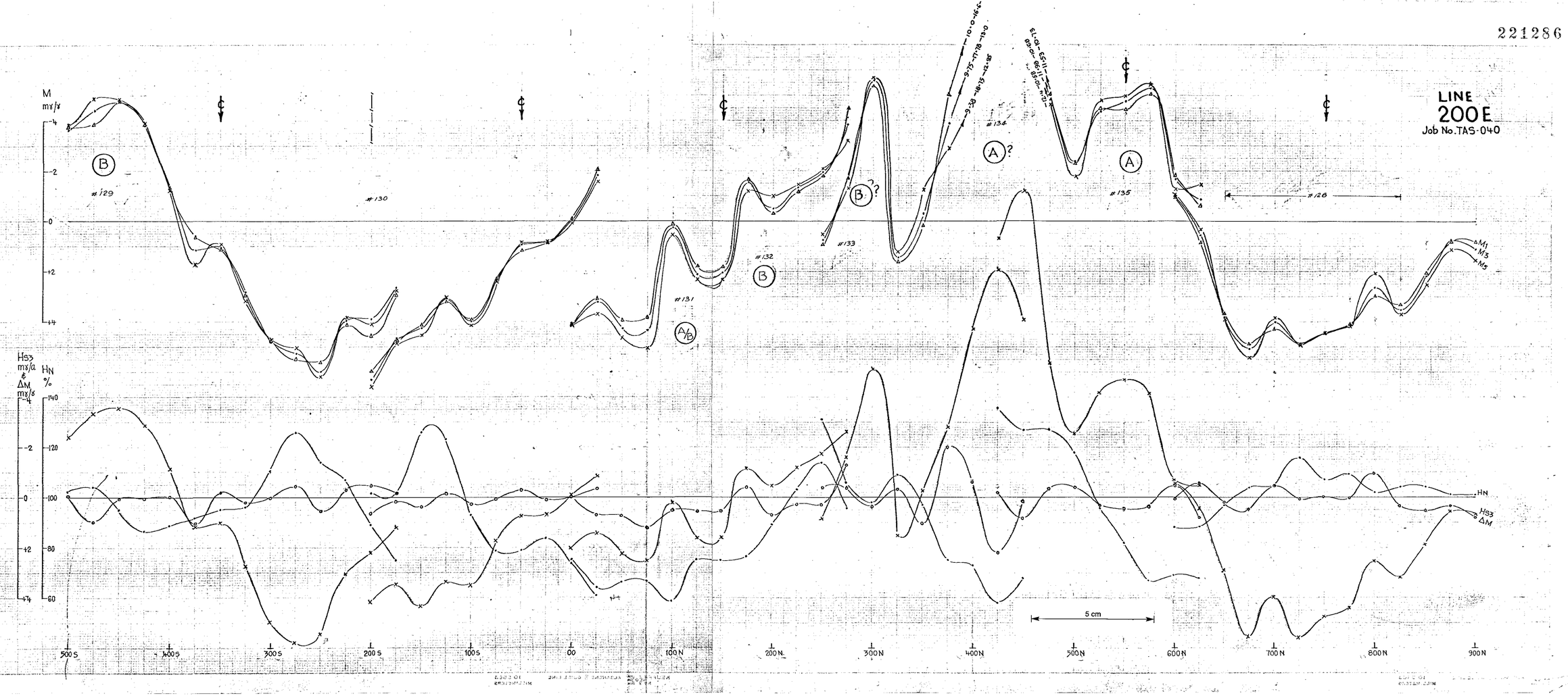
2410

221285


M
mγ/s
+4
-2
0
+2
+4
LINE
150 E
Job No. TAS-040
A
B
#122
#128
M1=11.40 M3=11.75 M5=12.05
M1=11.00 M3=12.00 M5=12.63
M1
M3
M5
HSS
mγ/a
s
ΔM
mγ/s
HN
%
+4
-2
0
+2
+4
140
120
100
80
60
300 S
200 S
100 S
00
100 N
200 N
300 N
5 cm

2411

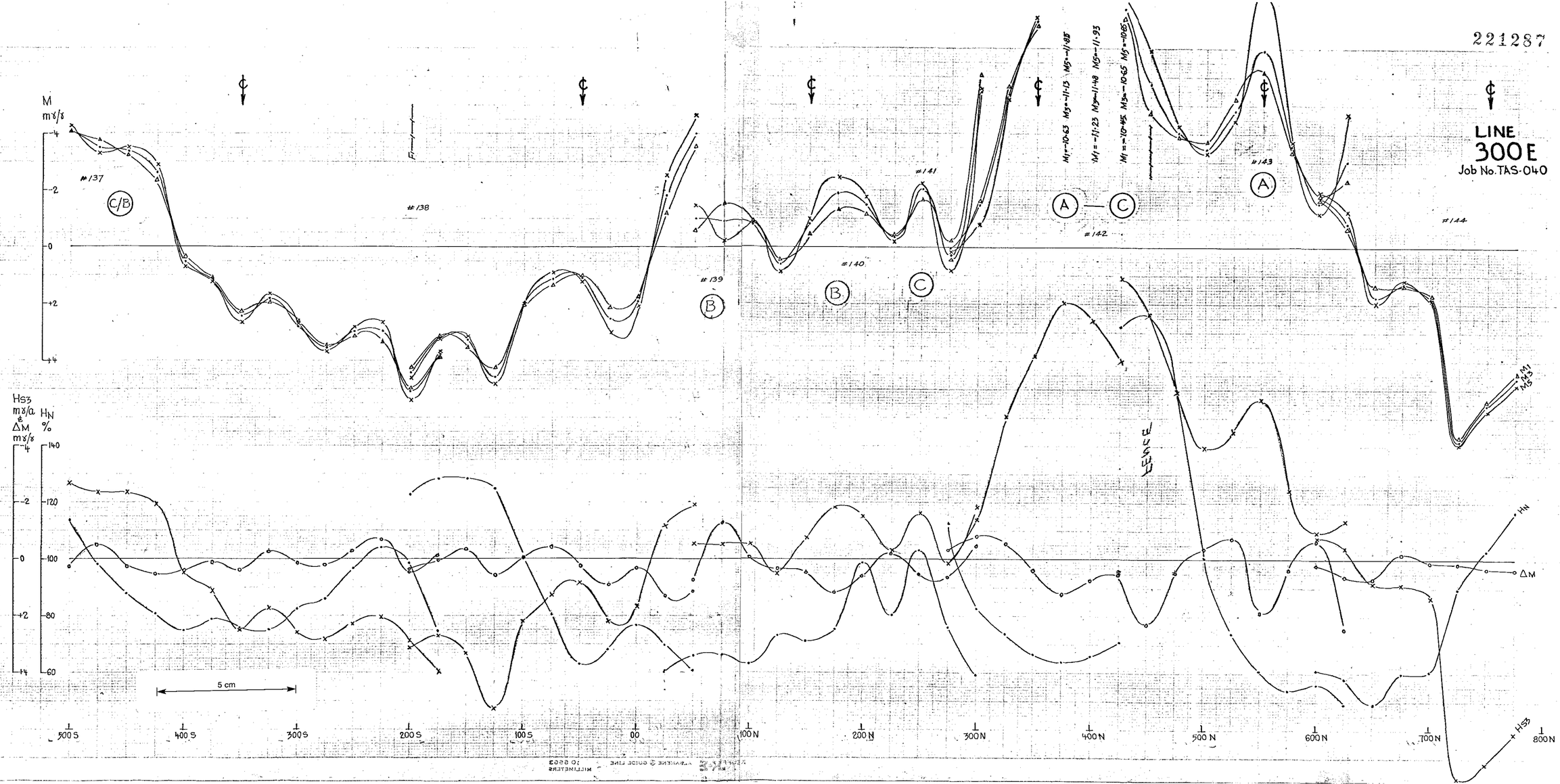
221286



LINE
200E
Job No. TAS-040
M
mγ/γ
-4
-2
0
+2
+4
B
#129
#130
#131
A/B
#132
B
#133
B?
#134
A?
#135
A
#128
M1
M3
M5
Hs3
mγ/a
&
ΔM
mγ/γ
HN
%
140
120
100
80
60
HN
Hs3
ΔM
5 cm
500S
400S
300S
200S
100S
00
100N
200N
300N
400N
500N
600N
700N
800N
900N

2412

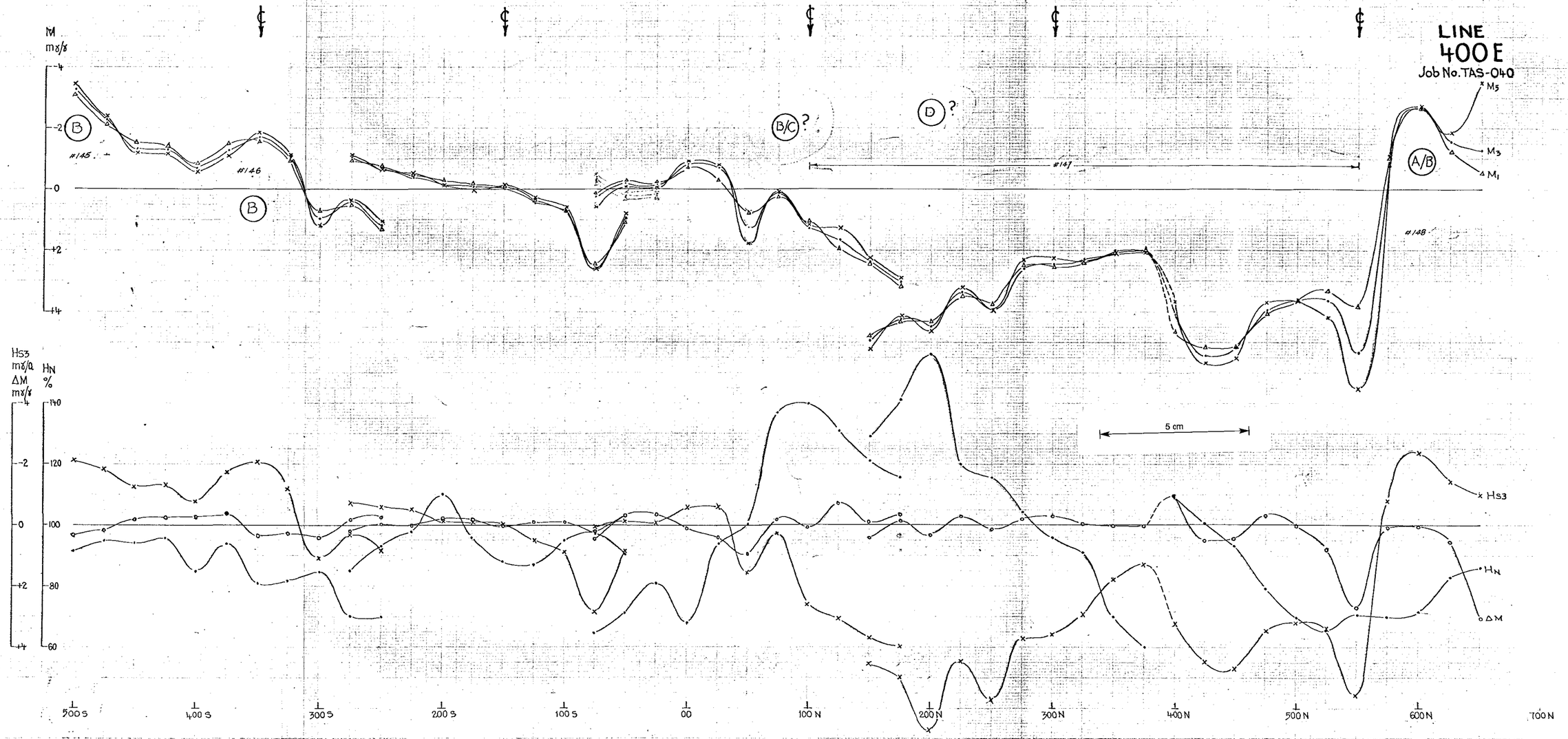
221287



LINE
300E
Job No. TAS-040
M
mγ/γ
#137
C/B
#138
#139
B
#140
B
#141
C
#142
A
C
#143
A
#144
M1=-10·63 M3=-11·13 M5=-11·85
M1=-11·23 M3=-11·48 M5=-11·93
M1=-10·45 M3=-10·65 M5=-10·65
Hs3
mγ/a
&
ΔM
mγ/γ
HN
%
5 cm
500 S
400 S
300 S
200 S
100 S
00
100 N
200 N
300 N
400 N
500 N
600 N
700 N
800 N
HN
ΔM
Hs3

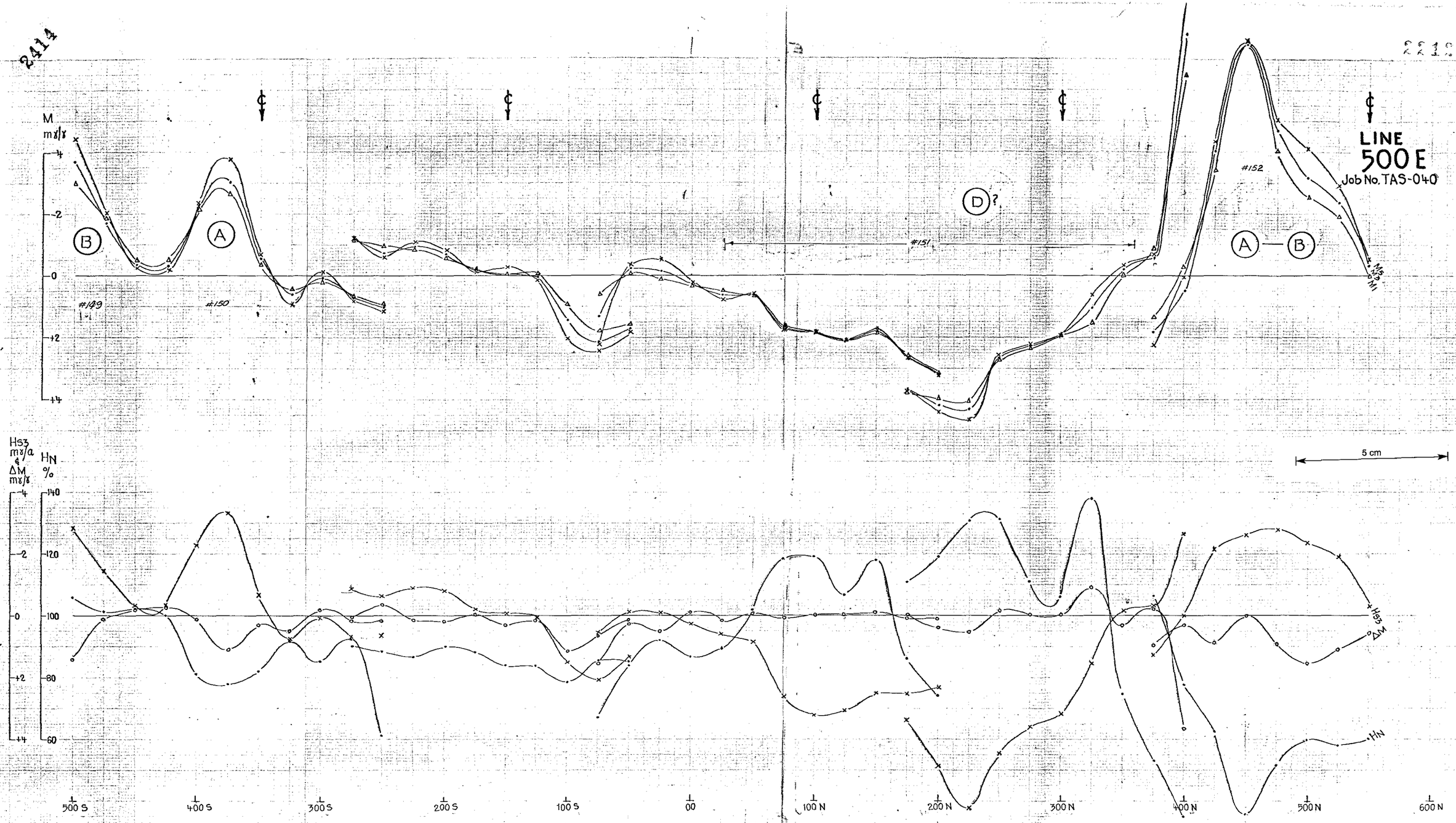
2413

221288


LINE
400E
Job No. TAS-040
M
mγ/γ
-4
-2
0
+2
+4
B
B
B/C ?
D ?
A/B
#145
#146
#147
#148
M3
M3
M1
5 cm
Hs3
mγ/γ
ΔM
mγ/γ
HN
%
-4
-2
0
+2
+4
140
120
100
80
60
Hs3
HN
ΔM
500 S
400 S
300 S
200 S
100 S
00
100 N
200 N
300 N
400 N
500 N
600 N
700 N

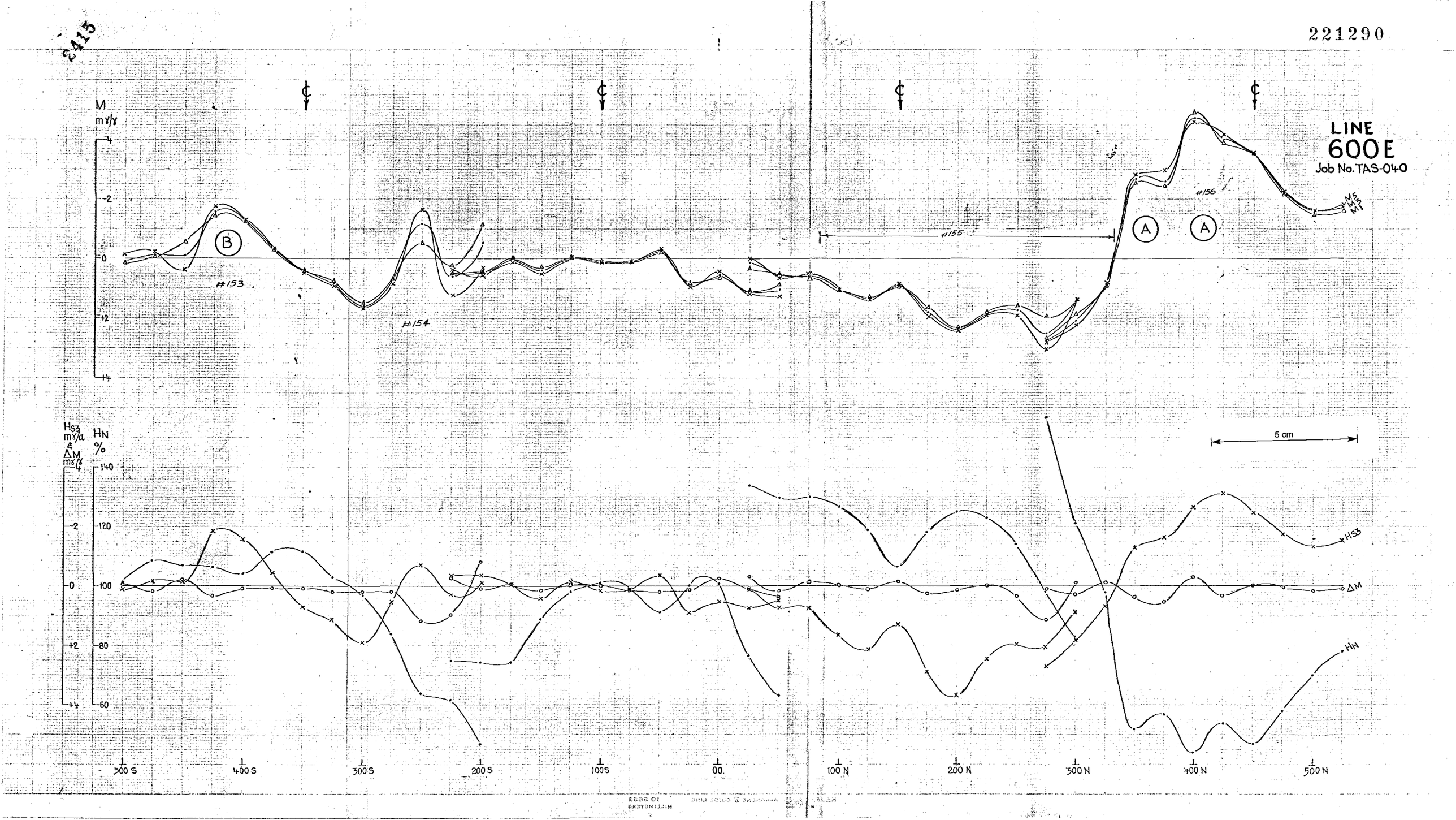
2414

221889


LINE
500 E
Job No. TAS-040
M
mγ/γ
+4
-2
0
+2
+4
B
A
D?
A — B
#149
#150
#151
#152
5 cm
Hs3
mγ/a
ΔM
mγ/γ
HN
%
-4
-2
0
+2
+4
140
120
100
80
60
Hs3 ΔM
HN
500 S
400 S
300 S
200 S
100 S
00
100 N
200 N
300 N
400 N
500 N
600 N

2415

221290

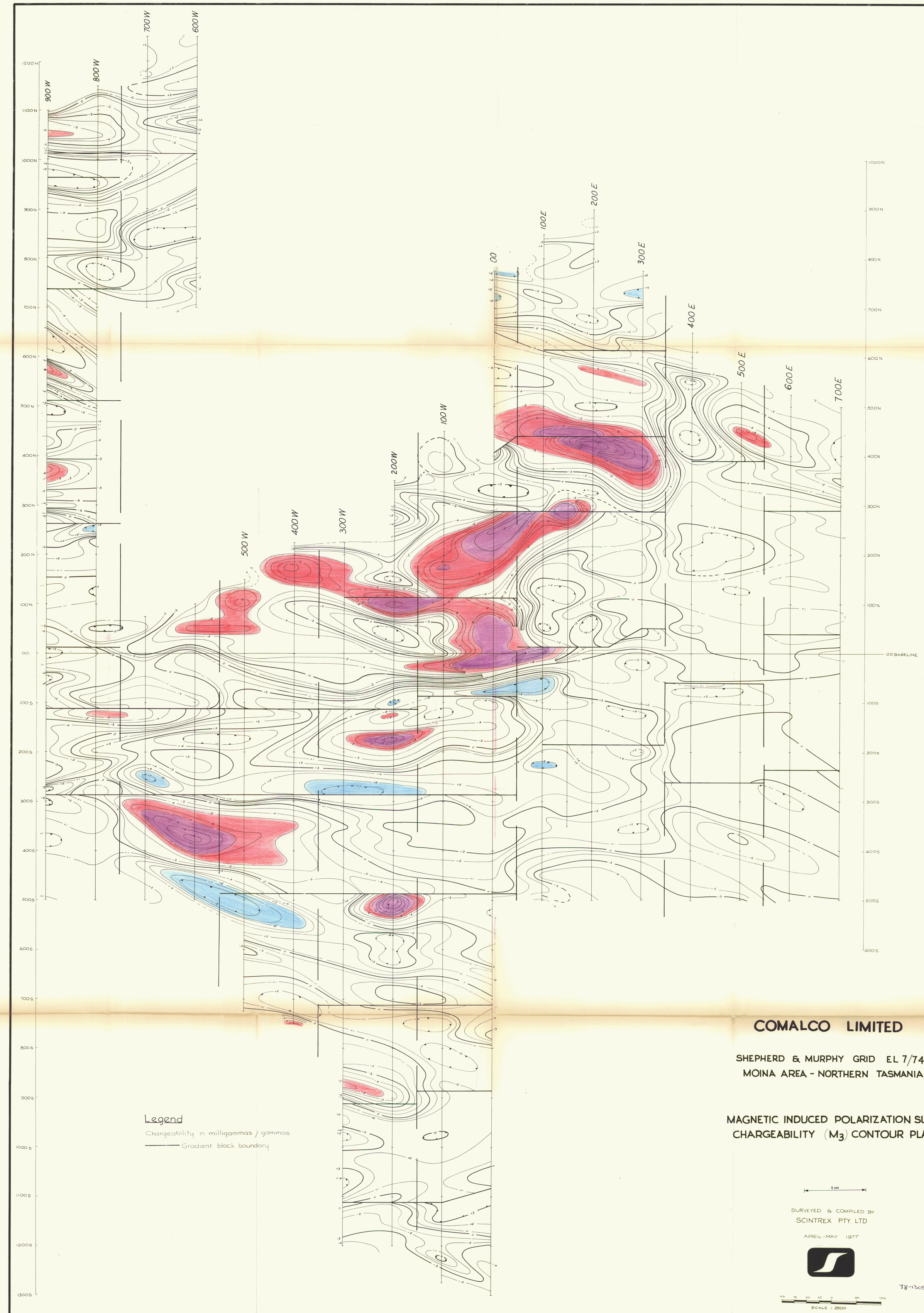

LINE
600E
Job No. TAS-040
M
mγ/γ
+4
-2
0
+2
+4
B
#153
#154
#155
#156
A
A
MS
MI
5 cm
HSS
mγ/a
&
ΔM
mγ/γ
HN
%
+4
-2
0
+2
+4
140
120
100
80
60
HSS
ΔM
HN
500 S
400 S
300 S
200 S
100 S
00
100 N
200 N
300 N
400 N
500 N

2416

221291



LINE
700E
Job No TAS-040
M
mγ/γ
-4
-2
0
+2
+4
HN%
140
120
100
80
60
Noisy Creek
A
#157
+M5
M3
M1
5 cm
HS3
ΔM
HN
500 S
400 S
300 S
200 S
100 S
00
100 N
200 N
300 N
400 N
500 N



Legend

Chargeability in milligammas / gammas

——— Gradient block boundary

COMALCO LIMITED

SHEPHERD & MURPHY GRID EL 7/74
MOINA AREA - NORTHERN TASMANIA

MAGNETIC INDUCED POLARIZATION SURVEY
CHARGEABILITY (M_3) CONTOUR PLAN

5 cm

SURVEYED & COMPILED BY
SCINTREX PTY. LTD

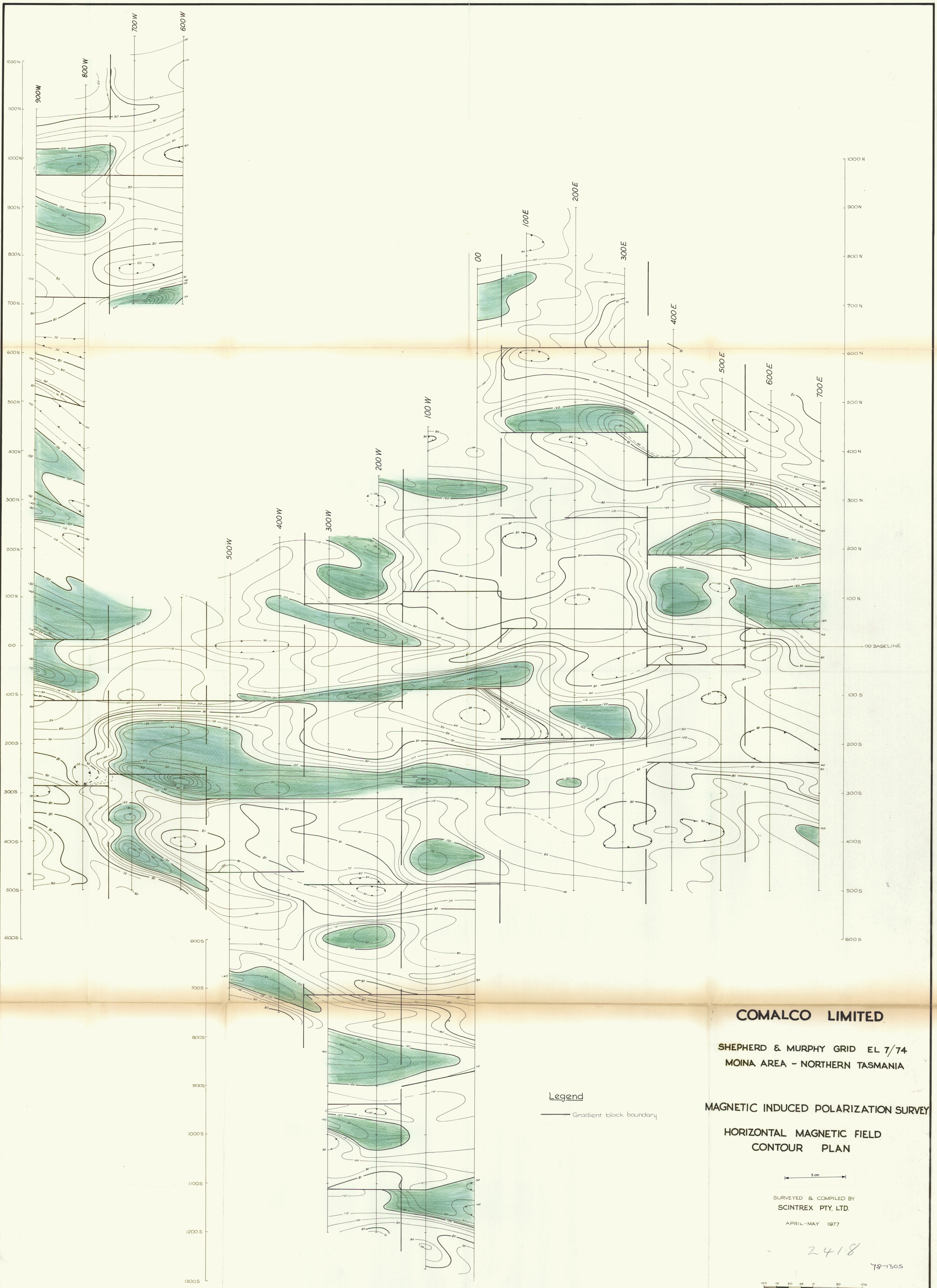
APRIL-MAY 1977

78-1305

100 75 50 25 0 50 100
SCALE 1:2500

Job Nº TAS-040 Sheet 1 of 1 PLATE 1.

2417



Legend

Gradient block boundary

COMALCO LIMITED

SHEPHERD & MURPHY GRID EL 7/74

MOINA AREA - NORTHERN TASMANIA

MAGNETIC INDUCED POLARIZATION SURVEY

HORIZONTAL MAGNETIC FIELD CONTOUR PLAN

5 cm

SURVEYED & COMPILED BY

SCINTREX PTY. LTD.

APRIL-MAY 1977

2418

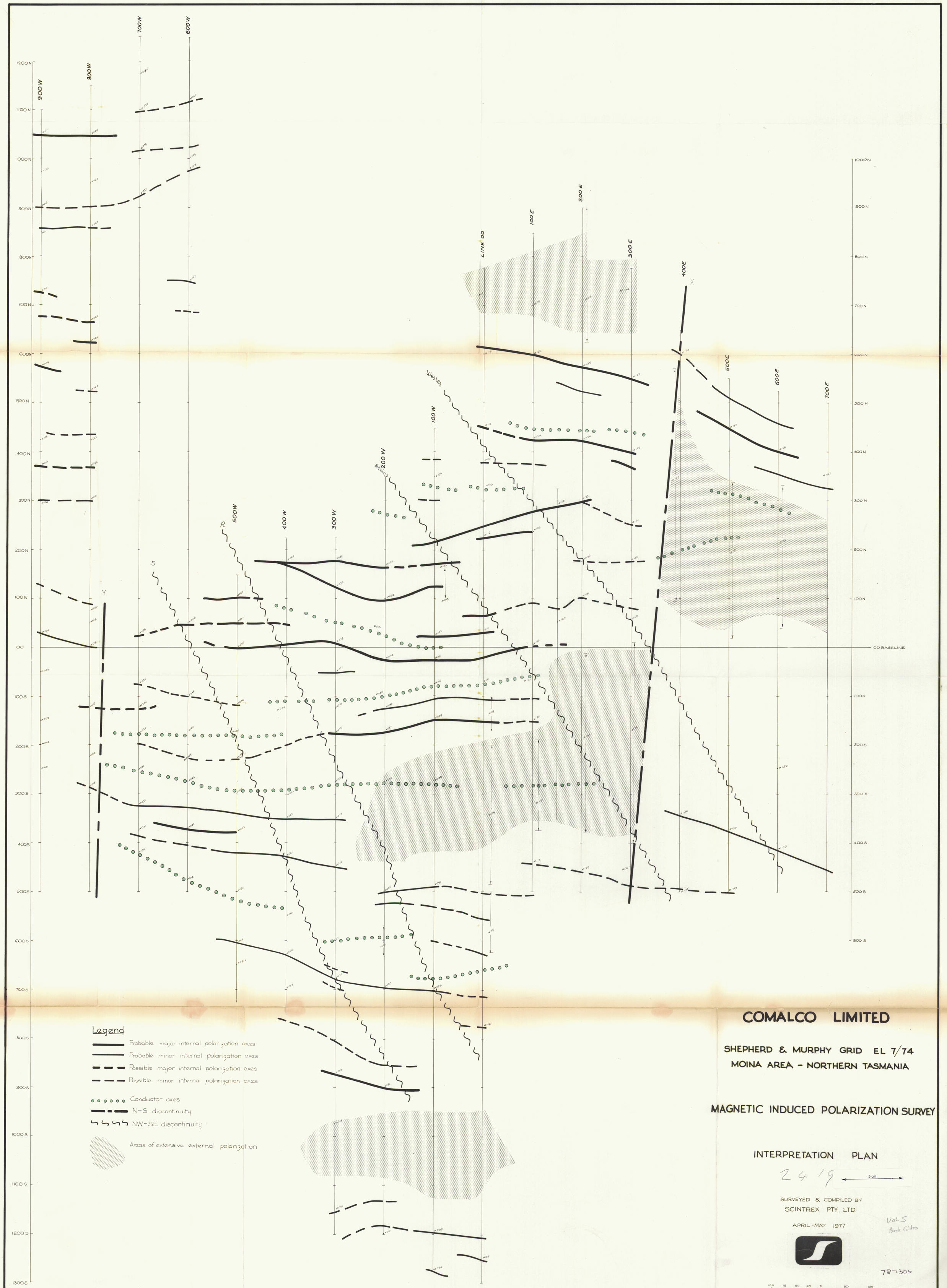
78-1305

100 75 50 25 0 50 100

SCALE 1:2500

Job N° TAS-040 Sheet 1 of 1 PLATE 2.

2418



COMALCO LIMITED

SHEPHERD & MURPHY GRID EL 7/74
MOINA AREA - NORTHERN TASMANIA

MAGNETIC INDUCED POLARIZATION SURVEY

INTERPRETATION PLAN

2419

5 cm

SURVEYED & COMPILED BY
SCINTREX PTY. LTD.

APRIL-MAY 1977

Vol 5
Back folders

78-1305

100 75 50 25 0 50 100
SCALE 1:2500

Job N° TAS-040 Sheet 1 of 1 PLATE 3

2419

APPENDIX 21

ELECTRICAL LOGGING OF DIAMOND DRILL HOLES - MOINA.

G. WESTE

GENERAL.

Time domain electrical induced polarisation (pole-dipole) logging was carried out down drill holes SMD 13, 15, 16, 21 and 22. The drill holes were logged as an aid to interpretation of surface gradient array induced polarisation, to provide information on the core rocks, and to look 'away' from the drill holes. The data was obtained as part of the search for massive pyrrhotite.

SMD 9 which intersected pyrrhotite rich skarn could not be logged because of a blockage down the hole.

Where possible the steel drill hole casing was removed and replaced with flexible plastic agricultural drainage pipe which has small slots which should enable electrical contact with surrounding rocks. Considerable difficulties were encountered in logging the holes. Very low resistivities and negative chargeabilities were often encountered within the plastic casing caused by either electromagnetic coupling or overcharging (even though currents were generally kept to 0.1 amp). Similar problems were encountered near the bottom of holes. In drill holes SMD 16, 21 and 22 erratic (mainly negative) and unstable chargeabilities rendered this data unplottable, although the resistivity was stable and was plotted.

A pole-dipole array was used to control the rock volume/distance from hole measured and avoids complicated geometric factors and earth-air interface problems. For simplicity a "Three Array" spacing (potential electrode spacing = potential to current electrode spacing) was used. Graphical plots of the data obtained for each hole are attached.

SMD 13.

Because of caving only the bottom section of the hole was logged. Coupling occurred near the bottom and instability rendered chargeability data almost useless.

Rock type	Resistivity (Ωm)	Chargeability (ms)	
siltstone and sandstone	200-500 (very low)	5?	(background)
		25	(where pyritic)

SMD 15.

Overcharging/coupling problems were encountered at the top (plastic casing) and the bottom of the drill hole. Resistivity data was

meaningful from 42 to 110 m and chargeability data from 42 to 102 m.

Rock type	Resistivity (Ωm)	Chargeability (ms)
wrigglite	250 - 500 600 - 800	8 - 15 (background) to 120 (where highly pyritic)
calc-silicate	800 - 3000	8 - 15

SMD 16.

Because of ? overcharging the chargeability data was not plottable (mainly negative). The resistivity data shows the effect of faulting (at 24 m) and strong facturing (24-27 m) on the conductivity. Again calc-silicate rocks appear to be more resistive than wrigglite. All resistivity data is considerably lower than anticipated. (Compare with SMD 15).

Rock type	Resistivity (Ωm)
metasiltstone	> 250
faulted, fractured sandstone & calc-silicate	30
calc-silicate	350 - 500
wrigglite	250 - 350

SMD 21.

In this hole the electrode spacings were varied to obtain information progressively away from the hole. A three array spacing was maintained whilst the potential electrode spacings were 2.5 m, 5.0 m and 10.0 m with corresponding reading intervals of 2.5 m, 5.0 m and 5.0 m.

?Overcharging rendered chargeabilities useless but good resistivity data was obtained over most of the drill hole. Resistivities were considerably higher than in SMD 13 and 16.

Rock type	Resistivity (Ωm)
limestone	6000 - 7000, <1000 (relatively conductive intervals)
calc-silicate and wrigglite	variable 500 - 1200
metasiltstone & sandstone	1500 - 3000

Variability of resistivities within the limestone sequence is too great to be due to variation of the lithology in the limestone. Most likely explanations are the nearby presence of a cave or strong fracturing of the rocks (not indicated in the drill logs).

Increasing the electrode spacing increased the volume of rock measured along and away from the drill hole. The main effect was to increase the width and conductivity of the low resistivity intervals in the limestone, increasing the probability of them being due to caves. If this is the cause then a cave occurs less than 2.5 m from the hole at 90m and deepens away from the hole to about 100 m at 10 m out.

SMD 22.

Cave material (wet clays and silts) in the top 53 m of the drill hole shows a marked resistivity contrast to the underlying limestones. Fracturing of rocks causes a sharp decrease in resistivity at 60 m.

Rock type	Resistivity (Ωm)
cave material	25 - 50
limestone	2500 - 16000
calc-silicate	300 - 500
metasiltstone & sandstone	1000 - 12000

CONCLUSIONS

1. Resistivity values obtained indicate that no massive pyrrhotite exists within 2.5 m of SMD 13, 15, 16 and 22 or within 10 m of SMD 21.
2. Low resistivities encountered in SMD 21 are due to water and saturated clays in limestone caves.
3. The most reliable resistivity of the rock types encountered are: -

limestone	3000 - 7000Ω m
metasiltstone & sandstone	1500 - 4000Ω m
calc-silicate	400 - 2000Ω m
wrigglite	250 - 800Ω m
clays & water	25 - 50Ω m

22:1295

DRILL HOLE SMD 13 - DOWN HOLE E.I.P. SURVEY

Pole-dipole a = 2·5m, n = 1

Instrument - time domain - 2·5 Kw. transmitter

- Scintrex IPR 7 receiv



Chargeability

Resistivity

5 cm

Resistivity in ohm metre

10,000 (Ω)
5000
3000
2000
1000
700
600
500
400
300
200
100
50
30
20
10

hole caved

SILTSTONE & SANDSTONE

Depth down hole (m)

40 50 60 70 80 90 100 110 120 130 140 150 160 170

Chargeability in Milliseconds

221296

DRILL HOLE SMD 15 - DOWN HOLE E.I.P. SURVEY

Pole-dipole a = 2.5m, n = 1

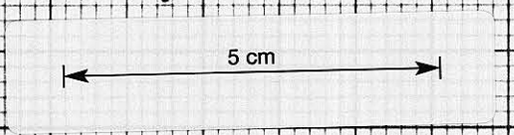
Instrument-time domain - 2.5Kw transmitter

- Scintrex IPR 7 receiv



Chargeability

Resistivity



Chargeability in Milliseconds

Apparent Resistivity in ohm metre

Depth down hole (m)

coupling or overcharging

unstable

coupling or overcharging

CLAY & SANDY SILT

CALC SILICATE & WRIGGLITE

WRIGGLITE

CALC SILICATE

WRIGGLITE

CALC SILICATE

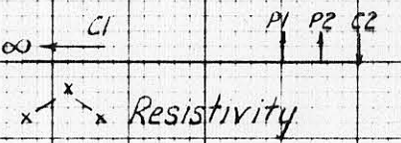
METASILTSTONE

DRILL HOLE SMD 16 - DOWN HOLE EIP SURVEY

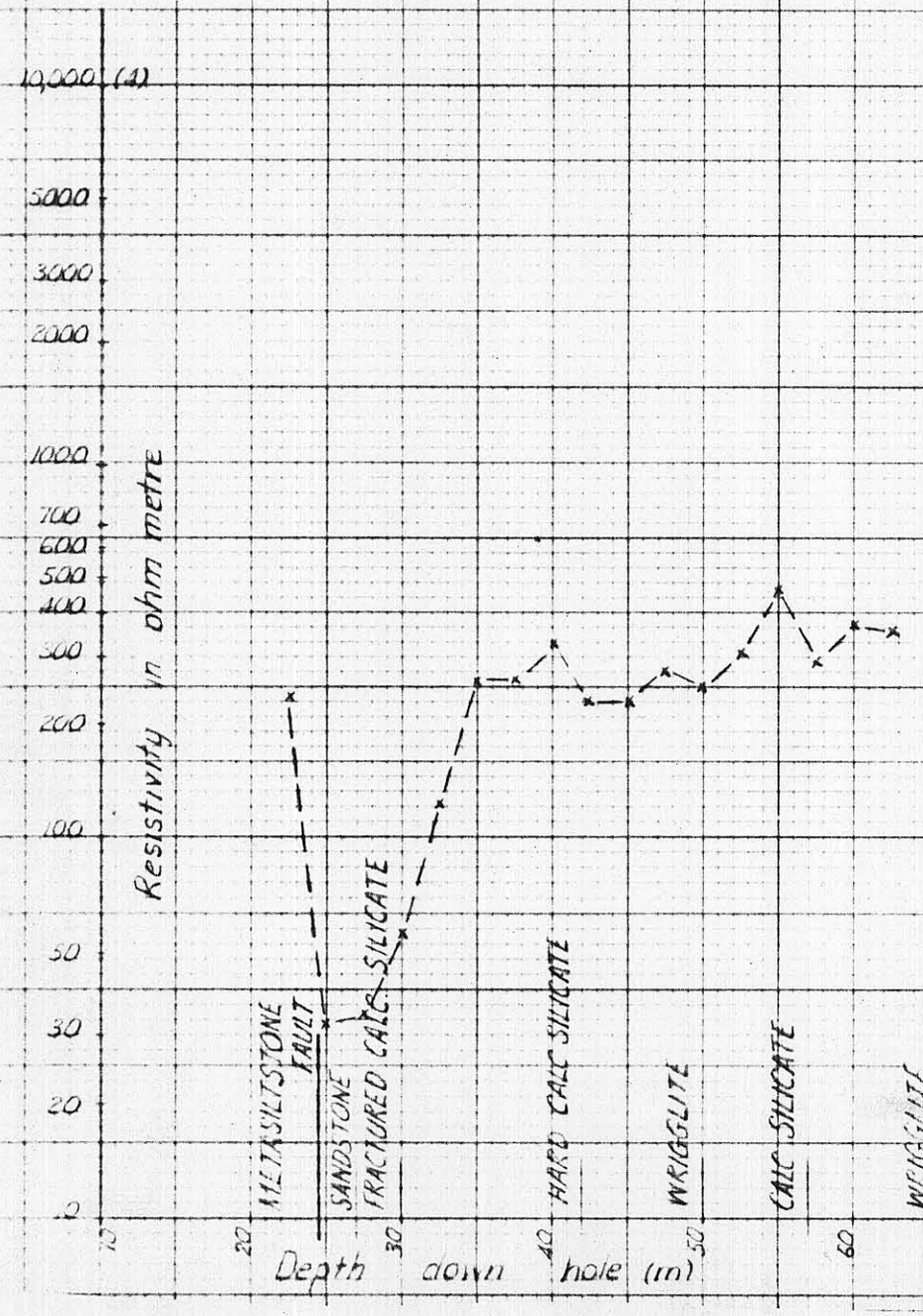
Pole-dipole $a = 2.5m$, $n = 1$

Instrument - time domain - 2.5 Kw. transmitter

- Scintrex IPR7 receiver



5 cm



221298

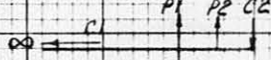
5 cm

DRILL HOLE SMD 21 DOWN HOLE RESISTIVITY SURVEY

Pole - Dipole a = 2.5m, n = 1

Instrument - Time domain - 2.5 Kw. transmitter

- Scintrex I.P.R. 7 receiver



RESISTIVITY in ohm metres

LIMESTONE

CAVE ?

LIMESTONE

CALC SILICATE

CALC SILICATE & AKPOGLYTE

METASILTSTONE & SANDSTONE

221299

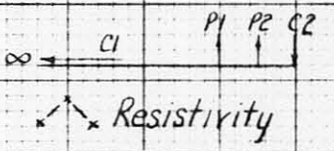
5 cm

HOLE SMD 21 - DOWN HOLE RESISTIVITY SURVE

Pole - Dipole a = 5·0m, n = 1

Instrument - Time domain - 2·5 Kw transmitter

- Scintrex I.P.R 7 recei



221300

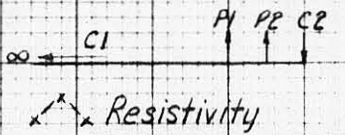
5 cm

DRILL HOLE SMD 21 - DOWN HOLE RESISTIVITY SURVEY

Pole - Dipole a = 10·0m, n = 1

Instrument - Time domain - 2·5 Kw. transmitter

- Scintrex IPR 7 receiver



Resistivity in ohm metres

10000
5000
2000
1000
700
500
300
200
100
50
30
20
10

30 40 50 60 70 80 90 100 110 120 130 140 150

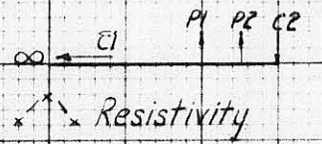
depth down hole (m)

DRILL HOLE SMD 22 - DOWN HOLE RESISTIVITY SURVEY

Pole - Dipole a = 2.5m, n = 1

Instrument - Time domain - 2.5 Kw transmitter

- Scintrex I.P.R. 7 receiv



206

OS. 1144

DENSITY OF VARIOUS ROCK TYPES, MOINA.

DEPARTMENT OF MINES—TASMANIA

221302

TELEPHONES:
Metallurgical Research Laboratory .. .. .. 44 2431-2 (2 lines)
Mines Inspection .. .. ..

APPENDIX 23

SEISMIC AND RESISTIVITY TRIALS, MOINA.

221303

COMALCO LIMITED

COMALCO

MEMORANDUM

Ref P. W. Askins

To G. Weste
G. Shovelton

At Devonport

Copies Cairns Office A.H.W./S.K.C.
Melbourne " P.W.S./I.D.P.

Date 14th February 1977

From A. H. Bartlett

At Melbourne

Subject Investigations with various geophysical techniques at E.L. 7/74 "Moina", Tasmania.

Introduction

Diamond drilling is an essential, but slow and costly, method of investigating the configuration and sampling of mineral deposits.

Investigations were carried out with seismic and resistivity equipment at E.L. 7/74 "Moina", Tasmania, to determine whether or not one or more of these geophysical techniques could be used in conjunction with drilling to outline the configuration of such a mineral deposit.

Conclusions

1. Resistivity useful to depths of about 30 metres.

2. Refraction seismic traverses limited to depths 50 metres and cannot solve problems where softer layers underlie harder ones.

3. Reflection seismic traverses with the "Bison" seismograph appear to be remarkably efficient and accurate in determining the depths of various strata.

Geophysical techniques used:

* Location

	D.O.M. 1	S.M.D. 10	S.M.D. 12	S.M.D. 15	ALONG BASELINE 600-800 m. East
Resistivity - expanding electrode - Wenner configuration	*		*		
Refraction Seismic "GT2B"			*		*
Refraction Seismic - "Bison"		*	*	*	*
Reflection Seismic - "Bison"		*	*	*	*

210

221304

Resistivity

Expanding (probing) electrode method with "Wenner" configuration using "YEW resistivity meter.

C_1 P_1 P_2 C_2

1 a 1 a 1 a 1 —— electrodes

Two "probes" were completed over bores D.O.M. 1 and S.M.D. 12. While there is excellent correlation between the bores and resistivity in these cases, there are two disadvantages with this method of resistivity probing -

(i) Probing is limited to about 50 metres depth.

(ii) The "traverse" length needs about 200 metres of ground with fairly uniform surface and underlying strata.

Refraction Seismic GT2B 12 channel

Two traverses were completed along the baseline between 600 and 800 metres east and over bore S.M.D. 12 using this equipment.

The equipment works well indicating -

0 to 5-7 metres	Weathered basalt
below 5-7 metres	Hard basalt
	8000 - 9000 feet / sec.

However, the refraction seismic method does NOT detect the "deep lead" sand and gravel layer beneath the basalt and is in consequence of little real value in solving this particular multi layer problem at Moina.

"Bison" signal enhancement seismograph

Refraction and reflection methods.

The "Bison" works well in the refraction mode down to depths of about 30 metres, but as with all refraction surveys it cannot solve problems where softer layers (deep lead) underlie harder layers (basalt).

To carry out a REFLECTION seismic survey it is first essential to carry out a REFRACTION survey, to determine the seismic wave velocities of as many of the top strata overlying the horizons whose depth are to be determined. For example, from the refraction traverse along baseline, we obtained:-

V_1 -	Topsoil	0 - 9'	1100 feet / sec.
V_2 -	Weathered basalt	9'-27.5'	4350 " "
V_3 -	Basalt	27.5'+	*7450 " "

(* GT2B 12 channel gave 8000 feet / sec. for the basalt)

cont'd .....

221305

The refraction survey does not give the base of the basalt nor the top of the next major strata of wrigglite or limestone which probably has gravel or deep lead material directly above.

Reflection method used with Bison:-

1. Two geophones were wired in series and connected to one of geophone intake sockets.

2. The geophones were placed either 40 or 50 feet apart equidistant from the bore or point whose strata depths were to be determined.

3. Hammer blows were struck at several points between the two geophones. Two or three blows were necessary to enhance the reflection signals.

4. Graphical plots were made of all enhanced signal times on the "Bison's" C.R.O.. True reflected times showed up on the graph as a series of points in an almost horizontal line.

5. After selection of the prominent reflected times, depths to the various strata boundaries can be computed as shown in the examples. Reflection times must be divided by two since they are the times the shock wave takes to travel down to the layer and back up to the geophone. Due allowance must be made for the depths of topsoil and weathered rock, also the seismic wave velocities of all strata must be carefully and systematically applied.

6. Refer to attached notes:-

"Reflection Seismology with the Bison Seismograph". (not attached with this appendix

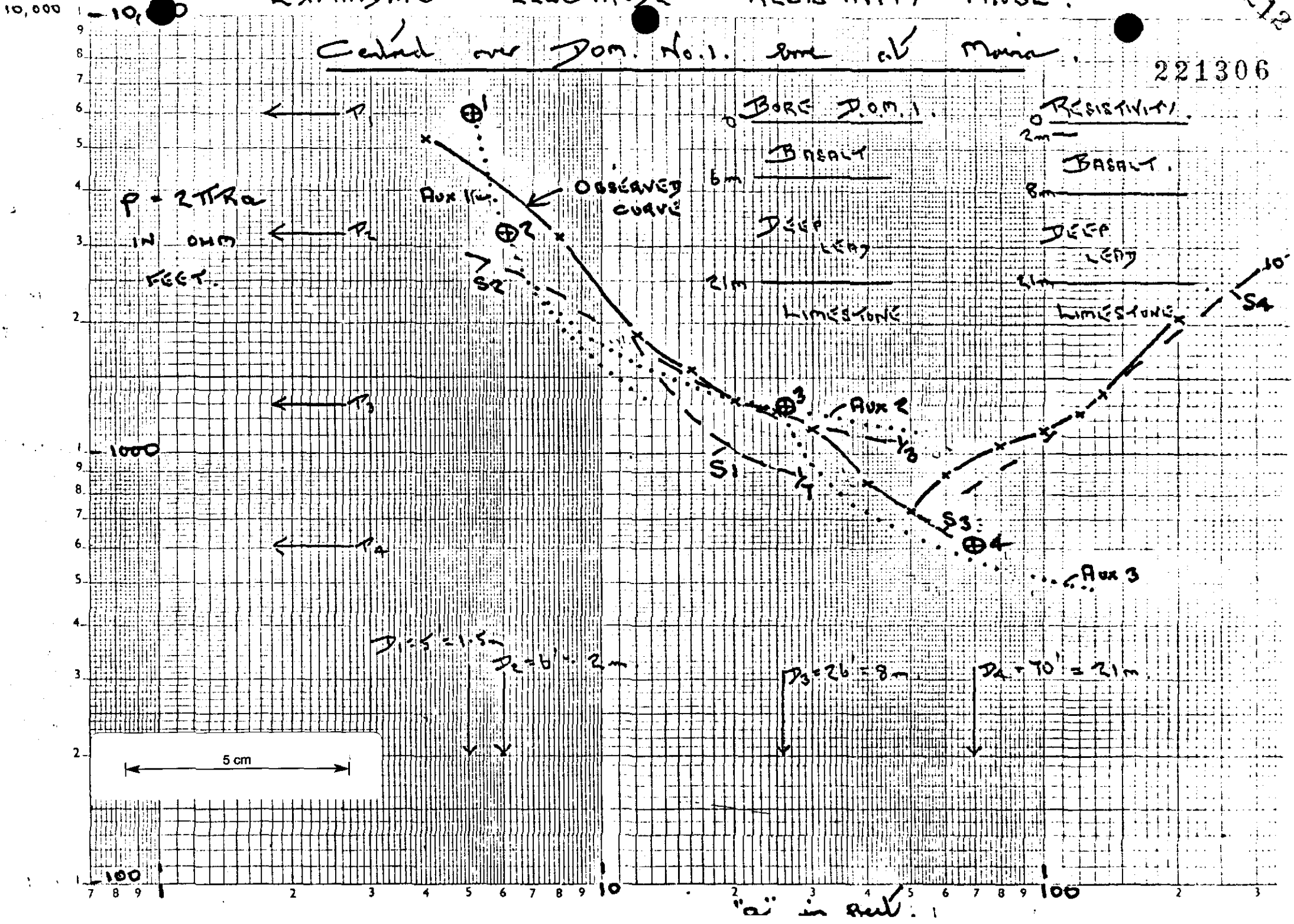
A. H. Bartlett.

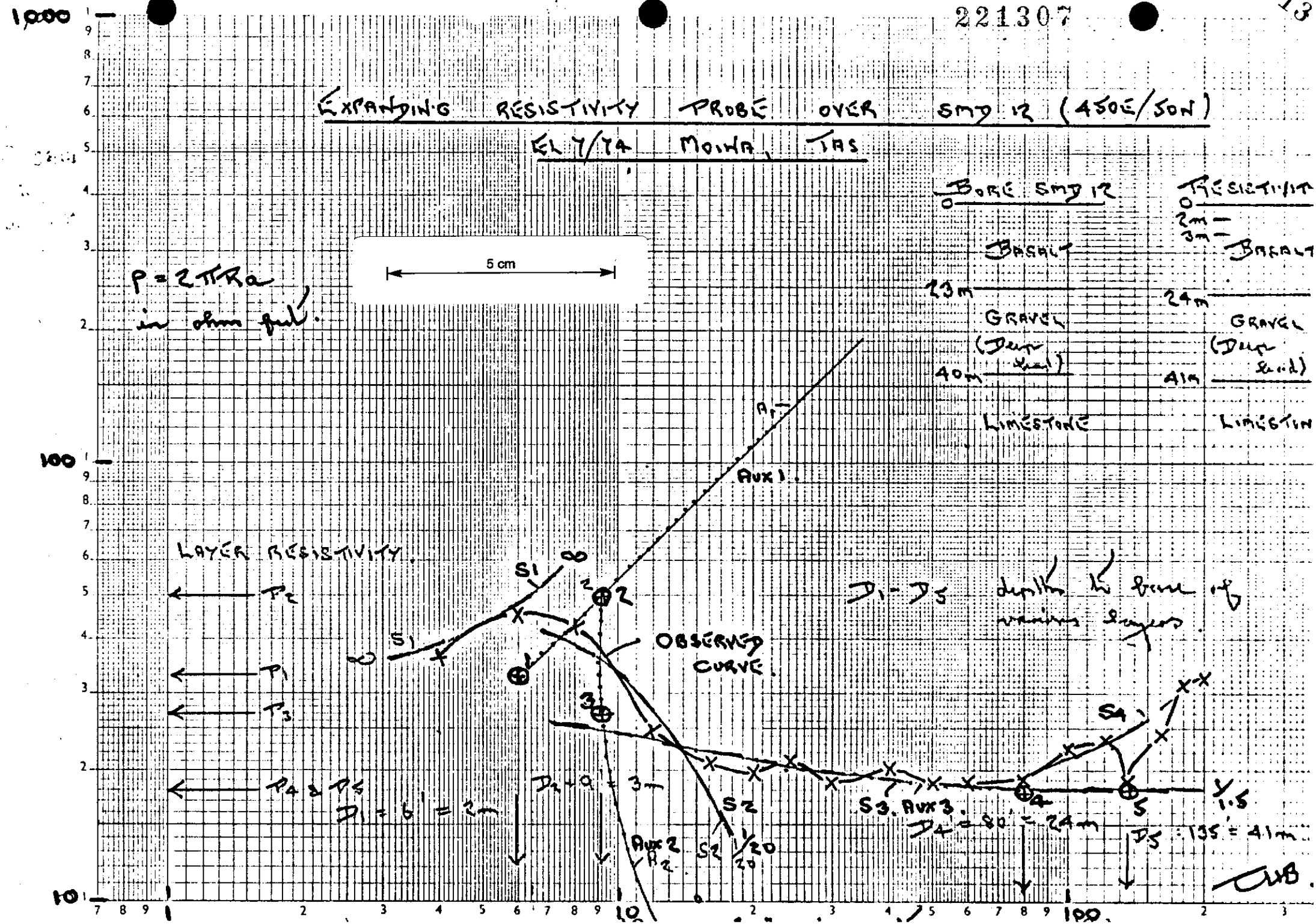
Enclosures
JMS

EXPANDING ELECTRODE RESISTIVITY PROBE.

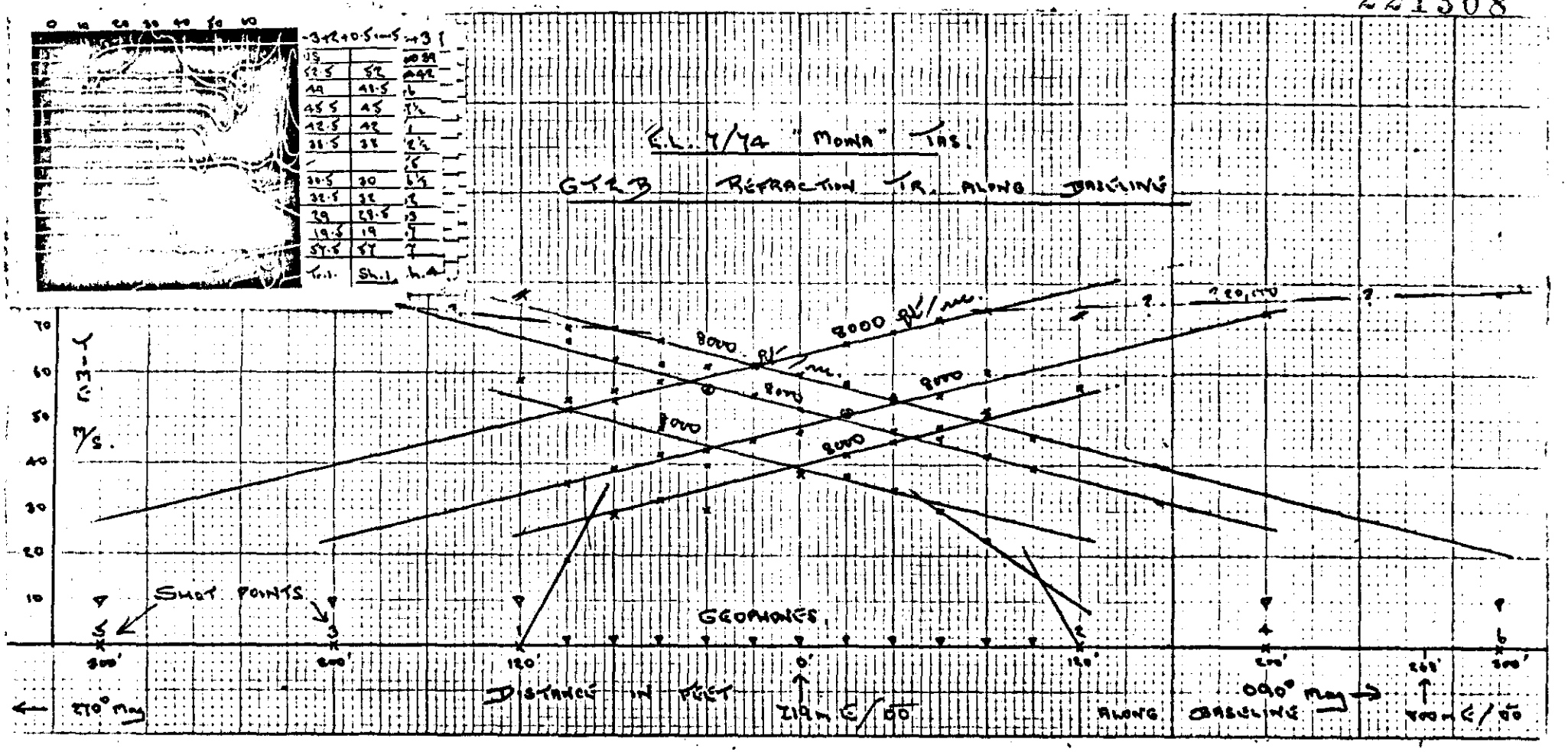
Centred over Dom. No. 1. bore at Moina.

221306




EXPANDING RESISTIVITY PROBE OVER SMD 12 (450E/50N)
EL 7/74 MOINA, TAS
BORE SMD 12
RESISTIVITY
BASALT
GRAVEL
(Deep head)
LIMESTONE
23m
24m
40m
41m
2m
3m
5 cm
ρ = 2πRa
in ohm feet.
LAYER RESISTIVITY.
P2
P1
P3
P4 & P5
D1 = 6' = 2m
D2 = 9' = 3m
D1 - D5 depths to base of various layers.
OBSERVED CURVE.
AUX 1
AUX 2
AUX 3
S1
S2
S3
S4
D4 = 80' = 24m
D5 = 135' = 41m
1/20
1/1.5
1,000
100
10
1
10
100

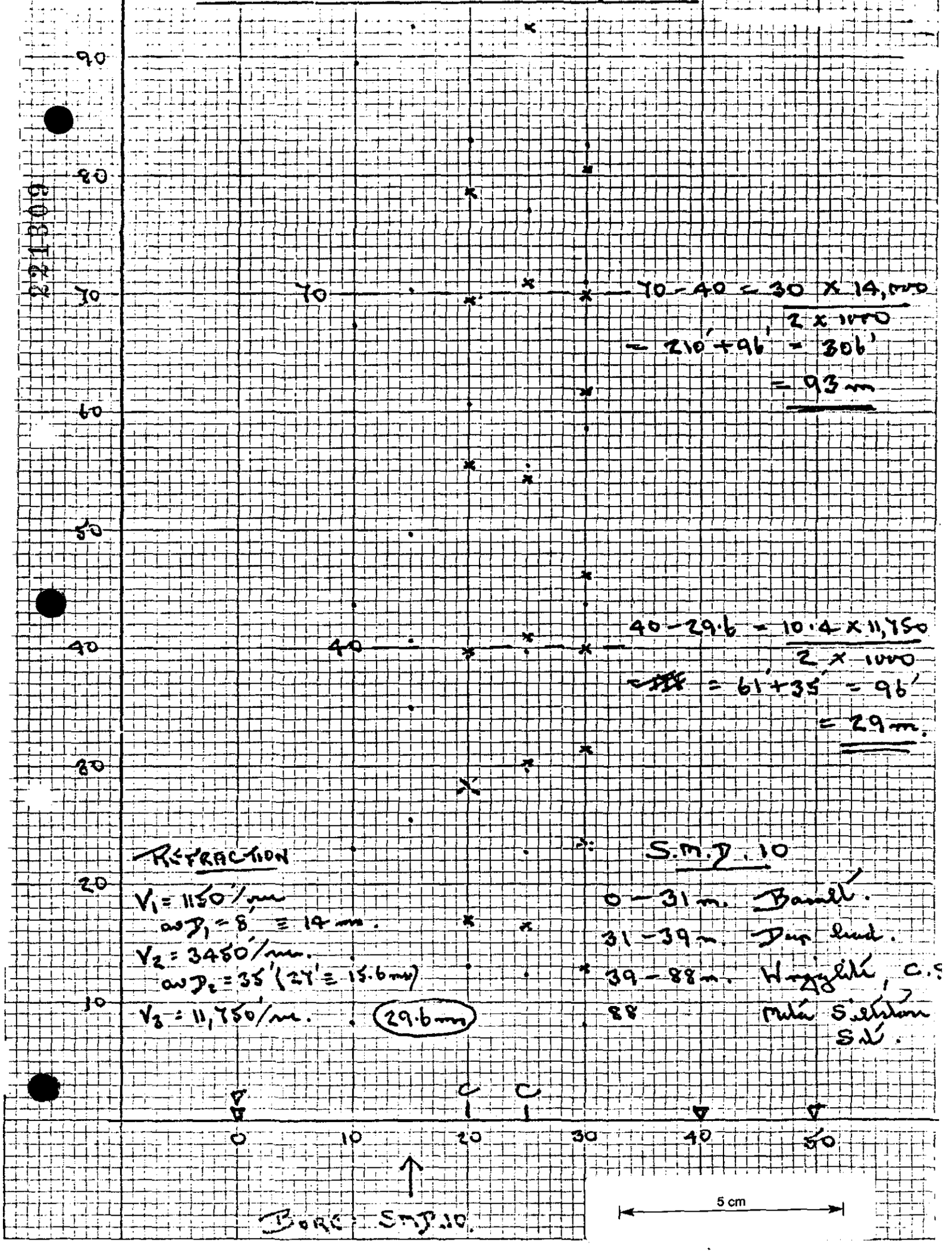
221308


E.L. 7/74 "MOINA" TAS.
GT2B REFRACTION TR. ALONG BASELINE
8000 ft/sec
8000 ft/sec
8000
8000
8000
8000
Y-ME m/s.
70
60
50
40
30
20
10
SHOT POINTS
GEOPHONES
300'
200'
120'
0'
120'
200'
300'
DISTANCE IN FEET
270° Mag
ALONG BASELINE
090° Mag

5 cm

221309

REFLECTIONS OVER S.M.D. 10.



$70 - 40 = \frac{30 \times 14{,}000}{2 \times 1000}$

$= 210' + 96' = 306'$

$= 93\,m$

$40 - 29.6 = \frac{10.4 \times 11{,}750}{2 \times 1000}$

$= 61' + 35' = 96'$

$= 29\,m.$

REFRACTION

$V_1 = 1150'/sec$

$av\,D_1 = 8' \equiv 14\,ms.$

$V_2 = 3450'/sec.$

$av\,D_2 = 35'\ (27' \equiv 15.6\,ms)$

$V_3 = 11{,}750'/sec.$ (29.6 ms)

S.M.D. 10

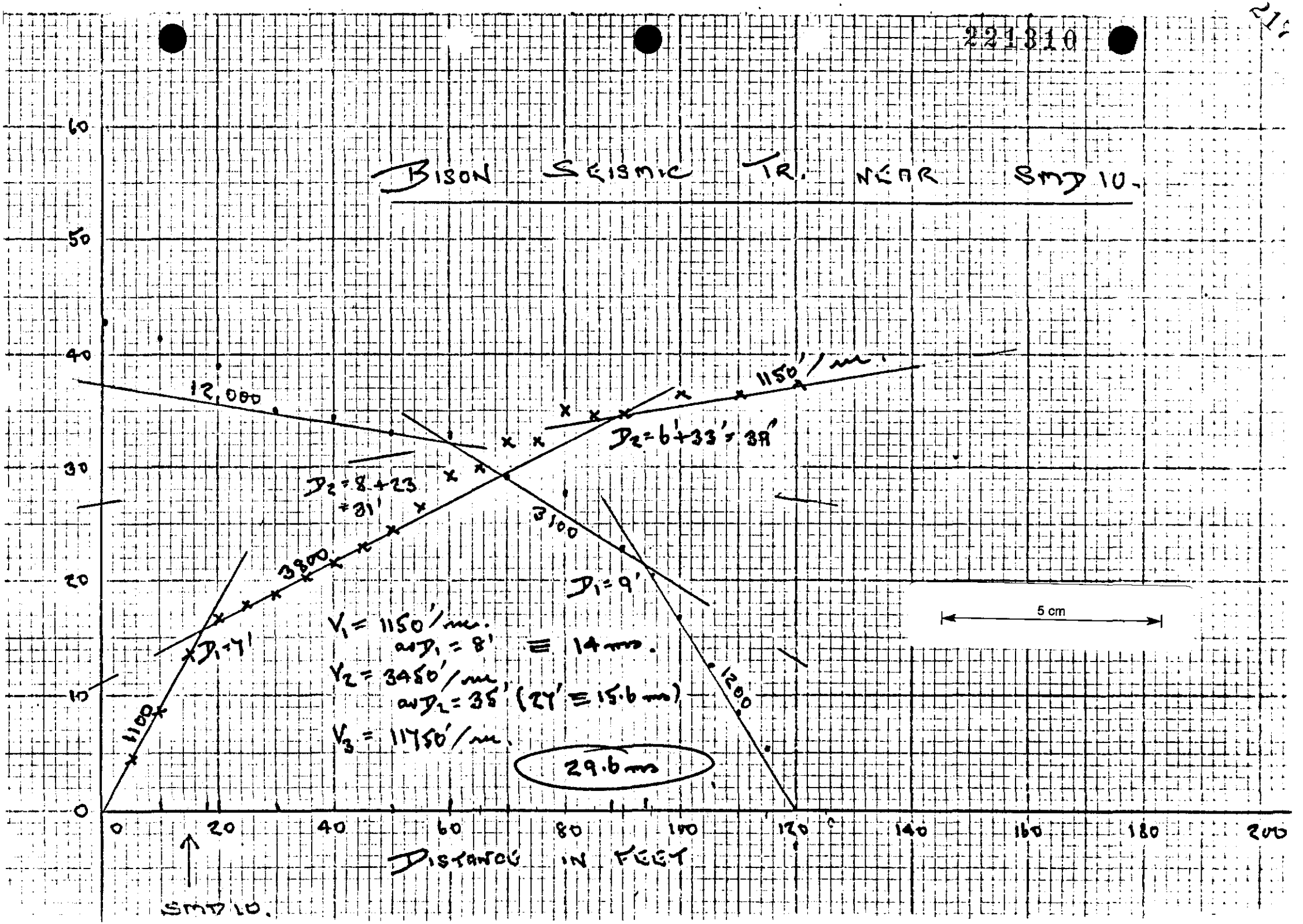
0 – 31 m. Basalt.

31 – 39 m. Dup lead.

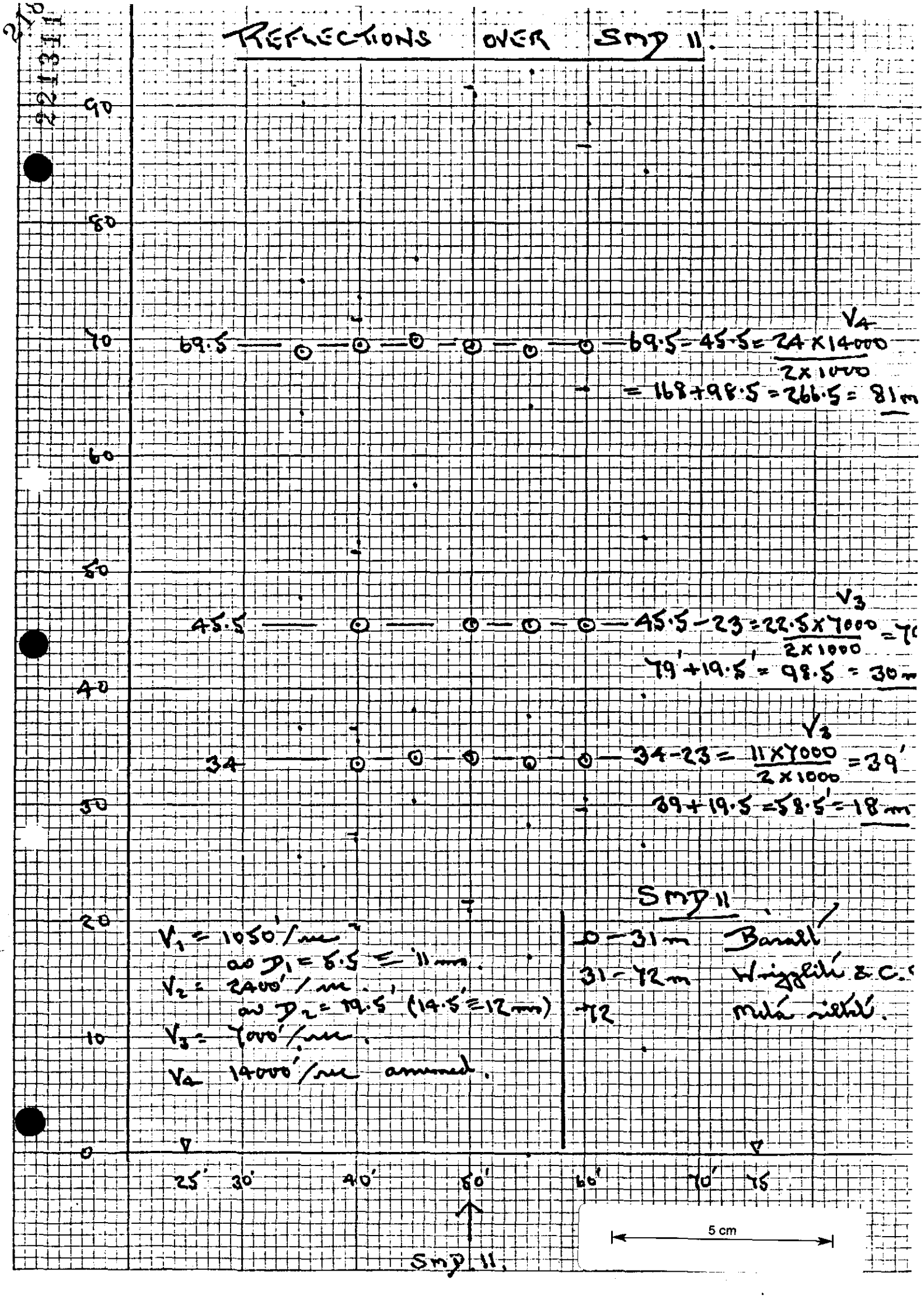
39 – 88 m. Wrigglite, C.S

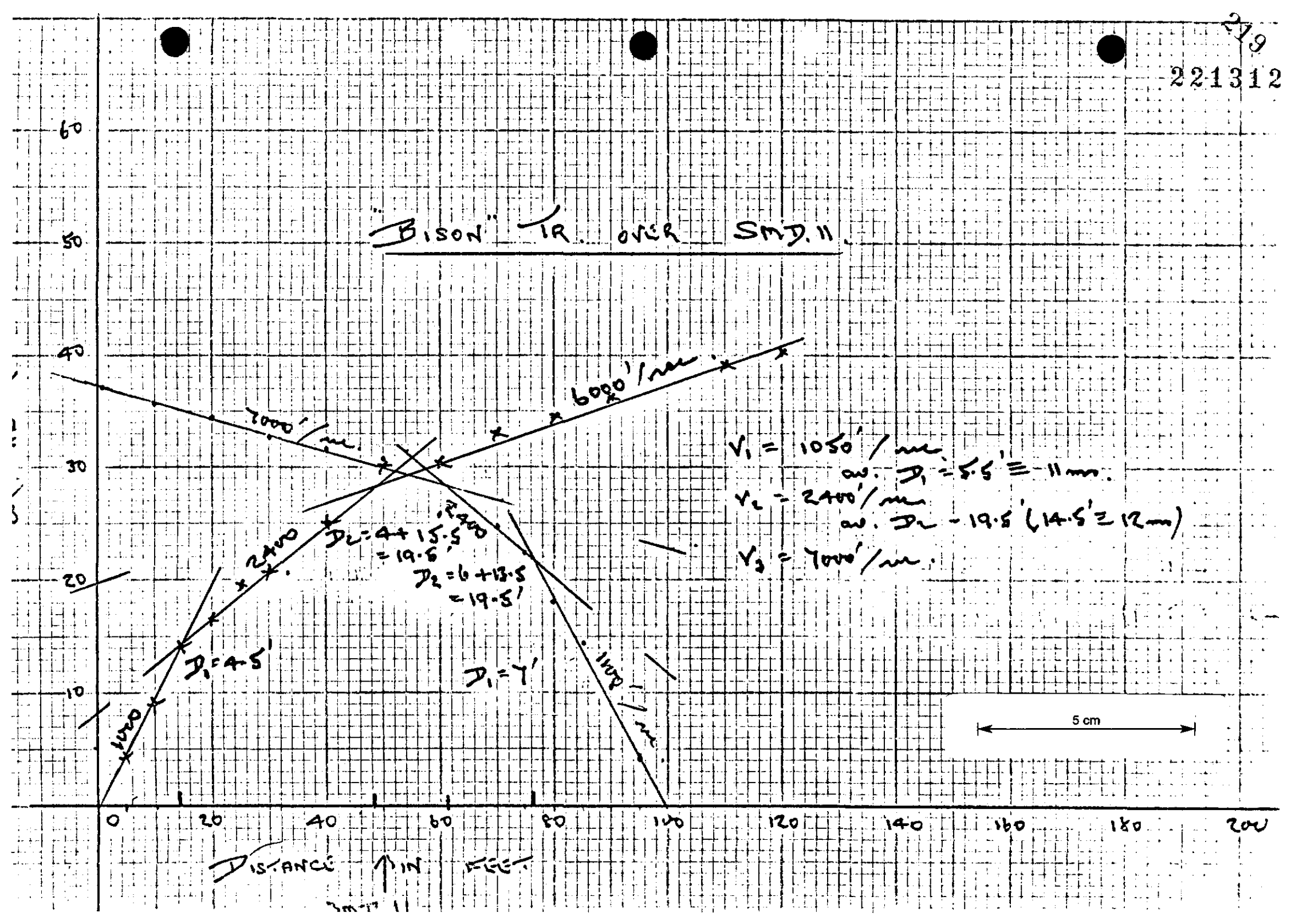
88 Mica Siltstone Sst.

BISON SEISMIC TR. NEAR SMD 10.

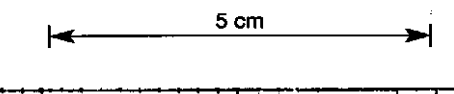


REFLECTIONS OVER SMP 11.




"BISON" TR. OVER SMD. 11
60
50
40
30
20
10
0
20
40
60
80
100
120
140
160
180
200
DISTANCE ↑ IN FEET
6000'/sec
7000'/sec
2400
2400
1000
1100'/sec
$D_2 = 4 + 15.5 = 19.5'$
$D_2 = 6 + 13.5 = 19.5'$
$D_1 = 4.5'$
$D_1 = 7'$
$V_1 = 1050'/sec$
av. $D_1 = 5.5' \equiv 11$ ms.
$V_2 = 2400'/sec$
av. $D_2 = 19.5'$ ($14.5' \equiv 12$ ms)
$V_3 = 7000'/sec$
5 cm

REFLECTIONS ON SMD. 12.



71 — ⊗ — ⊗ — ⊗ ← 71-51 = 20 × 14,500 / 2 × 1000

= 145 + 79' = 224' = 68 milli

51 — ⊗ — ⊗ — ⊗ ← 51-27 = 24 × 3700 / 2 × 1000

= 444 + 34·5 + 79' = 24 mi

40·5 — ⊗ — ⊗ — ⊗ ← 40·5-27 = 13·5 × 3700 / 2 × 1000

= 25 + 34·5 = 18 milli

SMD. 12

0-23 m. Basalt

23-40·5 m. Sand, gravel, hal. C.S. mula S.

40·5-110 m. mula Silicates

110-123·25 m. End of hole

V_1 = 1100'/sec. at T_1 = 6·5

12 ms

V_2 = at 3700'/sec. at T_2 = 34·5'

15 ms

V_3 = at. 14,500'/sec

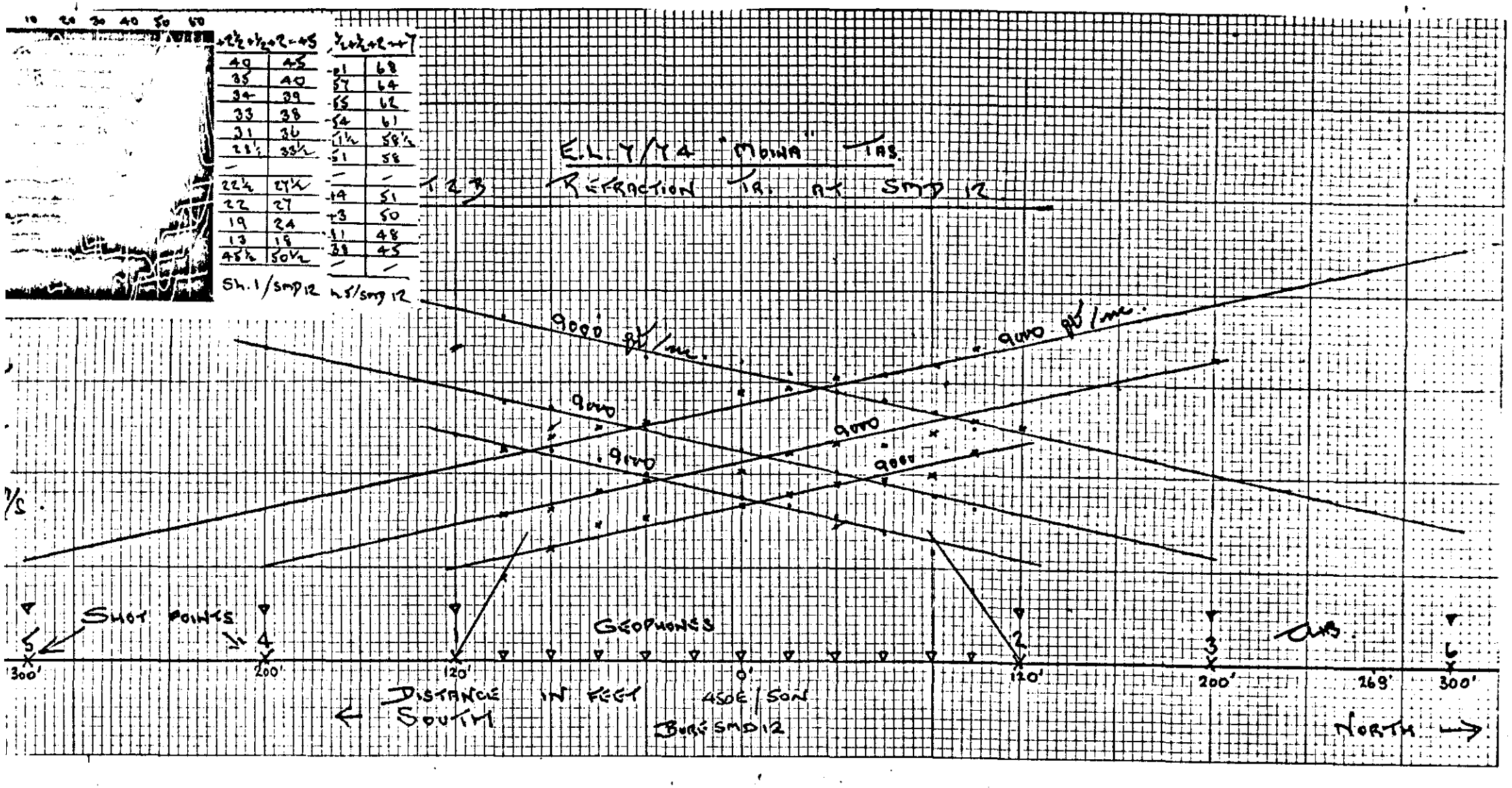
G

G

90 80 70 60 50 40 30 20 10 0

30 20 10 0 10 20 30

221314



+2½+½+2=+5	
40	45
35	40
34	39
33	38
31	36
28½	33½
—	
22½	27½
22	27
19	24
13	18
45½	50½

Sh. 1/SMD12

½+½+2=+7	
61	68
57	64
55	62
54	61
51½	58½
51	58
—	—
44	51
43	50
41	48
38	45
—	—

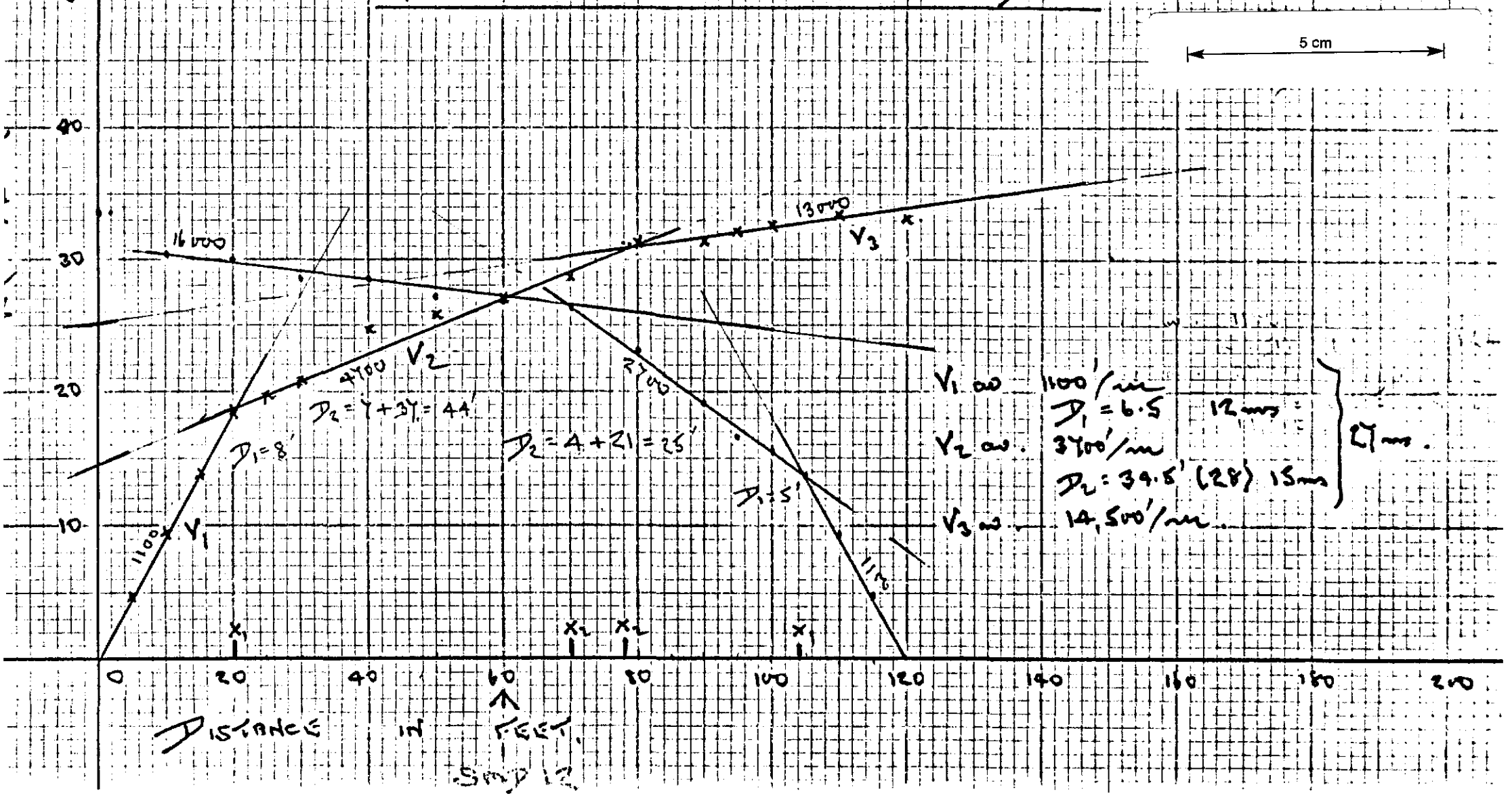
h.5/SMD12

5 cm

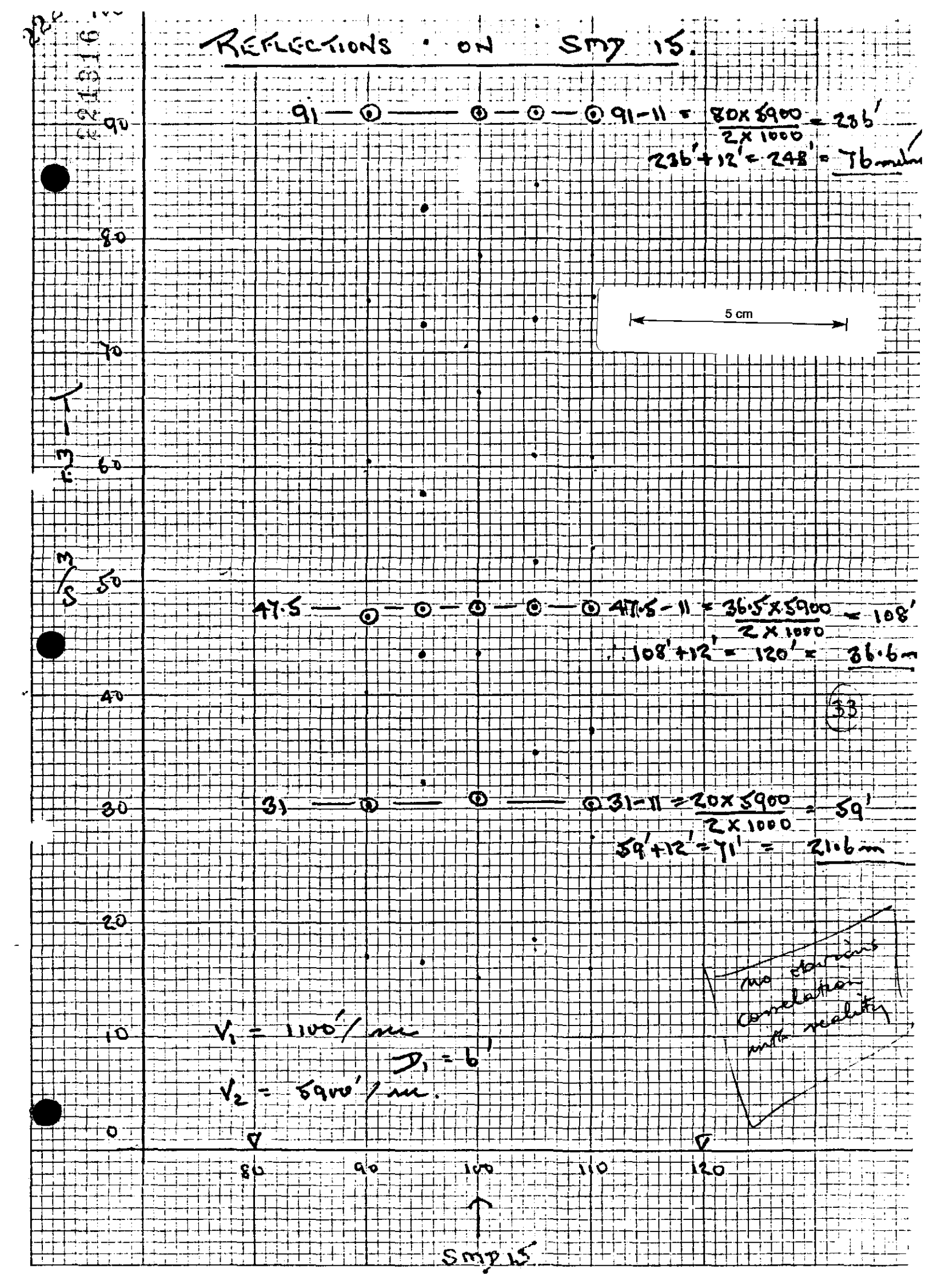
221315

$$D_1 = \frac{X_1}{2}\sqrt{\frac{V_2 - V_1}{V_2 + V_1}} \qquad D_2 = \frac{5}{6}D_1 + \frac{X_2}{2}\sqrt{\frac{V_3 - V_2}{V_3 + V_2}}$$

REFRACTION TR. ACROSS SMD 12



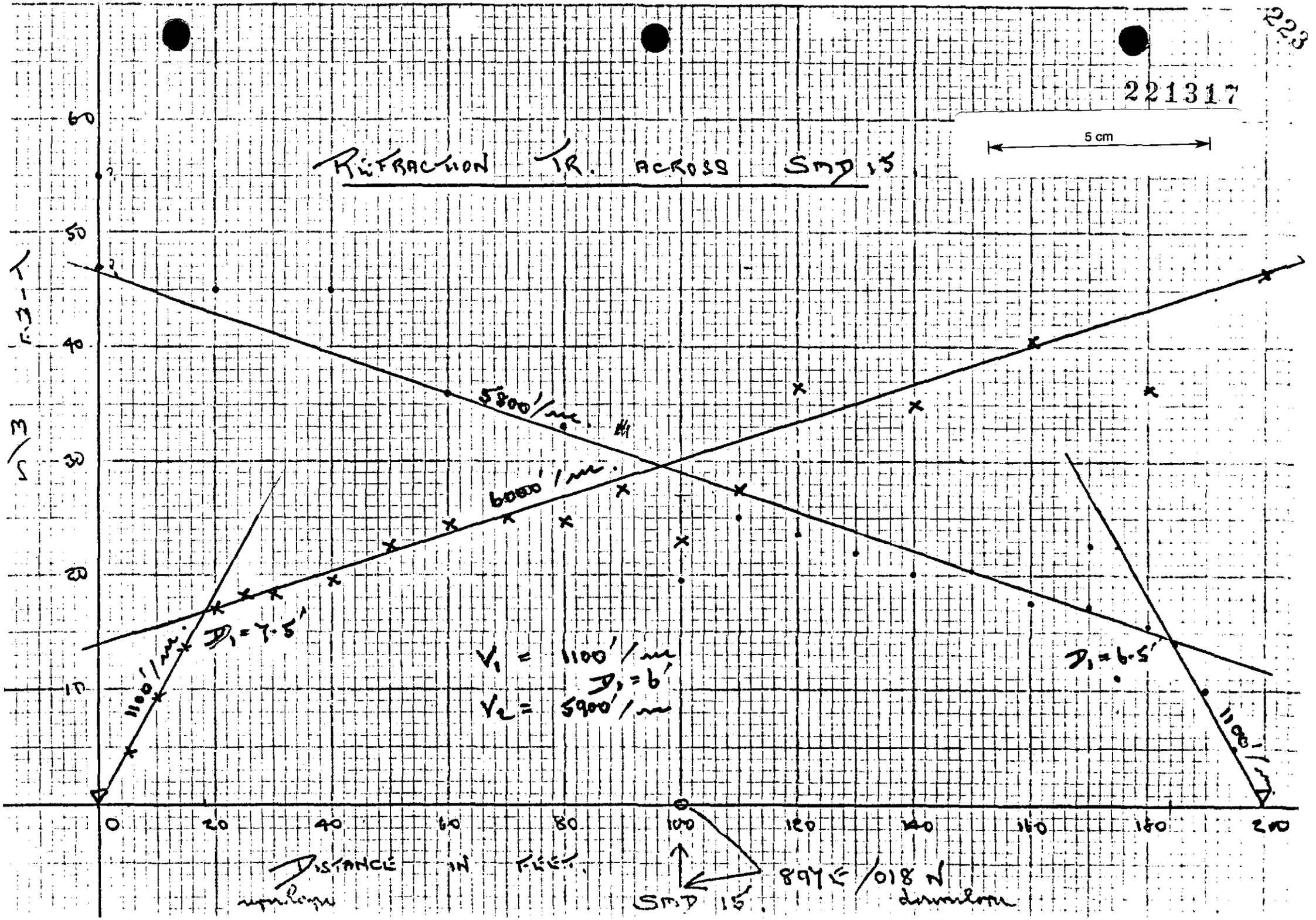
REFLECTIONS ON SMP 15.



221317

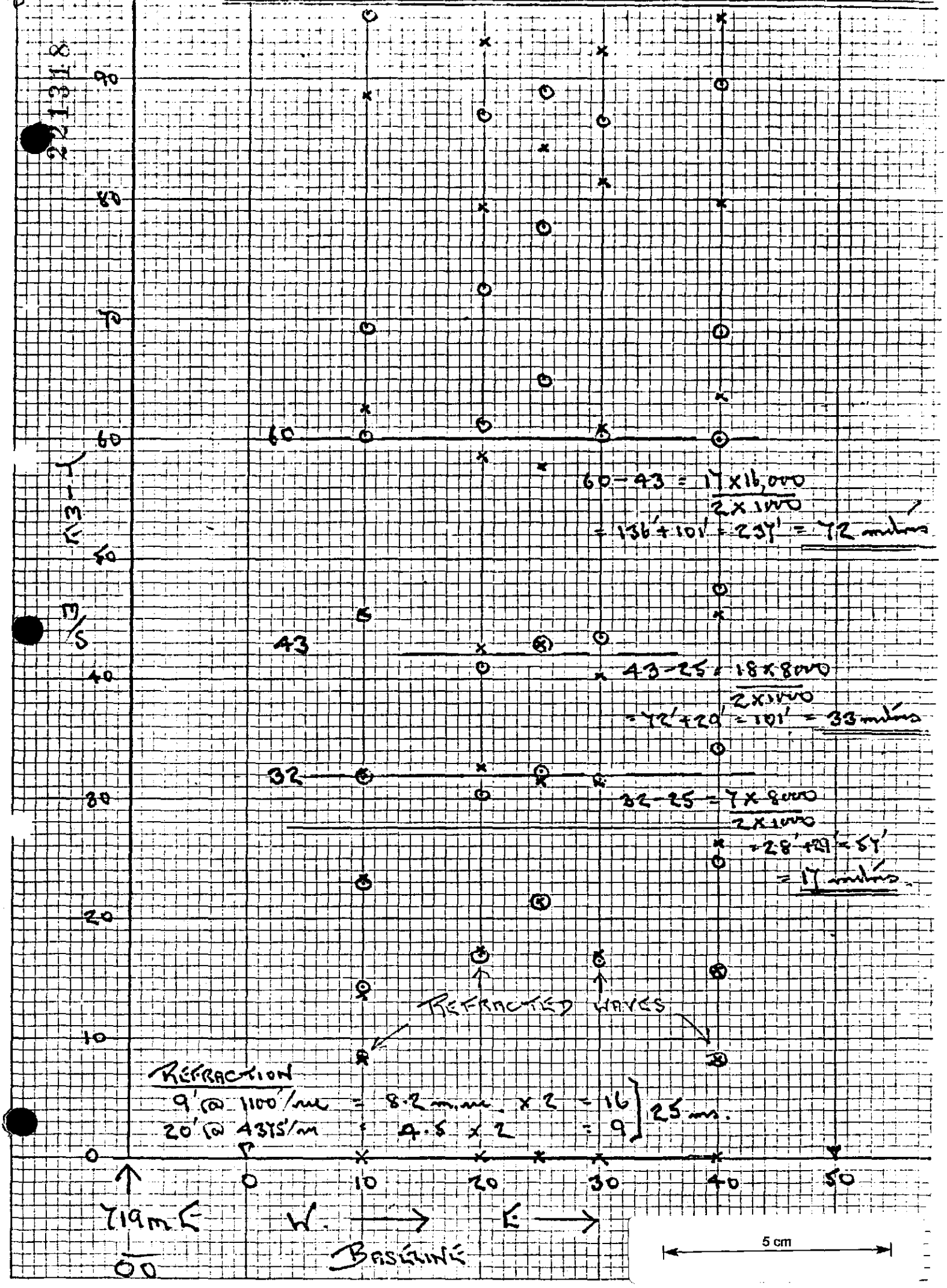
5 cm

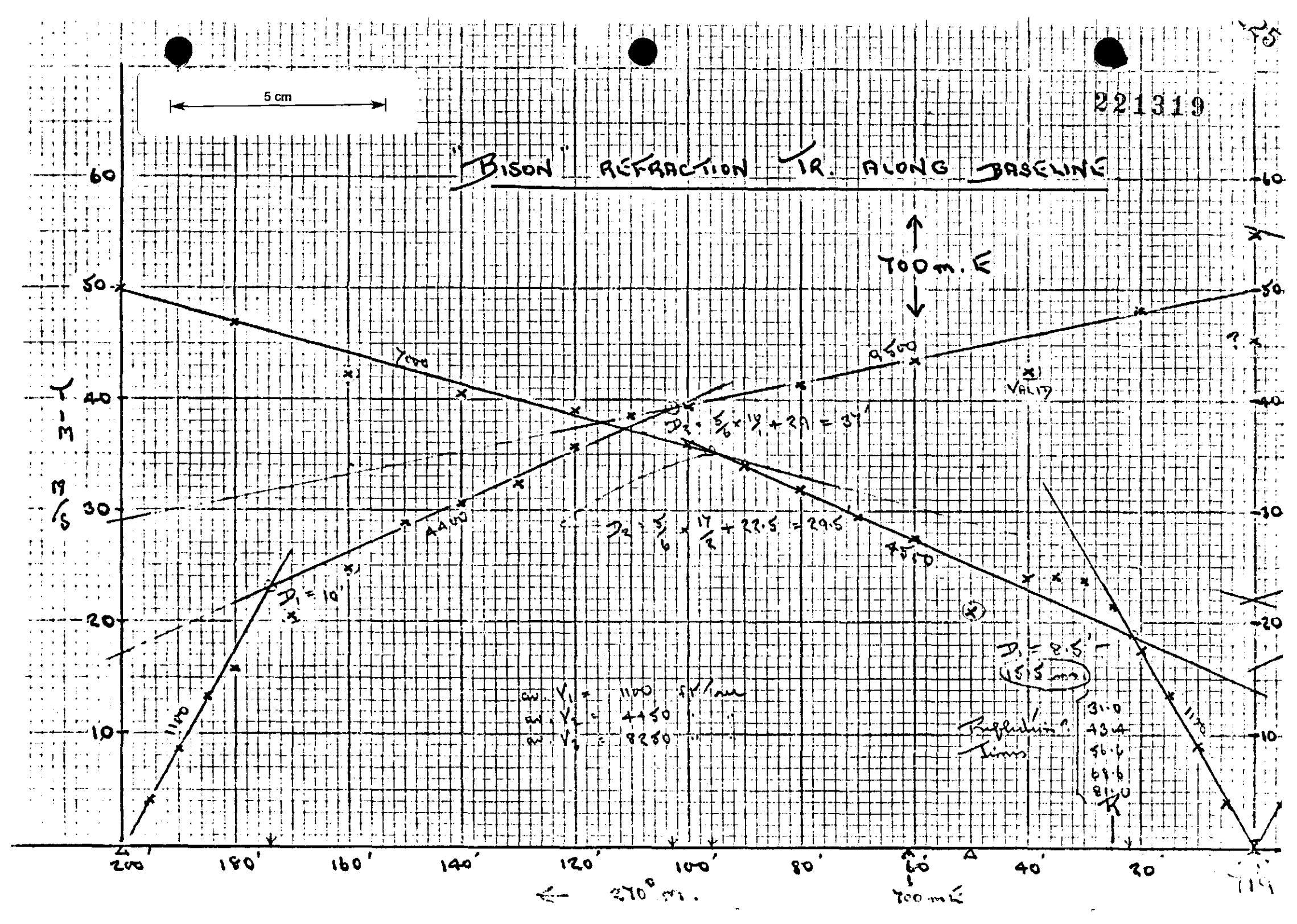
REFRACTION T.R. ACROSS SMD 15



224

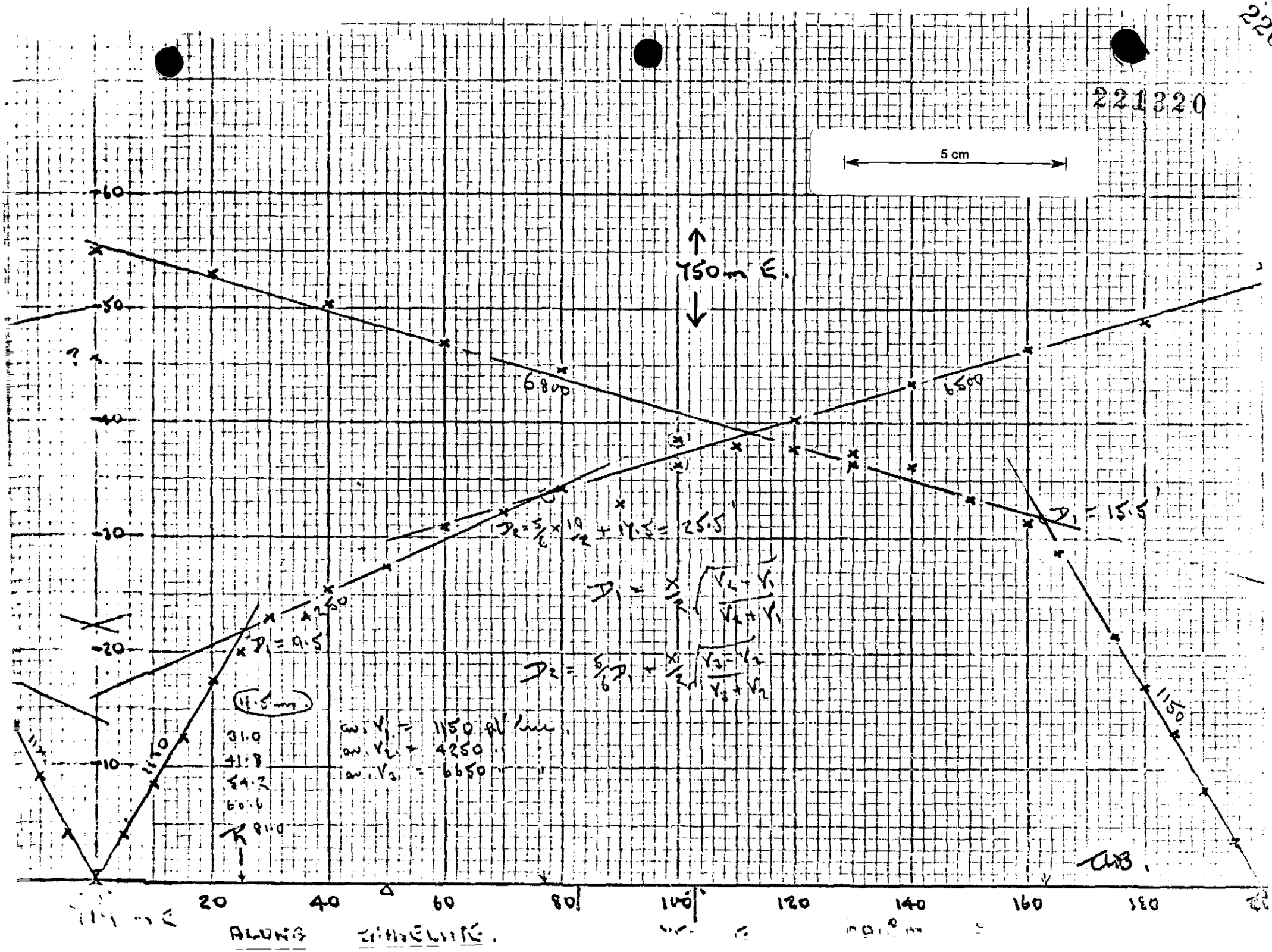
REFLECTIONS AT ~ 700 m E. ON BASELINE





5 cm
221319
"BISON" REFRACTION TR. ALONG BASELINE
700 m. E
Y
m/s
60
50
40
30
20
10
7000
9500
4400
4500
1100
VALID
D2 = 5/6 × 17/2 + 22.5 = 29.5
D1 = 10'
D1 = 8.5'
av. V1 = 1100 ft/sec
av. V2 = 4450
av. V3 = 8250
Reflection Times
31.0
43.4
56.6
68.6
81.0
200'
180'
160'
140'
120'
100'
80'
60'
40'
20'
← 270° m.
700 m E

221320


5 cm
750 m E.
6800
6500
1150
$D_1 = 15.5$
$D_2 = \frac{5}{6} \times 19 + 12.5 = 25.5$
$D_1 = \frac{x}{2}\left(1 + \frac{V_2 - V_1}{V_2 + V_1}\right)$
$D_2 = \frac{5}{6} D_1 + \frac{x}{2}\left(1 + \frac{V_3 - V_2}{V_3 + V_2}\right)$
$D_1 = 9.5$
11.5 ms
31.0
41.8
54.2
60.6
81.0
ω. V_1 = 1150 ft/sec.
ω. V_2 = 4250 "
ω. V_3 = 6650 "
60
50
40
30
20
10
20
40
60
80
100
120
140
160
180
ALONG LINELINE.

APPENDIX 24

DIAMOND DRILL LOGS, MOINA

NOTES:

A. Bedding angles quoted are measured from the core axis

B. Abbreviations used:

(a) Grain sizes

f	very fine grained	grains not visible to naked eye.
f	fine grained	less than 1 mm
m	medium grained	1 - 10 mm
crs	coarse grained	1 - 2 cm
crs	very coarse grained	2 cm +

(b) Degrees of Chemical Weathering:

CW	Completely weathered	* Crumbles and disintegrates when gently shaken in water.
HW	Highly weathered	* All of fabric weathered. * Iron oxide staining permeates the fabric grain boundaries. * Weakened to the extent that pieces the size of 5 cm diameter core can be broken and crumbled by hand. * Gives a dull sound when struck with a hammer. * Does not disintegrate when soake in water
MW	Moderately weathered	* Most of the fabric weathered. * Iron oxide staining permeates the fabric grain boundaries. * Pieces the size of 5 cm diameter core cannot be easily broken by hand. * Does not ring when struck with a hammer.
SW	Slightly weathered	* Some of the fabric weathered (Much felspar weathered to clay/ sericite). * Fine grained rocks may have a bleached appearance. * Iron oxide staining not usually throughout the fabric. * Strength approaches that of fresh rock. * Rings when struck with a hammer.
Frst.	Fresh with limonite stained joints	* Fabric not weathered. * No iron oxide staining in the fabric. * Joint faces are superficially coated or stained with limonite. (Possibly also stained with manganese oxides).
Fr	Fresh	* Fabric not weathered. * Joints not limonite stained (but may be coated with other minerals including chlorite, calcite, pyrite, quartz or clay.)

No sulphide minerals such as pyrite are present in any rock which is more weathered than slightly weathered. In a slightly weathered rock most sulphides are usually converted to oxides, carbonates, etc.

221323

PROJECT: MOINA

CORE SIZES: 0' - 16' NX casing
16' - 179' NX
179' - 871.2' BX

LOGGED BY: P. ASKINS

HOLE NO: ML 1 A

DRILLED: 1970 for Mt. Lyell Mining and Railway Co. Ltd.

ORIENTATION: 50° ⟶ 180°m

CO-ORDS: 1170E 70N

FROM	TO	DESCRIPTION
Feet and tenths of Feet		
0	17	CALC-SILICATE ROCK, SW-MW, f-m, bluish, tends to metasiltstone. Typical irregular texture with a little malachite staining on joints at 8.00. Bedding, f, 20° at 6.00. No scheelite.
17.0	108.0	NEAR NIL RECOVERY - FAULT ZONE - A little breccia recovered - angular fragments of quartz and calc-silicate minerals in f dark matrix. Also some quartz, fractured with chlorite and in one place crs molybdenite. No scheelite.
162.0	188.0	ALTERED SANDSTONE, Fresh - Frst, f, with much chlorite (as above) in matrix and on joints/fractures. Very fractured. No scheelite.
188.0	195.0	SANDSTONE - fresh - Frst, f, as above, but chlorite in matrix nearly absent. Bedding 10°. No scheelite.
195.0	211.0	SANDSTONE as at 162 - 188 but extremely crushed, fracutured, 200 - 202.5, 204 - 206. No scheelite.
211.0	739.5	SANDSTONE fresh - Frst, mostly massive, m, off white, also finely bedded and tends to f sandstone and siltstone, greys, browns, e.g. 510.7 - 524. Coarser sandston is often silicified and recrystallized to quartzite. Quartzite more abundant below 524. Minor chloritic alteration throughout, locally chlorite rich e.g. 345 - 353. (Chlorite with disseminated molybdenite 273 - 273.5, and ditto also with pyrite 290.5 - 290.6.) Weak greisenization along joints. Pyrite common in places, f-m, disseminated, also as stringers and on joints. Quartz veinlets and stringers scattered throughout - mostly barren of sulphides e.g. 276 - 277 but some with traces molybdenite; 430.5 - 431.1 quartz vein with disseminated yellowish oxides after pyrite or bismuthinite? Contact 90°; 503.5 - 504 quartz vein with m - crs molybdenite, contact at 45°. No scheelite. Bedding 80° 287'; 40° 307'; 50° 317'; 70° 357'; 70° 400'; 70° 418'; 70° 435'; 70° 460'; 90° 490'; 80° 515'; 65° 563'; 70° 583'; 65° 626'.

221224

739.5	744.0	ALTERED GRANITE, fresh, m - crs, white. Consists of white clay and quartz. Soft; i.e. extensively kaolinized. No scheelite
744.0	755.0	GREISENISED GRANITE, fresh, m, greenish grey; consists of m quartz and pale mica, massive. No scheelite.
755.0	757.0	ALTERED GRANITE, fresh, buff, m; Felspars altered to yellowish stained sericite and clay. Alteration is later than greisenization because weak sericite and clay alteration of greisen near contacts above and below. No scheelite.
757.0	759.0	GREISENISED GRANITE, as above. Contact with quartzite below either gradational or confused by jumbled core. Mere trace disseminated scheelite.
759.0	771.0	QUARTZITE, SANDSTONE, fresh, similar to above. No scheelite.
771.0	771.5	GREISENISED GRANITE, as above.
771.5	774.2	ALTERED GRANITE, fresh, as above.
774.2	784.2	GREISENISED GRANITE, as above but coarser. Some m - crs weakly fluorescing disseminated colourless fluorite.
784.2	871.2	GRANITE, fresh, altered as above, or with slight greenish colour; in places elsewhere relatively unaltered, flesh colour. Disseminated m molybdenite and mere trace f scheelite at 803.2 - 804.2; a trace molybdenite disseminated elsewhere. Traces f disseminated scheelite 813 - 858.

END OF HOLE

NOTE: Some scattered core sticks - jumbled - positions not known. Possibly around 750 - 780; some greisenized granite, f - m, with veinlets coarse wolframite. One stick with disseminated aggregates up to 2 mm of f hematite.

221325

PROJECT: MOINA CORE SIZES: 0 - 80 NX LOGGED BY: P.ASKINS

HOLE NO: ML 2 80 - 854 BX

DRILLED: 1971 for Mt. Lyell Mining & Railway Co. Ltd. ORIENTATION: 50° ⟶ 180°m CO-ORDS: 910E 150S

FROM Feet and tenths	TO of Feet	DESCRIPTION
00	68.0	BASALT, agglomeratic and, MUDSTONE dark grey indurated in part. Poor recovery so proportions of these rock types are unknown, but apparently mudstone units are between basalt flows.
68.0	77.0	MUDSTONE, medium grey, finely bedded in places.
77.0	79.4	SANDSTONE, Frst, f, buff, soft, massive, except at 77 where bedded at 85°.
79.4	90.0	SANDSTONE - SILTSTONE, Frst, f, dark grey, soft, mostly massive, with a few angular fragments of other soft sediments, basalt etc. Indurated towards 90.
90.0	108.0	BASALT, Frst, dark greenish, relatively massive; a few xenoliths or rafts of indurated mudstone.
108.0	126.0	BASALT, Frst, agglomeratic.
126.0	133.5	BASALT, Frst, mostly massive.
133.5	140.3	MUDSTONE, Frst, with some included semi-consolidated conglomerate with pebbles up to 1 cm
140.3	168.0	CALC-SILICATE ROCK, WRIGGLITE, CHLORITE SKARN. Calc-silicate is MW-HW and leached and wrigglite is SW-MW. Proportions difficult to establish due core losses but recovered proportions are 25% wrigglite, 75% calc-silicate, 5% chlorite skarn. Trace disseminated scheelite and traces scheelite in veinlets, only in wrigglite.
168.0	175.0	LIMESTONE + WRIGGLITE, Frst. Wrigglite as patches and veinlets in limestone. Good partial replacement features.
175.0	200.0	CALC-SILICATE ROCK + WRIGGLITE, similar to above but less weathered in most places.
200.0	272.5	WRIGGLITE, Frst - SW to 235.0, fresh to Frst below. Typical type with calc-silicate bands defining gross bedding at 80°; burrowed complicated calc-silicate/ wrigglite relationships; calc-silicate content increases 254.0 - 260.0; pink felspar veinlets throughout but more common 240.0 - 250.0. Up to 235 some calc-silicate is altered to buff chlorite which is locally leached out. Scheelite disseminated, and in some veinlets of felspar. (NOTE: In core trays 200 - 203 there is sandstone - this is obviously out of place)
272.5	275.0	CALC-SILICATE ROCK & WOLLASTONITE HORNFELS, Frst, f-m. Wollastonite hornfels is patchy. Grossly bedded at 80°, defined by pure white and greenish bands - pure white contains small quartz grains, m, and f wollastonite. Many cross-cutting veinlets with greenish alteration.

221326

275.0	279.0	CALC-SILICATE ROCK, METASILTSTONE, WRIGGLITE, Frst. Much m-crs brown garnet; gross bedding 90°. Scheelite as above.
279.0	284.0	CALC-SILICATE ROCK, MW-HW, leached.
284.0	297.0	CALC-SILICATE, WRIGGLITE, similar to 175 - 200. Wrigglite overall 30%; gross bedding 75° - 80°. Scheelite disseminated and in veinlets; in veinlets some crs.
297.0	318.0	CALC-SILICATE ROCK - Frst, typical type with usual veinlets of various types. Some metasiltstone (Calc-silicate is pinkish to buff in overall colour - metasiltstone is pale blue-greenish). A little scheelite, predominently in veinlets.
318.0	333.0	METASILTSTONE - Frst, tends to calc-silicate in places. Contact at 318 gradational. Bedded at 80° distinctly in places. Many cross cutting veinlets with green alteration and some greisen veinlets. A little scheelite in veinlets.
333.0	1058.0	SANDSTONE, Frst, variable in colour and texture. Tends to SILTSTONE in places. Overall off white. Mostly silicified pure white sandstone ("quartzite") 946 - 1010. Very minor pebbly sandstone in places e.g. 3 inches at 858. Frst joints may be partially due to storage of core in wet conditions; original core may have been fresh. Quartz veins 451.0 - 451.5; 453.0 - 453.5; 487.5 - 487.6; 501.0 - 502.0; 564.0 - 565.0; around 599 - 600; 655.5 - 655.6 (30° to core, with blue topaz and wolframite); 672.0 - 674.0; 843.5 - 849.5. Some veins are clear quartz, some are quartz plus silicified sandstone with a little mica. Quartz and altered sandstone with pyrite etc, banded in places at 45°, 843.5 - 849.5. They contain only traces of visible mineralization e.g. wolframite. Quartz veinlets often tend to greisen in places; some pyrite-chlorite alteration associated in places. Sandstone is more intensely greisenized 910 - 944. Bedding: mostly 60° up to 740; 30° at 750; 795; 60° at 825; 30° at 841; 80° at 862; 70° at 900; 80° at 924; 40° at 1010. A little scheelite in veinlets down to 424; rare beyond to 517; absent below except in veinlets at 766, 797.5, 808, 857 and in siliceous sandstone at 944 - 947 where scheelite is disseminated, f, with yellow fluorescence.
1087.0	1100.0	GREISEN, m-crs, bluish green to buff. Consists of pale mica and quartz. (NOTE: footages may be incorrect due jumbled and lost core)

END OF HOLE

221327

PROJECT: MOINA

HOLE NO: ML 3

DRILLED: 1971 for Mt. Lyell Mining & Railway Co. Ltd.

CORE SIZES: 0 - 36 NX casing
36 - 57 NX
57 - 319 BX

LOGGED BY: P. ASKINS

ORIENTATION: $50^{o} \longrightarrow 000^{o}m$

CO-ORDS: 1450E 125S

FROM feet	TO feet	DESCRIPTION
0	300.0	SANDSTONE, Recovered core SW to 57. Fresh - Frst 57 - 193, Fresh below 193.f-m, white altered to clay + sericite in matrix and on joints. Much scattered pyrite. Mere trace f scheelite, disseminated. No obvious visible bedding in most places; 50° 154. Becomes more intensely fractured especially after 265; Intense alteration with blue-green chlorite in matrix, 262-273.
300.0	319.0	WRIGGLITE, fresh, soft, fractured, highly altered. Trace disseminated scheelite and trace coarse scheelite in veinlets (one veinlet intersected).

END OF HOLE

(See also log of hole ML 3A)

221328

PROJECT: MOINA

HOLE NO: ML 3A

DRILLED: 1971 for Mt. Lyell Mining & Railway Co. Ltd.

CORE SIZES: 0 - 338 NX casing
338 - 818 BX

LOGGED BY: P.ASKINS

ORIENTATION: 50° ——→ 000°m

CO-ORDS: 1450E 125S

From	To	Description
Feet and tenths of Feet		
0	338.0	SANDSTONE, similar to ML 3.
338.0	340.0	SANDSTONE, fresh, f-m, highly siliceous with disseminated pyrite.
340.0	346.0	GOOP, soft broken chloritic, clayey intensely altered fault gouge. Possibly completely altered calc-silicate and wrigglite.
346.0	351.6	BRECCIA, SW, consists of fragments of siliceous sandstone and wrigglite up to 0.7 cm in iron oxide cement.
351.6	477.0	WRIGGLITE, Frst, typical type. Trace scheelite in veinlets to 360; absent to 408; a little disseminated and in veinlets 408 - 432; much f disseminated, with a little less in veinlets, 432 - 460, disseminated and in veinlets, (with lesser amount disseminated than to 460) 460 - 477. Pink felspar veinlets up to 3 cm wide more common after 407 and up to 12 inches wide e.g. 464 - 465. Traces wolframite in veinlets in places with molybdenite at edges of veinlets.
477.0	597.0	CALC-SILICATE ROCK, fresh - Frst, typical overall pink colour, variable textures. Some pink felspar veining especially near top. Traces disseminated and veinlet scheelite. Traces wolframite, molybdenite in veinlets of felspar in places.
597.0	700.0	METASILTSTONE, SILTSTONE, fresh, typical blueish colour. Grades to SILTSTONE after about 614. Very fractured and altered in places. e.g. 639 - 642. Traces disseminated and veinlet scheelite. Serpentine minerals on fractures in places, especially 687 - 700 (and up to 717). Quartz veinlet at 35° same sense as bedding, 647; Bedding 40° at 646; 10° at 645; 20° at 661; 10° at 669.
700.0	706.0	CALC-HORNFELS, fresh, consists of silt size green minerals, probably high Ca, Mg content. True width probably only 6 inches. Slightly more scheelite in and near this bed. Bedding 0° 703.
706.0	787.0	SILTSTONE, fresh, similar to above but overall tends to sandstone.
787.0	793.0	METASILTSTONE, fresh, - tends to green silty calc-hornfels as at 700 - 706. Irregular alteration along fractures.
793.0	797.0	CALC-HORNFELS, MW-HW, m, Quartz and limonitic clay, originally chlorite plus actinolite Fault zone .
797.0	809.0	METASILTSTONE, Frst, as above.

221329

809.0	818.0	CALC-HORNFELS, MW-HW, as above but some unweathered actinolite rich green hornfels. Bedded 0°. Contains disseminated scheelite and a little scheelite in veinlets.
818.0	854.0	SANDSTONE, Frst 818 - 836; fresh 836 - 854. m, white, mostly massive but some finely bedded; some with f green actinolite disseminated throughout. Scheelite nearly absent.

END OF HOLE

NOTE: Core storage - ML 3 core 0 to 319 mostly in aluminium trays
ML 3 A core 0 to 338 in original wooden trays. Some gaps in core 338 - 818 in aluminium trays.

221330

PROJECT: MOINA
HOLE NO: SMD 4
DRILLED: FEBRUARY 1976

CORE SIZES: 0 - 71.00 NQ
71.00 - 109.00 BQ

ORIENTATION: VERTICAL

LOGGED BY: P. ASKINS

CO-ORDS: 700E 240S

FROM	TO		DESCRIPTION
0	65.90	TERTIARY	BASALT, Frst to 50 m; fresh below 50 m. Agglomeratic and scoriateous throughout - in places crumbly, except for massive 8 - 20m. Indurated jungle green coloured mudstone/tuff with contained basalt and scoria fragments in places e.g. 20.70 - 21.00, 26.70 - 27.00 29.80 - 30.50, vicinity of 34 (core loss), 40.00 - 40.70 (soft crumbly), 45.95 - 46.05 52.55 - 52.75.
65.90	67.70		MUDSTONE, Frst, dark khaki brown, soft, crumbly along bedding at 80°.
67.70	67.85		SANDSTONE, Frst, pale brown, m, bedding 80°.
67.85	68.37		POLYMICTIC CONGLOMERATE, Frst, with variable highly angular rock types (weathered skarn, etc.)
68.37	70.70		WRIGGLITE, Frst, normal type - a few pale f. calc-silicate layers at 80° - original beds?
70.70	71.30		CALC-SILICATE ROCK + WRIGGLITE, predominantly f pale calc-silicate rock with irregular and in places ghostlike wrigglite suggesting replacement of pre-existing wrigglite.
71.30	80.63		WRIGGLITE, fresh, with much f po, py, cp in the layers in places. Minor siliceous originally slumped? beds 2 cm wide at 76.80 - 77.00, overall at 80°.
80.63	81.40		METASILTSTONE, fresh, pale grey, f, with many quartz-chlorite veinlets
81.40	81.66		METASILTSTONE, fresh, pale grey and white, bedded at 80°. Possibly originally marly.
81.66	83.70		CALC-SILICATE ROCK + WRIGGLITE, similar to 70.70 - 71.30.
83.70	88.00		WRIGGLITE, fresh; has 30 - 40% siliceous bands which are overall at 80° about 1 cm wide but irregular outline. Possible original silty beds.
88.00	90.60		CALC-SILICATE ROCK + WRIGGLITE similar to 70.70 - 71.30 but calc-silicate is more siliceous in places and shows relict metasiltstone suggesting that this unit was originally impure bedded limestone.
90.60	91.25		CALC-SILICATE ROCK + METASILTSTONE, highly fractured and altered to dark chlorite; core very broken.
91.25	92.45		CALC-SILICATE ROCK + a little wrigglite, as at 88.00 - 90.10 but slightly leached.
92.45	93.80		FAULT ZONE - fractures at 0 - 15° containing calcite and dark khaki coloured chlorite, in calc-silicate rock and wrigglite.

221331

93.80	94.40	CALC-SILICATE ROCK, fresh, similar to above.
94.40	95.90	METASILTSTONE, with minor (10%) wrigglite; fresh. In most places original texture of siltstone is preserved with bedding at 80°; in places calc-silicate minerals predominate; wrigglite is replacement of limey beds. 2 cm quartz vein 90° at 94.60.
95.90	97.15	METASILTSTONE, fresh, bedded at 70°; network of micro veinlets, many with greisen type alteration.
97.15	109.25	SILTSTONE - SANDSTONE, fresh, grey, micro veinlets as above especially 97.00 - 98.40 with associated py. Highly fractured 103 - 108 with chlorite and zeolites on fractures at various angles. Probable late stage fault zone.

END OF HOLE

REMARKS: Scheelite present throughout after 68.37: Two types; (a) f, white fluorescence, in quartz-felspar veinlets and (b) f, limegreen fluorescence, disseminated in wrigglite.

221332

PROJECT: MOINA
HOLE NO: SMD 5
DRILLED: MARCH 1976

CORE SIZES: 0 - 10 NQ casing
10 - 36 NQ
36 - 81 BQ

LOGGED BY: P. ASKINS
ORIENTATION: VERTICAL
CO-ORDS: 370E 130S

FROM	TO		DESCRIPTION
0	9.00	TERTIARY	MUDSTONE, dark greenish grey; SILTY SANDSTONE semi consolidated pale grey; MUDSTONE, black indurated.
9.00	15.18		BASALT, SW-MW, agglomeratic, scoriateous. Angular basalt fragments up to 10 cm.
15.18	16.40		MUDSTONE, khaki to dark grey brown. Up to 15.30 contains abundant devitrified glass fragments.
16.40	18.15		SILTSTONE, dark grey to black, poorly consolidated, bedding at 80°. Effloresces on exposure to air.
18.15	22.00		SILTSTONE - SANDSTONE, finely bedded. Siltstone dark grey and sandstone pale grey. Bedding at 0°-30°. Slumped and brecciated from 20.00 m with angular sandstone fragments - possibly Ordovician sandstone fragments.
22.00	23.20		LIMESTONE, Frst, pale grey, m, massive. Flecked appearance (due to calcite crystals?)
23.20	24.30		LIMESTONE, Frst, pale grey, m, with fine siliceous siltstone beds becoming frequent toward 24.30. Bedding at 40°.
24.30	24.79		SILTSTONE, Frst, off-white, calcareous and siliceous with thin lensoidal layers of limestone f-m pale grey. Bedding at 45°.
24.79	25.40		LIMESTONE, Frst, pale grey, f-m and SILTSTONE, as above, as irregular alternating beds at 50°.

25.40	30.80	LIMESTONE, Frst, pale greys, f-m, with slumped lensoidal beds of darker colour (argillaceous generally) and paler colour (arenaceous generally) overall at 50°. Slumped m-crs pale limestone lenses about 28.00 to 29.00.
30.80	38.10	LIMESTONE, Frst, pale and medium grey, f-m, well defined lensoidal and somewhat slumped paler beds which are apparently purer and less silty. Beds generally less than 2 cm wide at 45° - 60°. A little white f-f mineral along hair fractures fluoresces white with SW UV - ? scheelite. Core loss in more weathered crumbly zone at about 32.75.
38.10	39.80	LIMESTONE & CALC-SILICATE ROCK, fresh, previous rock type partially replaced by irregular, layers of calc-silicate minerals (green garnet, pyroxene and white wollastonite? Overall tendency is for layers to follow bedding at about 60° Some rare stringers of wrigglite.
39.80	40.35	WRIGGLITE 60%, CALC-SILICATE ROCK 40%. fresh. calc-silicate is f-m, finely layered and seems to be earlier than wrigglite. f po, py in wrigglite. No scheelite.
40.35	40.67	LIMESTONE, fresh, with veinlets of wrigglite. Irregular bleached appearance in places due to formation of wollastonite? No scheelite.
40.67	46.60	WRIGGLITE, fresh, with about 15% calc-silicate bands, at about 40°-70° up to 5 cm wide. Complex replacement relationships between wrigglite and calc-silicate. Minor secondary veinlets of py. Scheelite with white fluorescence in white felspar veinlets; with yellow fluorescence disseminated in wrigglite and minor amounts in veinlets. More calc-silicate bands (about 40%) after 45.00.
46.60	51.30	SKARN, Frst, cream to greenish, m, consists of pale calc-silicate minerals and fluorite. Also patchy calc-silicate rock and metasiltstone. Highly altered 46.60-48.80, chlorite-calcite alteration. Chlorite, dark greenish, through fabric; and calcite along veinlets. Scheelite occurs as above.
51.30	55.90	WRIGGLITE, CALC-SILICATE ROCK, METASILTSTONE, fresh, in various proportions:

		51.30 - 52.80 70% wrigglite 30% siltstone + calc-silicate, as layers probably originally bedding at 70°; 52.80 - 53.30 10% wrigglite 90% metasiltstone; 53.30 - 55.90 10% wrigglite 20% metasiltstone 70% calc-silicate with layering at 70°. Scheelite occurs as above.
55.90	56.35	METASILTSTONE, fresh, f, greyish, tending to creamy, calc-silicate in places. Network of microveinlets. One veinlet contains scheelite.
56.35	57.90	CALC-SILICATE ROCK, fresh, m, with minor metasiltstone layers, widest being 57.00 - 57.20 Scheelite absent except in greisen veinlet 57.45 - 58.00, parallel to core.
57.90	60.85	WRIGGLITE, METASILTSTONE, CALC-SILICATE ROCK, fresh, in various proportions: 57.90 - 58.30 80% wrigglite, 20% metasiltstone; 58.30 - 58.70 70% metasiltstone, 30% calc-silicate; 58.70 - 60.00 40% wrigglite, 60% metasiltstone; 60.00 - 60.85 25% wrigglite, 30% calc-silicate, 45% metasiltstone. Layering of different rock types 60° - 70°. Scheelite absent.
60.85	81.15	SILTSTONE, SANDSTONE, fresh, pale grey, f, bedding defined by silty-sandy layers a few mm thick at 70°. Shaley-silty bed 77.70 - 78.00. Minor calc-silicate and magnetite skarn layers up to 10 cm - contain a little py po and trace scheelite disseminated at 76.50. 10 cm magnetite po skarn at 72.90 - 73.00. Throughout, but especially to 64.50 and 72.90 - 74.40 network of veinlets and micro-veinlets with greisen assemblage; some of these veinlets contain trace scheelite, especially 63.30 - 64.50 in veinlet parallel to core. Core very fractured and broken 64.80 - 67.00 and 73.50 - 74.50 and 76.00 - 79.00 with pink and white zeolite along joints.

END OF HOLE

221335

PROJECT: MOINA
HOLE NO: SMD 6
DRILLED: APRIL 1976

CORE SIZES: 0 - 26.00 NQ
26.00 - 102.50 BQ

LOGGED BY: P. ASKINS
ORIENTATION: VERTICAL
CO-ORDS: 860E 50S

FROM	TO	DESCRIPTION
0	21.10	BASALT, SW - MW 0 - 19.70 ; Frst 19.70 - 21.10, scoriaceous, agglomeratic, as in SMD 5.
21.10	21.35	MUDSTONE, fresh, dark grey, with fragments of basalt.
21.35	21.60	WRIGGLITE & CALC-SILICATE ROCK, SW, bleached. No scheelite
21.60	22.75	FAULT ZONE, pale blue grey chlorite recovered
22.75	26.15	WRIGGLITE, SW - MW to 23.15, Frst elsewhere, some metasiltstone and calc-silicate layers, usually 1 - 3 cm, in places at 75°. Anhedral garnet prophyroblasts disseminated in dark biotite, fluorite matrix 24.00 - 24.10 and to lesser extent elsewhere. No scheelite in SW - MW material, elsewhere in veinlets & disseminated as previous holes.
26.15	28.00	CALC-SILICATE ROCK & METASILTSTONE, Frst - MW, f-m, highly fractured, with brownish chloritic alteration and calcite veinlets in several places. Minor scheelite in veinlets of felspar.
28.00	30.80	WRIGGLITE, Frst except SW 28.00 - 28.30 and 29.15 - 29.50. Minor calc-silicate and metasiltstone layers at about 70°. Core fractured in SW zones, Scheelite as above.
30.80	32.15	METASILTSTONE, Frst - SW, f-m, grades from calc-silicate at 30.80. Altered in places with chlorite similar to that as at 26.15 - 28.00. Layered in places, probably bedding - at 70°. 2 cm khaki efflorescent chlorite? at 32.15 - possible fault & zone where core lost.
32.15	67.70	WRIGGLITE, Frst 32.15 - 36.40, some SW where fractured, Fr after 36.40. Calc-silicate and metasiltstone 33.90 - 34.50. Fractured 34.90 - 36.40. Textural variants in places between 36.90 - 39.00 spotty texture, porphyroblastic, with garnet ? in magnetite or biotite. Minor calc-silicate throughout but especially around 38.50 and 40.00 to 40.50. Minor metasiltstone beds throughout - about 1 - 3 cm wide at 50° - 80° overall about 15% by volume. Abundant f disseminated scheelite in places; scheelite in veinlets throughout but especially prominent in veinlets sub parallel to core 41.00 - 43.00. Pinkish zeolite along joints particularly where more highly fractured e.g. 63.40 - 63.60.

221336

67.70	74.00	METASILTSTONE, fresh, f-crs (tends to f sandstone); tends to calc-silicate rock in places; every where extensive chloritic alteration. Fractured in places with pink and white zeolites on joints. Probably is f-m granular garnet-quartz rock in many places. Contains disseminated and veinlet scheelite. Probably bedding is defined by variations in grain sizes, at about 70°. A little wrigglite after 73.50 as minor layers.
74.00	83.05	WRIGGLITE, fresh, similar to above but more calc-silicate and metasiltstone layers - about 25% by volume. Widest calc-silicate layers at 78.65 - 78.80, 79.60 - 79.85, 82.00 - 72.45. Greisen veinlet 82.30 - 82.40. Scheelite in usual association.
83.05	85.00	CALC-SILICATE ROCK, fresh, m with patchy blebs and layers of wrigglite. Swarm of ribbon like contorted pink felspar veinlets 84.30 - 84.50. Layering, probably bedding at 80°.
85.00	85.80	METASILTSTONE and lesser CALC-SILICATE, fresh, layering at 60° - 80°. Scheelite in usual association.
85.80	87.25	CALC-SILICATE ROCK, with minor metasiltstone, fresh. Dark brownish olive green chlorite alteration 86.20 - 86.40. Layering 70° - 80°. Scheelite in usual association.
87.25	91.70	CALC-SILICATE ROCK fresh, m, flesh colour. About every 50 cm there are 1 cm wide greisen veinlets with dark selvedge, easily mistaken for metasiltstone. Greisen veinlets contain a little scheelite, wolframite, molybdenite.
91.70	102.50	SILTSTONE, SANDSTONE, fresh,similar rock types to that in other holes. Network f veinlets containing biotite(?) 91.70 - 94.00. Scheelite mostly in veinlets up to 97.20 and not present below except a little disseminated at 100.50 and 101.50. Bedding 70° - 90°. Some joints with pink zeolite and white calcite coatings. Irregular chlorite veinlets up to 3 cm wide containing crs cassiterite, 102.00 - 102.50. Scattered quartz veins up to 3 cm thick mostly at 70° - 85° containing a little wolframite, cassiterite, molybdenite, pyrite.

END OF HOLE

221337

PROJECT: MOINA
HOLE NO: SMD 7
DRILLED: APRIL 1976

CORE SIZES: 0 - 14.00 NQ
14.00 - 71.50 BQ

LOGGED BY: P. ASKINS
ORIENTATION: VERTICAL
CO-ORDS: 975E 90N

FROM	TO	DESCRIPTION
0	45.35	WRIGGLITE, HW - CW 0 - 1.00 MW - HW 1.00 - 8.00 (as recovered) MW 8.00 - 12.00 SW 12.00 - 13.50 Frst - 26.00 Fr beyond Minor calc-silicate and metasiltstone layers especially 17.00-20.20; 35.00-45.35. Thickest calc-silicate layer is 21.75 - 22.65. Quartz veinlets 28.20 - 28.25 and 38.10 - 38.20 at 70° containing coarse wolframite. 1 cm wide quartz veinlet at 10° at 24.60 containing wolframite. Minor pink felspar veining. 2 cm wide quartz greisen veinlet at 43.00. Scheelite not visible up to 11.50 probably due to masking by iron oxides.
45.35	46.15	CALC-SILICATE ROCK, METASILTSTONE, WRIGGLITE, fresh, highly irregular - about 15% wrigglite. Scheelite near absent.
46.15	48.55	WRIGGLITE, CALC-SILICATE ROCK, METASILTSTONE, fresh, about 50% : 25% : 25%. Scheelite in veinlets.
48.55	52.75	CALC-SILICATE ROCK, minor METASILTSTONE, fresh, m, flesh colour to greenish. Patchy chloritic alteration at 48.30. Minor pink felspar veinlets. Scheelite in veinlets.
52.75	55.00	CALC-SILICATE ROCK, WRIGGLITE, fresh, irregular and patchy distribution of these types Overall 60% calc-silicate, 40% wrigglite. Scheelite in veinlets and minor disseminated.

221338

55.00	65.00	CALC-SILICATE ROCK, fresh, flesh colour, m, with irregular patches, layers of metasiltstone. Irregular blebs of m biotite. More fractured and altered with dark olive green chlorite 60.95 - 65.00. A little scheelite in veinlets, some greisen like.
65.00	71.50	METASILTSTONE, fresh, highly fractured and altered to dark chlorite along fractures. Tends to Calc-silicate in places e.g. 65.75 - 66.75. Minor magnetite in places. Scheelite in veinlets and disseminated in places in coarse metasiltstone.

END OF HOLE

221339

PROJECT: MOINA
HOLE NO: SMD 8
DRILLED: MAY 1976

CORE SIZES:
0 - 00.50 4 inch
0.50 - 3.00 NQ casing
3.00 - 42.50 NQ
42.50 - 60.80 BQ

LOGGED BY: P.ASKINS
ORIENTATION: VERTICAL
CO-ORDS: 00 550S

FROM	TO		DESCRIPTION
0	17.40	TERTIARY	BASALT, Mostly SW - Frst but more weathered in zones of core loss. Agglomeratic in most places.
17.40	22.25		BASALT, fresh, f-m, massive. Some included wisps of mudstone.
22.25	24.00		MUDSTONE, fresh, f, dark khaki brown; indurated.
24.00	25.05		BASALT, fresh, f-m, massive.
25.05	26.00		SAND, white, m, becoming coarse and gravelly toward 26.00. Panned: no cassiterite, trace f pyrite. Sheelite present as f grains with yellowy fluorescence. (Deep Lead)
26.00	31.00		MUDSTONE/SILTSTONE, MW to 27.50, brown; fresh 27.50 - 31.00, grey. f-m finely bedded, possibly glacial varve. Trace disseminated pyrite.
31.00	32.50		RUBBLE OF SANDSTONE, Crs, ironstained. Possibly Ordovician sandstone rubble of glacial origin, or sink hole in limestone
32.50	32.65		MUDSTONE/SILTSTONE, as above.
32.65	40.00		LIMESTONE, fresh, f-m, pale grey with about 25% irregular wispy slumped beds of calcareous siltstone, dark grey. Overall bedding around 80°. Trace f disseminated py, po throughout.
40.00	41.60		WOLLASTONITE HORNFELS, fresh, f, pale grey. Texturally identical to limestone above but slightly creamish and no effervescence with HCl - probably all $CaCO_3$ has been converted to wollastonite. Disseminated f py. Bedding 70° - 90°.
41.60	42.50		CALC-SILICATE ROCK, leached, weathered, very poor recovery.
42.50	43.20		WOLLASTONITE HORNFELS, fresh, becoming SW to 43.20.
43.20	45.95		CALC-SILICATE ROCK , SW, leached in most places, m - crs, Irregular layering, probably bedding, 70° - 85°. Scheelite absent. Core broken. Contact at 43.20 gradational
45.95	46.35		WRIGGLITE, fresh, f-m. Scheelite absent.
46.35	49.60		METASILTSTONE, fresh, f-m pale greenish, with network of fractures with black biotite(?) alteration. Wrigglite layer 47.80 - 47.90. Scheelite absent.
49.60	51.70		METASILTSTONE, fresh, f, pinkish, more compact and silicified than previous type. Finely bedded. Resembles rock at 40.00 - 41.60 except probably more metasomatized and may have been more calcareous originally than metasiltstone at 46.35 - 49.60. Trace scheelite in veinlets near 49.50

221340

51.70	60.80	METASILTSTONE, fresh, f-m, variable texture and colour - probably reflecting calcareous and non calcareous units. Bedding 70°. Scheelite absent. Complicated alteration along fractures. A little po, py disseminated throughout . Calcite, zeolite in fracture zones 53.80 - 54.50.

END OF HOLE

PROJECT: MOINA

HOLE NO: SMD 9

DRILLED: MAY 1976

CORE SIZES: 0 - 17.00 NQ
17.00 - 129.70 BQ

LOGGED BY: P. ASKINS

ORIENTATION: VERTICAL

CO-ORDS: 150E 100S

FROM	TO	DESCRIPTION
0	12.00	CLAY, pale brown
12.00	91.50	LIMESTONE, - see separate log by P.W. Stainton for details.
91.50	104.60	SKARN, fresh, f-m, sharp contact with limestone, dark grey green to black-consists of magnetite, actinolite, biotite (?), and in places abundant po. Po rich 94.20 - 99.50 and 101.40 - 104.60. Crudely layered in many places at 30° - 70°; layers 1 - 5mm. Scheelite nearly absent. 3 cm wide fault brecciated angular fragments of skarn in calcite cement, 99.55 - 99.60 Zone of fault breccia, at 30°, 103.20 - 103.55.
104.60	105.20	SKARN, fresh, f-m, patchy dark green and pale green and in places layered at 70°. Consists of actinolite(?), biotite(?), and pale calc-silicate minerals. Scheelite absent.
105.20	106.50	ACTINOLITE(?)ROCK, fresh, f,relatively structureless.
106.50	112.00	SKARN, fresh, f-m, similar to 104.60 - 105.20 but texture slightly wrigglite-like in most places. Some layers of metasiltstone (?), open folded with amplitude about 0.5 m - original slumps in bedding? or tectonic folds(?). Richer in actinolite 107.65 - 108.40. Becomes paler and noticeably more pale garnet rich 111.00 - 112.00. Scheelite nearly absent.
112.00	113.15	SKARN, fresh, similar to above but distinct metasiltstone layers f, pale bluish, about 1-2cm wide, at 50°. Trace disseminated scheelite.
113.15	113.75	SKARN, fresh, f, overall dark grey to black, rich in actinolite and/or biotite, tends to wrigglite in places. Trace scheelite in veinlets.
113.75	118.65	SKARN, fresh, f-m, rather variable in texture but overall similar to 105.20 - 112.00. Minor calcite filled fractures up to 3 mm wide sub-parallel to core. Magnetite has patchy distribution - overall approximately 10%. No metasiltstone clearly recognizable - bedding possibly outlined by calc-silicate layering but is irregular. Trace disseminated scheelite, and in veinlets.

221342

118.65	125.00	FAULT ZONE, fresh, consists of crs calcite, purple crs fluorite and pale grey-greenish chlorite, f, with angular fragments of altered skarn and metasiltstone. Fault zone oriented sub-parallel to core.
125.00	128.25	METASILTSTONE, fresh, f, greenish-grey with network of fine veinlets with associated black biotite alteration. Trace disseminated scheelite, and in veinlets.
128.25	129.70	FAULT ZONE, fresh, broken, fractured, altered metasiltstone with patchy m white calcite filling fractures. End of fault zone possibly appearing at 129.50. Oriented sub-parallel to core.

END OF HOLE

221343

PROJECT: MOINA
HOLE NO: SMD 10
DRILLED: JUNE 1976

CORE SIZES: 0 - 45.00 NQ
45.00-117.50 BQ

LOGGED BY: P. ASKINS
ORIENTATION: VERTICAL
CO-ORDS: 700E 50N

FROM	TO		DESCRIPTION
0	29.00	TERTIARY	BASALT, 0 - 6 poor recovery probably HW - CW 6 - 22 matrix scoria mostly MW with basalt fragments fresh - SW 22 - 29 mostly SW matrix Scoriaceous as in previous holes. 0 - 27 much white f clay (or zeolite(?)) in scoria interstices.
29.00	31.00		SILTSTONE-MUDSTONE, fresh, grey, soft, friable, bedding folded at 30.00
31.00	34.00		BASALT, Frst becoming HW towards 34.00, scoriaceous
34.00	36.70		GRAVEL-SAND, angular and sub-rounded pebbles up to 5 cm of mostly quartz-rich sandstone, quartzite (Moina sandstone and Roland conglomerate fragments?) - areas of no core recovery probably are sand. Trace f disseminated scheelite with golden fluorescence.
36.70	38.75		SILTSTONE-MUDSTONE, fresh, graphite rich, some disseminated pyrite 1 cm patch of m sphalerite at 37.00, possibly preserved because of reducing graphitic conditions. Bedding 70°.
38.75	46.30		WRIGGLITE, some calc-silicate rock, some metasiltstone. 38.75 - 41.00 HW - CW, 41.00 - 46.30 Frst. Calc-silicate and metasiltstone are irregular probably because of original burrowed beds. One calc-silicate layer 44.00 - 44.30. Metasiltstone bedding 65° at 43.40. Patches pyrite locally. Scheelite nearly absent, only a little in scattered places.
46.30	47.55		SANDSTONE, METASILTSTONE. Frst. Pale cream, off-white, m, quartz sandstone in most places, with silty beds e.g. at 47.30 at 60°.
47.55	56.50		WRIGGLITE, METASILTSTONE interlayered. 47.30 - 51.00 Frst, 51.00 and below fresh. Metasiltstone is off white, quartz rich, tends to f sandstone, and to calc-silicate in places.

221044

47.55	56.50	.....(continued) Main units are 47.55 - 50.00 wrigglite 50.00 - 50.75 f sandstone 50.75 - 51.65 wrigglite 51.65 - 52.40 metasiltstone 52.40 - 52.75 wrigglite 52.75 - 53.30 metasiltstone/calc-silicate 53.30 - 53.90 wrigglite 53.90 - 54.10 metasiltstone/calc-silicate 53.10 - 54.30 wrigglite 54.30 - 54.70 wrigglite + metasiltstone, irregular 54.70 - 55.30 wrigglite 55.30 - 55.80 metasiltstone, bedding 70° 55.80 - 56.50 wrigglite with disturbed metasiltstone beds at 65° Scheelite nearly absent but traces f, disseminated, towards 56.50
56.50	63.65	WRIGGLITE, fresh, with lesser metasiltstone in finer distrubed beds.Scheelite disseminated, and in veinlets.
63.65	73.25	WRIGGLITE, fresh, with more metasiltstone beds, mostly disturbed,at 50°- 70°, about 1 - 2 cm thick. Except 63.65 - 64.25 where metasiltstone partly leached by meteoric water. Pink felspar veinlets with no scheelite at 69.70. Scheelite in veinlets and traces disseminated.
73.35	73.60	QUARTZ vein, fresh, crs, containing crs wolframite, bismuthinite,red-brown cassiterite. Upper contact at 30°.
73.60	76.00	WRIGGLITE + METASILTSTONE, fresh, similar to 63.65 - 73.35 but more metasiltstone beds at 55°- 60°. Scheelite as above.
76.00	79.00	METASILTSTONE, CALC-SILICATE ROCK,fresh, irregularly layered - disturbed burrowed bedding? at 70°. Some disseminated scheelite.
79.00	79.50	WRIGGLITE, fresh, core broken, fractured
79.50	80.90	METASILTSTONE, fresh fractured, chloritic alteration.
80.90	82.20	QUARTZ vein, fresh, contacts at 20°. (contains patches of biotite in places, rare cassiterite, rare sulphides; brecciated in parts with included fragments of country rock.)

221045

82.20	83.35	WRIGGLITE + METASILTSTONE, fresh, irregularly layered. A little scheelite in veinlets and disseminated.
83.35	85.85	METASILTSTONE, fresh, similar to above.
85.85	86.60	QUARTZ vein, sub parallel to core, with some included wrigglite (country rock); contains fluorite, cassiterite and a little py, po, cp, bismuthinite or molybdenite, f.
86.60	87.40	WRIGGLITE + METASILTSTONE, fresh, irregularly layered. A little scheelite disseminated and in veinlets.
87.40	90.50	METASILTSTONE, with some wrigglite layers, fresh. Bedding 70° where distinct.
90.50	117.50	METASILTSTONE, SILTSTONE, SANDSTONE, etc., fresh. Quartz vein similar to above but relatively unmineralized, 93.50 - 94.50.

END OF HOLE

221346

PROJECT: MOINA

HOLE NO: SMD 11

DRILLED: JULY 1976

CORE SIZES: 0 - 30.50 NQ
30.50 - 120.00 BQ

LOGGED BY: P. ASKINS

ORIENTATION: VERTICAL

CO-ORDS: 750E 100S

FROM	TO		DESCRIPTION
0	14.00	TERTIARY	BASALT, poor recovery, probably mostly HW - CW.
14.00	15.00		MUDSTONE, fresh?, semi consolidated, grey.
15.00	17.25		BASALT, poor recovery, probably mostly HW - CW.
17.25	20.50		MUDSTONE - SILTSTONE, grey.
20.50	21.50		BASALT, SW, vesicular.
21.50	26.25		MUDSTONE - SILTSTONE, fresh?, dark grey, partly indurated to 24.00, below 24.00 is more sandy with some small (up to 1 cm) pebbles of various rock types.
26.25	29.50		SAND-GRAVEL, white, semi-consolidated.
29.50	29.80		MUDSTONE, fresh, dark grey, indurated.
29.80	31.00		BASALT, F Frst, agglomeratic.
31.00	31.65		WRIGGLITE and CALC-SILICATE ROCK, fresh. Fracture parallel to core axis around 31.50, with py coating. A little disseminated scheelite.
31.65	32.20		NO CORE - probable fault, at 0°.
32.20	32.90		METASILTSTONE - SANDSTONE, fresh, pale green and off white, with greenish microveinlets. No scheelite. (Possibly same bed as in SMD 10, 46.30 - 47.55.)
32.90	35.00		WRIGGLITE, fresh, with calc-silicate irregularly "mixed" around 33.50. Contact towards 35.00 is irregular. Traces scheelite disseminated and in veinlets.
35.00	36.00		METASILTSTONE, fresh, pale cream to green, bedding 90°.
36.00	37.00		WRIGGLITE, fresh.
37.00	38.00		CALC-SILICATE ROCK + METASILTSTONE, fresh, similar to that occurring above.
38.00	60.70		WRIGGLITE, fresh, with calc-silicate locally to 42.50, and metasiltstone locally but increasing to 60.70. Layers of calc-silicate and metasiltstone very disturbed at about 80° overall. A little scheelite in veinlets, very little disseminated.
60.70	63.10		METASILTSTONE, fresh, tends to calc-silicate in places. Minor wrigglite in places. Scheelite, disseminated and in veinlets.

221347

63.10	68.50	WRIGGLITE + METASILTSTONE + CALC-SILICATE ROCK, fresh. Rock types complexly disturbed; around 40% wrigglite. Bedding disturbed, beds generally around 2 cm wide, at 80°. Greisen filled 1 cm thick vein 0° - 30° 67.00 - 68.50.
68.50	69.20	CALC-SILICATE ROCK, leached, green and cream. Scheelite occurs mostly disseminated, rare in veinlets.
69.20	69.65	WRIGGLITE + METASILTSTONE, fresh, similar to above.
69,65	70.85	CALC-SILICATE ROCK, fresh, tending to pale f sandstone and metasilstone. Bedding 90° Scheelite in only one veinlet parallel to core.
70.85	71.85	CALC-SILICATE ROCK AND METASILTSTONE, fresh, as above but with more dark alteration along micro-veinlets.
71.85	73.00	METASILTSTONE, fresh, with some small patches of wrigglite. Scheelite in a veinlet parallel and sub parallel to core.
73.00	120.00	METASILTSTONE, fresh, variable texture and fabric as in other holes; variously altered, tends to off-white f quartz sandstone in places. Around 107 - 111 much green ac actinolite alteration. (probably reflects more Mg rich original sediment) Beyond 113 rock is more fractured with yellow-brown thin coating on joints. Scattered veinlets scheelite.

END OF HOLE

221348

PROJECT: MOINA
HOLE NO: SMD 12
DRILLED: AUGUST 1976

CORE SIZES: 0 - 27.25 NQ
27.25 - 123.25 BQ

LOGGED BY: P. ASKINS
ORIENTATION: VERTICAL
CO-ORDS: 450E 50N

FROM	TO	DESCRIPTION
0	12.40	BASALT, agglomeratic 0 - 6.00 MW-CW, poor recovery 6.00 - 10.70 SW-MW, 10.70 - 12.40 Frst.
12.40	14.00	MUDSTONE, Frst, black indurated.
14.00	23.10	BASALT, Frst. 14.00 - 17.60; locally more weathered and fractured. Fresh 17.60 - 23.10 massive, f - f, shows thin (usually less than 3mm) dark alteration along joints and microfractures which are mostly 80° to core - has appearance of bedding at first glance.
23.10	24.60	MUDSTONE, fresh, dark grey, indurated at 23.10. grading down to SILTSTONE, SW, pale grey to khaki, semi-consolidated.
24.60	27.00	SAND, m, white, contains f - f scheelite, yellow fluorescence.
27.00	27.50 (approx)	GRAVEL off-white quartz - sandstone and quartz.
27.50	34.00	SAND, m, white, contains scheelite as above.
34.00	40.25	MUDSTONE, SILTSTONE, fresh except SW 40.00 - 40.25, grey to brown grey, semi-consolidated, finely bedded at 90°.
40.25	41.50	LIMESTONE, fresh, pale grey, burrowed? See log of P.W. Stainton for details of limestones.
41.50	42.00	MUDSTONE, fresh, with contained angular rock fragments - probably cave-fill.
42.00	43.40	LIMESTONE, CALC-DOLOMITE, SW-MW, f-m, grey and off-white respectively, irregularly bedded.
43.40	44.20	MUDSTONE, containing angular rock fragments - probably cave-fill
44.20	45.00	LIMESTONE, CALC-DOLOMITE, as above.
45.00	45.75	probable no core

221349

45.75	47.00 (approx)	SILTSTONE, fresh, white, massive, compact, very slightly calcareous.
47.00	50.75	MUDSTONE, with some angular fragments - poor recovery - probably semi-consolidated cave-fill.
50.75	54.70	LIMESTONE, fresh - Frst, grey, f, some disturbed bedding, overall at 80°.
54.70	55.75	SILTSTONE, fresh, white, similar to above but some CALC-DOLOMITE beds.
55.75	61.00	LIMESTONE & CALC-DOLOMITE, fresh, grey, interbedded, burrowed bedding, overall at 80°.
61.00	61.75	SILTSTONE, fresh, white, grades to calcarenite (?) or Calc-dolomite(?).
61.75	63.00	LIMESTONE, CALC-DOLOMITE, fresh, mostly dark grey, interbedded.
63.00	68.00	LIMESTONE, fresh, mostly white to pale grey, (looks like siltstone above but effervesces vigorously with acid), finely bedded at 70°, gradational contact at 68.00. Possibly slightly recrystallized, bleached by very weak metasomatism.
68.00	77.00	LIMESTONE, fresh, pale grey, off-white and dark grey, burrowed bedding.
77.00	81.00	LIMESTONE, fresh, pale grey and off-white. Differs from above - more off-white disturbed (burrowed) layers, at 70° - 80°.
81.00	84.60	LIMESTONE, fresh, pale grey, with irregular beds 0.5 - 10.0 cm wide of white ? dolomite usually with dark grey edges. (This seems to be good marker horizon)
84.60	91.50	LIMESTONE, fresh, pale grey and mid grey, finely bedded, less disturbed at 70° - 80°.
91.50	94.00	LIMESTONE, CALC-SILICATE ROCK WRIGGLITE, fresh, (partial replacement of limestone - shows good sequential development of wrigglite - passage for solutions partially controlled by fractures (probable main avenues) and in detail controlled by permeable dolomites. Traces of Scheelite disseminated and in veinlets in calc-silicate and wrigglite.
94.00	95.35	SILTSTONE, fresh, white, f-m, massive, has irregular patches of wollastonite, f, replacing limestone after 94.80.
95.35	97.00	CALC-SILICATE ROCK, fresh, contains very minor wrigglite especially at 96.00
97.00	104.80	WRIGGLITE & CALC-SILICATE ROCK, fresh; wrigglite overall about 40%, in patchy distribution. Original rock texture: variable disturbed burrowed bedding overall 70°-80° in places. Original calc-dolomite (now calc-silicate) was fairly thick (see photos). Traces of scheelite disseminated in places, and in veinlets.
104.80	109.50	CALC-SILICATE ROCK, METASILTSTONE, with minor WRIGGLITE, fresh; proportions: 60%, 30%, 10% Scheelite as above.
109.50	123.25	METASILTSTONE, fresh, f-m, typically grey to off-white some brownish beds up to 3 cm at

221350

65° - 80°. Has characteristic network of fractures of dark greenish alteration to 113. Very minor wrigglite in small isolated patches to 110.50. Scheelite nearly absent.

END OF HOLE

221351

PROJECT: MOINA
HOLE NO: SMD 13
DRILLED: DECEMBER 1976

CORE SIZES: 0 - 1.00 NQ casing
1.00 - 40.00 NQ
40.00 - 182.25 BQ

LOGGED BY: P. ASKINS
ORIENTATION: VERTICAL
CO-ORDS: 1450E 50S

FROM	TO	DESCRIPTION
0	32.50	SANDSTONE, Frst - SW, f-m, off-white to cream, highly fractured with pyrite in micro-fractures and disseminated, generally 1-5% but locally greater than 10%. Less fractured core is more siliceous, and broken core is more fractured, silty and generally contains more pyrite. Less fractured sandstone is "tubicular". Bedding 70° at 31. No scheelite.
32.50	34.20	SANDSTONE, SW, m, with much cream clayey alteration throughout. No scheelite.
34.20	35.00	SANDSTONE, HW, m; orange brown limonitic clay throughout fabric. Bedding 65° at 34.90 No scheelite.
35.00	36.15	METASILTSTONE, Frst - HW, f-m, greenish, finely bedded at 45°- 60°. No scheelite.
36.15	36.95	MAGNETITE-CHLORITE SKARN, Frst - MW, coarse magnetite, fine chlorite and pale calc-silicate minerals with streaky and spotty texture - bedded at 65°. Merest trace of f scheelite.
36.95	37.50	METASILTSTONE, similar to above, no scheelite.
37.50	39.00	SILTSTONE - SANDSTONE, Frst - MW. f-m. Finely bedded at 55°. No scheelite.
39.00	53.00	SANDSTONE, Frst, silicified ("quartzite"); tends to siltstone in places. A little disseminated pyrite and some pyrite on joints. Bedding 45° at 44; 40° at 46 and 30° at 48. See core photographs for location of fractured areas. No scheelite.
53.00	54.75	SILTSTONE-SANDSTONE, Frst, f-m, cream; much cream clay through fabric. Bedding 20°.
54.75	67.00	SILTSTONE-SANDSTONE, Frst, f-m, off white, mottled by discontinuous microfractures containing f pyrite; pyrite also disseminated, f, and along joints, m. Minor quartz veinlets. Bedding 45° at 62.....50° at 67. No scheelite.
67.00	79.00	SANDSTONE, Frst, Silicified as above interbedded with sandstone-siltstone similar to above. Bedding 90° at 72, 90° at 73.25. Very broken and some lost core after 74.50. (Lost water at 74.30 when drilling).No scheelite.

79.00	83.70	SILTSTONE-SANDSTONE, Frst, finely bedded but highly broken. Bedding 70°. Minor disseminated pyrite. No scheelite.
83.70	86.30	SKARN, SW-MW, coarse, leached in part, very broken. Consists of pale and dark green minerals. Dark green mineral is chlorite after amphibole (?). No scheelite
86.30	87.35	SKARN, SW, coarse, massive, similar to above but not broken; leached - leached cavities probably mostly after sphalerite. No scheelite.
87.35	97.00	SPHALERITE-SKARN, Frst, coarse,massive, with coarse sphalerite aggregates up to 1 cm disseminated throughout. Greatest sphalerite content of 30%-40% at 89.10 - 90.00. Greenish, highly altered with pale green chlorite replacing most original calc-silicate minerals, 87.35 - 95.00. Pinkish, not altered mostly garnet - sphalerite, 95.00 - 97.00. No scheelite.
97.00	98.25	GARNET-SKARN, similar to above but with only traces of sphalerite; much pale green needle like hexagonal mineral, easily scratched with knife, with (0001) cleavage - ? apatite? No scheelite.
98.25	103.25	SPHALERITE-SKARN, fresh, as above but slightly to moderately altered, with patches of ? apatite. Sphalerite content decreases towards 103.25. No scheelite.
103.25	112.00	SKARN, Frst, slightly to moderately altered, massive, similar to above but with little or no sphalerite. Leached in many places. Much ? apatite in places. No scheelite. (Rock is actually calc-silicate, as logged in other holes, but is here called skarn because of the presence of sphalerite i.e. a clear <u>meta-somatic</u> mineral in what otherwise looks like a <u>metamorphic</u> rock).
112.00	113.00	SKARN, fresh, highly to completely altered and very broken at 112.50 - consists mostly of dark green chlorite. No scheelite.
113.00	117.25	CALC-SILICATE ROCK, similar to 103.25 - 112.00, but not leached and with lesser dark calc-silicate minerals. Overall pinkish. Diffuse layering 30° at 114.20. No scheelite.
117.25	123.12	CALC-SILICATE ROCK, fresh, mostly f - m, massive medium to dark green due (?) actinolite. Rock is pinkish in places;locally highly altered, soft and broken e.g. 118.25 - 119.00. Layering 40° at 122.60. Traces disseminated scheelite; and scheelite in veinlets 120.30 - 121.50
123.12	124.00	CALC-SILICATE ROCK, fresh, completely altered, with soft pale chlorite. Cross-cutting veinlets of dark chlorite? and pale zeolite?
124.00	182.25	SILTSTONE-SANDSTONE, fresh, variable in texture and grain size and colour. Darker and harder 134.50 - 140. Soft buff sandstone, probably altered and SW, 162.50 - 168.25. A few minor beds of originally calc-siltstone - now f greenish

221383

calc-silicate rocks variously altered, up to 15 cm wide. Network of typical alteration microveinlets throughout but especially 132.00 - 134.00. A little pyrite on some joints. Quartzveins and veinlets:-

128.75 - 130.00 at 10° - 1 cm drusy quartz and clear fluorite.
131.00 - 131.50 sub parallel core - 1 cm quartz + wolframite, a little scheelite
141.00 - 143.00 sub parallel core - 1-2 cm drusy quartz + fluorite + wolframite + ? apatite, a little molybdenite. The ? apatite is soft pale green delicate needle like crystals.
159.25 sub parallel core - quartz + a little molybdenite.
160.75 - 161.50 sub parallel core - 5 cm drusy quartz + wolframite + a little scheelite
168.25 - 179.25 some patchy quartz containing a little pyrite, chlorite, mica Some greisen selvedges

Bedding: fine, 50° at 127.00
80° at 137.50
70° at 146.50
80° at 151.50
indistinct 30° at 175.50
distinct 50° at 182.25

END OF HOLE

221354

PROJECT: MOINA
HOLE NO: SMD 15
DRILLED: JANUARY 1977

CORE SIZES: 0 - 49.35 NQ
49.35 - 116.12 BQ

LOGGED BY: P. ASKINS
ORIENTATION: VERTICAL
CO-ORDS: 900E 18N

FROM	TO	DESCRIPTION
0	20.30	BASALT, SW-MW 0 - 12.5, Frst 12.5 - 20.30, agglomeratic, scoriaceous.
20.30	23.30	WRIGGLITE, HW-CW, very limonitic, has relict wrigglite texture. Possibly boulder.
23.30	25.50	MUDSTONE, soft semi-consolidated, pale bluish and finely laminated, irregular bedding in places, broken by drilling or brecciated.
25.50	28.50	no core
28.50	33.00	SANDY SILT, unconsolidated
33.00	42.00	CALC-SILICATE ROCK + WRIGGLITE, SW-MW to 34.15, Frst 34.15 to 42.00, variable proportions each type, network of fractures, and veinlets of pink felspar. Much disseminated f scheelite in places; with a little in veinlets. Relict bedding 80° at 41.
42.00	43.60	CALC-SILICATE ROCK + WOLLASTONITE ROCK, Frst, network of fractures, nil or traces scheelite.
43.60	86.30	WRIGGLITE, Frst, f-f, a few pink felspar veinlets. Scheelite in veinlets and disseminated, abundantly in places. Green f calc-silicate rock 49.00 - 40.35 with bedding 60° at 49.00. Calc-silicate predominates 51.00 - 51.50; 55.50 - 56.50; after 71.00; irregular distribution elsewhere. Much very dark f chlorite in places where rock is not so magnetic (Are these SnO_2 rich zones?) Dark green chlorite or actinolite rich 76 - 77. Relict bedding 70° at 75.
86.30	90.00	CALC-SILICATE ROCK, fresh, variable colour and texture. Wollastonite rich to 87.50; altered with dark khaki chlorite to 88.50, and 89.20 - 89.40. A little wrigglite after 89.00. Little or no scheelite in wollastonite rich and dark khaki chlorite altered zones. Probably relict bedding 80° at 87.50.

221355

90.00	100.70	WRIGGLITE, fresh, interbedded with increasing calc-silicate layers towards base. Typical disturbed chaotic bedding 80° throughout. Much f disseminated scheelite, and coarse scheelite in veinlets e.g. 2 cm wide, coarse scheelite, with a little molybdenite, at 90.10.
100.70	109.25	CALC-SILICATE ROCK, fresh, typical variable textures. Typical network fine dark micro-veinlets. Irregular metasiltstone defines bedding at 80°. Rare wrigglite patches. Scheelite mainly confined to veinlets, especially in pink felspar veinlets.
109.25	116.12	METASILTSTONE, fresh, f-m, pale to medium greenish grey. Network fine dark microveinlets especially to 112.00 and 115.00 - 116.12. 2 cm quartzvein with a little wolframite at 30° at 114. Bedding 85° at 114.50.

END OF HOLE

221356

PROJECT: MOINA
HOLE NO: SMD 16
DRILLED: FEBRUARY 1977

CORE SIZES: 0 34.55 BQ
34.55 - 171.00 NQ

ORIENTATION: VERTICAL

LOGGED BY: P.ASKINS

CO-ORDS: 1450E 125S

FROM	TO	DESCRIPTION
0	4.25	SANDSTONE, SW, m, off white, siliceous; no obvious bedding; pyrite on some joints. No scheelite.
4.25	5.95	CALC-SILICATE ROCK, CW, soft, ochreous
5.95	7.80	SANDSTONE, SW, m, brown weathered equivalent of rock below. No scheelite
7.80	18.90	SANDSTONE-METASILTSTONE, Frst, f-m, variable wispy textures, altered with patches of weathered soft chloritic calc-silicate. Very broken in most places. Pyrite on joints, and a little disseminated. Pyrrhotite disseminated at 9.00. Bedding 20° at 9.00, 45° - 60° at 18.70. Network dark microfractures and veinlets 14.25 - 15.25; probably occur elsewhere but are weathered. Trace disseminated shceelite in veinlets at 8.85 and 10.00. Soft puggy khaki chlorite at 15.75 with disseminated pyrite, sphalerite (resembles sphalerite but streak is black)
18.90	24.33	METASILTSTONE, Frst to 22, fresh to 24.33, f-m, grey, finely bedded 45° - 60° throughout. A little disseminated pyrite, and pyrite on joints. Some dark microfractures and veinlets. Core very broken in places. Scheelite absent.
24.33		FAULT
24.33	26.27	SANDSTONE, HW-CW, m, cream, soft.
26.27	30.22	CALC-SILICATE ROCK, SW-CW, altered and weathered equivalent of rock below. Core very broken, soft, 28.27 - 29.27. Much disseminated sphalerite 27.30 - 28.10; lesser amounts elsewhere.
30.22	42.35	CALC-SILICATE ROCK, Fresh, to SW in places before 33.00, m, pale greenish, weak wispy layering usually rich in magnetite in places. e.g. 50° - 80° at 36.30. Sheeted system of microfractures and pink felspar veinlets up to 5 cm wide at about 30°. Coarse scheelite in these fractures and veinlets. Sphalerite disseminated and in patches, (m-crs, black), 38.00 - 42.35.
42.35	50.20	WRIGGLITE, fresh, coarser grained than most; much very dark green chlorite in places (where wrigglite is relatively non magnetic). Many pink felspar veinlets in various orientations, with a little disseminated coarse scheelite. Much f disseminated scheelite in places.

221357

50.20	57.20	CALC-SILICATE ROCK, fresh, m - crs, pale green, mostly massive with trace disseminated sphalerite, elsewhere with wispy dark green chlorite. Some wrigglite at top and bottom where there are also pink felspar veinlets. A little disseminated scheelite.
57.20	65.50	WRIGGLITE, fresh, similar to above but less scheelite. Pink felspar veinlets similar, but very intense after 61.20; after 61.20 wrigglite is not so magnetic and more chloritic; tends to calc-silicate in places.
65.50	68.10	CALC-SILICATE ROCK, fresh, pale greenish, m, with variable amounts disseminated coarse black sphalerite, over 50% in patches. Softer and leached in places. Scheelite virtually absent.
68.10	70.00	SKARN, fresh, with irregular about 5 cm wide pink felspar vein sub parallel to core; Consists of irregular m pale calc-silicate minerals, magnetite, sphalerite and f chlorite. A little scheelite in veinlet.
70.00	about 74.00	CALC-SILICATE ROCK, as at 65.50 - 68.10. Rare scheelite.
74.00	81.60	CALC-SILICATE ROCK and SKARN similar to 65.50 - 70.00 but very broken, leached, altered. Sphalerite content very low. Scheelite confined to felspar veinlets.
81.60	85.10	WRIGGLITE, fresh, typical type, only a few felspar veinlets, with a little disseminated scheelite. Chlorite rich at 83.00; also at 83.00 drusy quartz-mica vein 2 cm wide at 30°.
85.10	87.90	CALC-SILICATE ROCK as at 65.50 - 68.10 Scheelite nearly absent.
87.90	88.70	WRIGGLITE & CALC-SILICATE ROCK, fresh. rigglite as irregular patches in calc-silicate. Scheelite nearly absent. Some minor pink felspar veinlets.
88.70	99.00	CALC-SILICATE ROCK, fresh as at 65.50 - 68.10; some dark chlorite alteration and zeolite filled fractures, 90.00 - 92.00. Pink felspar vein about 3 cm wide sub-parallel to core containing coarse wolframite, scheelite, a little f molybdenite, 88.70 - 89.50.
99.00	104.27	CALC-SILICATE ROCK, as above but with around 20% sphalerite 99.00 - 101.70 and around 5 - 10% to 104.27. A little disseminated scheelite. NOTE: Pink fels veinlets rare in Calc-silicate rock.
104.27	120.50	WRIGGLITE, fresh, typical type. Chlorite rich in places. Felspar veinlets at high angles to core axis. Much disseminated f scheelite, rarer scheelite in veinlets.
120.50	125.43	CALC-SILICATE ROCK, fresh pale greenish, m - crs, massive, with variable amounts of disseminated coarse magnetite. Felspar veinlets rare. Scheelite virtually absent. Contact at 120.50 gradational.
125.43	128.75	WRIGGLITE, fresh, similar to above.

128.75	148.00	CALC-SILICATE ROCK, fresh, massive, pale greenish to pinkish, f-m; colour and grain size differences define banding 60° - 80° to 133.00, and 40° at 153.00. Gradational contact at 148.00. Pink felspar and f chlorite - ? biotite veinlet with scheelite sub parallel to core 2 cm wide, 135.75 - 136.75. Veinlets of pink felspar rare, and scheelite overall virtually absent. Vein 6 cm wide of quartz-mica-wolframite-a little molybdenite, scheelite at 40° at 147.00
148.00	162.00	CALC-SILICATE ROCK, METASILTSTONE, fresh. Calc-silicate as above but irregular patches of greenish-bluish f - f metasiltstone, increasing in quantity towards 162.00. Bluish alteration associated with network of microfractures. A little scheelite in veinlets of pink felspar. Greisen like veinlets in places Veinlets around 5 cm wide at 30° of quartz-topaz at 150.27, 152.50. Contact at 162.00 gradational. Bedding 45° - 70° at 159.00.
162.00	171.00	SILTSTONE - SANDSTONE, fresh, interbedded, various grey shades. Network of microfractures decreases towards 171.00. Quartz felspar veinlet 1 cm wide, 30° at 170.35. Rare scheelite in veinlet at 162.50. Bedding 20° at 164.00, 70° at 165.00, 60° at 168.00, 50° at 170.00.

END OF HOLE

PROJECT: MOINA

HOLE NO: SMD 17

DRILLED: APRIL 1977

CORE SIZES: 0 - 24.20 NQ
24.20 - 74.15 BQ

ORIENTATION: VERTICAL

LOGGED BY: P. ASKINS

CO-ORDS: 810W 60S

From	To	Description
0	19.50	SOIL (no core recovered)
19.50	20.75	LIMESTONE, fresh, f, pale to dark grey - irregularly bedded and tends to be stylolytic parallel to bedding. Dark layers are more silty and ? dolomitic and contain a little disseminated pyrite. Bedding 70° - 90°.
20.75	21.80	SANDSTONE, fresh, m, off-white; slightly calcareous, f matrix.
21.80	23.40	LIMESTONE, fresh, f as above but in places tends to be m.
23.40	25.25	SANDSTONE, fresh, as above but more metamorphosed, with patches up to 3 cm of pale brown m, garnet or white f wollastonite.
25.25	26.20	LIMESTONE, fresh, as at 21.80 - 23.40, but with chert (?) layers up to 7 cm thick, white f, with f disseminated pyrite or pyrrhotite in streaks parallel to bedding. "Chert" may be actually a quartzite of metamorphic origin - does not effervesce with HCl. Bedding 70° - 90°.
26.20	26.50	CALC-SILICATE ROCK + LIMESTONE. Transition to skarn below. Calc-silicate is m, buff, replaces dark ? dolomitic calc-siltstone layers. Limestone recrystallizes and becomes whiter. Bedding 70° - 90°.
26.50	34.00	CALC-SILICATE ROCK/WRIGGLITE: fresh, variable proportions. Wrigglite replaces original pure limestone and calc-silicate is after ?dolomitic calc-siltstone - overall fabric reflects original distrubed bedding at 70°. Traces of scheelite in veinlets and disseminated. Some f green chlorite replaces (?) calc-silicate at 27.00 - 27.40.
34.00	35.15	CALC-SILICATE ROCK, fresh, f, Fabric resembles f calcareous sandstone.
35.15	38.90	METASILTSTONE/SANDSTONE/CALC-SILICATE ROCK, Variable proportions, fresh, bedded 70°, network of microfractures. Traces scheelite in veinlets. Pyrite common in microfractures. Wrigglite occurs in minor amount.
38.90	41.90	WRIGGLITE/CALC-SILICATE ROCK, similar to above.
41.90	42.60	CALC-SILICATE ROCK, fresh, f-m, buff to greenish.
42.60	49.90	METASILTSTONE/SANDSTONE etc, fresh, as at 35.15 - 38.90. Only trace scheelite in veinlets. Network of microfractures prominent especially to 45.50. Bedding 70°. Pink zeolite in some joints.
49.90	74.15	SANDSTONE/SILTSTONE, fresh, relatively unaltered, little fracturing, pink zeolite with a little pyrite common in some joints. Scheelite absent in most places, trace along veinlets especially 70 - 72. Bedding 50° - 70°, mostly 70°.

END OF HOLE

PROJECT: MOINA
HOLE NO: SMD 20
DRILLED: JULY 1977

CORE SIZES: 0 - 75.50 NQ
75.50 - 94.75 BQ

LOGGED BY: P.ASKINS.
ORIENTATION: Vertical
CO-ORDS: 300W 275S

FROM	TO	DESCRIPTION
0	12.0	SILTY SOIL
12.0	13.30	GRAVEL (only recovered 20 cm) of f grey hornfels or chert
13.30	around 22.80	MUDSTONE - SILTSTONE, off white - buff - grey, semi-consol. Tertiary sink hole fill.
round 22.80	41.25	LIMESTONE - 22.80 - 23.40 MW - HW, soft, white, broken 23.40 - 24.50 Fr well bedded 70° - 90°, grey 24.50 - 26.25 Fr as above but locally HW especially along sandy and silty layers. Some bioturbation. Bedding 70° - 80°. 26.25 - 34.50 MW, soft, in places leached, off white to stained yellowish Possibly overall a more silty limestone originally. 34.50 - 37.10 Frst - SW. Bioturbated bedding at 45° - 60°. 37.10 - 41.25 MW - HW, leached, otherwise as above.
41.25	53.60	MUDSTONE - SILTSTONE, greys, grey - green, buff etc. Semi-consol. Bedded 80°. Tertiary to recent cave-fill.
53.60	67.00	CALC-SILICATE ROCK, CW, orange-buff, soft, poor recovery, no scheelite.
67.00	67.65	CALC-SILICATE ROCK, MW - HW, buff to pale greenish. Relict banding at 45° - 60°. Some effervescence with HCl indicates relict limestone or secondary calcite. No scheelite.
67.65	69.60	CALC-SILICATE ROCK and METASILTSTONE, SW - MW, pale greenish to grey, f, bedding 80°, no scheelite.
69.60	72.00	METASILTSTONE, Fr, grey to pale brown, f, finely bedded 70° - 90°. No scheelite.
72.00	73.30	QUARTZITE, Fr, very pale greenish, f - m, massive, Disseminated f py in places. (Metamorphosed f sandstone.) No scheelite.
73.30	75.20	SILICIFIED SILTSTONE, Fr, pale greenish or buff, or pink where very siliceous, f. Pink coloured material (73.90 - 74.30) is recemented breccia with blobs po and disseminated po. Bedding not recognizable except 80° at 73.30. No scheelite.

75.20	94.75	SILTSTONE - METASILTSTONE, Fr, similar to other holes. Bedding 80° - 90° to 85.00; 85.00 - 89.50 silicified, very mottled, some veinlets with po at 45°. Network of veinlets of calcite - white zeolite overall at 15° - 20° after 91.75. No scheelite.

END OF HOLE

PROJECT: MOINA
HOLE NO: SMD 21
DRILLED: Sept 1977

CORE SIZES: 0 - 10 HQ
10 - 16 NQ
16 - 155.25 BQ

LOGGED: P.ASKINS
ORINETATION: Vertical
CO-ORDS: 00/75S

From	To	Description
0	105.70	LIMESTONE, 0 - 10 Frst; 10 - 104.70 Fr. No Significant concentrations of dark dolomitic silty layers. For details see log of P.W. STAINTON.
105.70	112.50	CALC-SILICATE ROCK, Fr, pale to dark greenish, m. Consists of pale diopside, dark actin, two types pale green gar in various proportions. Trace disseminated po. Layering 60° at 106.25, 108.25, 112. Layering very distinct 110.75 - 112.50. No scheelite.
112.50	122.10	CALC-SILICATE ROCK + WRIGGLITE, Fr. Calc-silicate is similar to above with patches of wrigglite at 112.50 - 112.85, 113.10 - 113.25, 115.50 - 115.80, 116.50 - 117.25 and very minor blebs elsewhere. Minor pink fels veinlets (with no scheelite) in and near wrigglite at 116.50 - 117.25. No scheelite except minor disseminated f scheelite 121.25 - 122.10.
122.10	124.25	WRIGGLITE, Fr, typical type. A few pink fels veinlets. A little scheelite, disseminated and in veinlets
124.25	155.25	METASILTSTONE - SANDSTONE, Fr, similar to that in other holes. Features of note: 124.25 - 127.25 variable texture and colour, greenish in places, many microfractures and dark veinlets and trace scheelite in pale veinlets. 127.25 - 132 finely bedded in many places, 70° 128; 70° 131. After 132 predominently sandstone, browns, greys in places mottled and greenish bedding 60° 136; 70° 144; 50° 153; 134.50 - 135.40 pale green sandstone (colour due pale pyroxene?) with cross cutting network of veinlets with thin (mostly 2 mm but up to 1 cm) dark alteration selvedge (these veinlets no doubt occur in adjacent rocks but are not visible because the rocks contain no pyroxene to be altered to dark bio chl amphibole etc). 136 bedding 60° 138.75 5 cm qtz - wolframite - scheelite vein, Contact at 30°. 139 3 cm qtz - sphalerite (pale brown) - epidote - py vein at 70°. A little f-m py disseminated in places. Very minor scheelite in veinlets throughout.

PROJECT: MOINA

HOLE NO: SMD 22

DRILLED: SEPTEMBER 1977

CORE SIZES: 0 - 19.80 No core
19.80 -58.00 NQ
58.00 -110.00 BQ

LOGGED: P.ASKINS

ORIENTATION: Vertical

CO-ORDS: 100W 275S

FROM	TO	DESCRIPTION
0	19.80	SOIL, (No Core)
19.80	69.40	LIMESTONE Frst to 65.00, Fr beyond, more weathered and leached close to fractures and caves; grey, bedded 60° - 70° to 41.50, 45° - 60° 53.50 - 69.40 (for detailed log of limestone see log of P.W. Stainton). Caves commencing at 23.60 up to 58.00 (see recovery log for most affected areas). Clay and rubble cave fill 25.90 - 26.20, 41.50 - 45.00, 47.50 - 49.00. Clay is mostly silty, greenish grey, yellowish due iron staining in places. Core very broken after 41.50.
69.40	74.00	CALC-SILICATE ROCK, Fr, olive green to green, m - crs, relict bedding as compositional irregular layering. Consists of gar. diopside, actin, a little calc, po. Gradational contact. 73.60 - 74.30, where sandy dolomitic beds (0.5 - 2 cm thick) are replaced by calc-silicates, and purer limestone beds are slightly bleached. No scheelite.
74.00	78.40	LIMESTONE, Fr, grey, with bioturbated 0.5 - 5 cm beds of off-white sandy dolomitic limestone at 45° - 60°. Gradational contact 78.00 - 78.40 with calc-silicate rock below; similar to gradational contact above. No scheelite.
78.40	87.85	CALC-SILICATE ROCK, Fr, similar to above but overall more pinkish due to more garnet. Relict bedding 50°-70°. No scheelite.
87.85	89.70	LIMESTONE, Fr, grey, with 1 - 3 cm beds replaced by pale calc-silicate minerals; and a few wrigglite stringers towards 89.70.
89.70	90.70	WRIGGLITE, CALC-SILICATE ROCK, METASILTSTONE, Fr, interbedded as 0.5 - 10 cm layers. Wrigglite overall about 30%; contains amber sphalerite at 90.60. No Scheelite.
90.70	93.00	METASILTSTONE, Fr, pale green, pink, f. Network of dark veinlets most at about 70°. Tr po in places. Tr scheelite at 91.50.
93.00	110.00	SANDSTONE, SILTSTONE, Fr, similar to other holes, bedding 50°-70°. No scheelite.

END OF HOLE .

PROJECT: MOINA

HOLE NO: SMD 23

DRILLED: OCTOBER 1977

CORE SIZES: 0 - 4 HQ
4 - 14.10 NQ
14.10 - 37.00 BQ

LOGGED: P. ASKINS

ORIENTATION: Vertical

CO-ORDS: 200W/675S

FROM	TO	DESCRIPTION
0	6.00	BASALT, Frst - CW, scoriaceous and agglomeratic
6.00	15.64	WRIGGLITE, CALC-SILICATE ROCK, Fr, mostly interbedded 1 - 5 cm; thickest calc-silicate layer 12.90 - 13.60. Wrigglite is dark green, f - f, very actin (?) rich with variable pyrite. Calc-silicate is pale greenish, garnet rich, f - crs. Py overall in wrigglite + calc-silicate about 1 - 2%. Original bedding 70° - 80°. A little fels veining. Slickensided fracture 20° at 7.70. Brecciated thin fracture 30° at 13.40. Scheelite dissem and in veinlets.
15.65	18.80	WRIGGLITE, CALC-SILICATE ROCK, METASILTSTONE, Fr. Wrigglite is dark green with very little mag and tends to f actin-chl? in places. Metasiltstone is grey, f - f almost cherty in places, very irregularly distributed. 15.65 - 18.10 fractured, softer, altered. Original bedding 70°. Trace dissem scheelite.
18.80	20.60	METASILTSTONE, Fr, pale grey and pale cream, f - m.
20.60	27.00	CALC-SILICATE ROCK, METASILTSTONE, Fr, interbedded, generally 1 - 4 cm irregular layers. Metasiltstone predominent 24.00 - 27.00. After 24.00 very fractured, softer, altered Bedding 70° - 80°. A few fels veinlets. Tr scheelite dissem and in veinlets.
27.00	37.00	METASILTSTONE, SANDSTONE, Fr, similar to other holes. Very fractured 27.00 - 28.50; 29.00 - 29.50; 31.10 - 32.50; 34.25 - 34.50; 35.20 - 35.60; 36.30 - 36.50. Scheelite in veins in fractures 28.50, 29.30, at about 20°.

END OF HOLE

APPENDIX 25

ANALYSES OF SPLIT CORE

All available analyses for diamond drill holes of Mines Department, Mt. Lyell Mining and Railway Co. Ltd and Comalco Limited.

Much of the core has been scanned with a McPhar TV1 scintillometer; no zones of high radiation count have been found.

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L. 7/74 "MOINA" SHEPHERD & MURPHY MINE AREA.

Analyses of split core

Bore No. DOM 1.

221366

Depth From	Metres To	P ppm	Sn ppm	WO_3% or W ppm	Bi ppm	Mo ppm	Be ppm	Pb ppm	Zn ppm	Cu ppm	Ag ppm	Ni ppm	Co ppm	Sb ppm	Au ppm	Ta ppm	V ppm	Fe %
22.00		230						65	35	15		25			-0.10			
24.70		280			45	35		65	25	-10	8	15			-0.10			
26.00		540			35	-10		70	30	15	6	30			-0.10			
28.40		270			40	-10		60	30	15	8	30			-0.10			
32.20		250			45	-10		60	20	-10	8	15			-0.10			
33.30		340			35	10		55	20	10	7	40			-0.10			
34.15		220			40	-10		65	15	-10	9	20			-0.10			
35.00		270			40	-10		65	15	-10	8	15			-0.10			
35.70		270			40	-10		70	10	-10	7	25			-0.10			
36.50		230			40	-10		70	60	-10	8	25			-0.10			
37.20		350			35	10		70	50	-10	7	25			-0.10			
38.06		230			40	-10		70	20	-10	8	20			-0.10			
38.96		220			40	10		70	20	-10	8	20			0.10			
39.70		300			40	-10		60	15	-10	6	30			-0.10			
40.20		340			40	-10		65	15	-10	6	20			0.13			
40.60		220			40	-10		65	15	-10	7	20			-0.10			
41.15		620			40	-10		65	15	10	6	20			-0.10			
42.00		400			35	-10		75	20	10	7	15			-0.10			
42.80		520			40	-10		70	20	-10	6	15			-0.10			
43.50		270			40	10		70	15	-10	7	15			-0.10			
44.30		470			35	-10		65	15	-10	7	10			-0.10			
45.20		400			40	-10		70	15	-10	6	20			-0.10			
46.00		540			40	-10		65	15	-10	6	20			-0.10			
46.90		450			40	10		70	15	-10	6	20			-0.10			
47.25		560			40	10		70	20	-10	6	25			-0.10			
49.30		300			35	-10		65	15	-10	7	20			-0.10			
50.90		210			45	10		60	15	-10	9	10			-0.10			
57.47		190			40	-10		55	20	-10	8	15			-0.10			
60.60		380			40	10		55	10	-10	8	10			-0.10			
63.90		210			45	10		55	20	-10	8	-10			-0.10			
67.20		250			35	-10		50	10	-10	7	35			-0.10			
70.50		290			35	-10		50	15	-10	6	25			-0.10			
74.10		300			40	-10		60	20	-10	6	20			0.10			
77.20		220			40	-10		45	15	-10	6	20			-0.10			
80.40		690			35	-10		50	20	-10	6	20			-0.10			
82.78		440			30	-10		60	15	-10	7	20			-0.10			
86.90		300			35	-10		60	20	-10	6	20			-0.10			
90.40		340			35	-10		70	20	-10	8	20			-0.10			
94.16		190			30	-10		65	15	-10	7	15			-0.10			
97.00		390			35	-10		70	10	-10	8	20			-0.10			
100.00		440			30	-10		35	20	10	4	50			-0.10			
103.20		250			25	10		30	-10	-10	3	155			-0.10			
107.10		900			30	-10		55	10	-10	6	40			-0.10			

Mineral Analysis data from bore core samples

Depth From	Metres To	F ppm	Sn ppm	W ppm	Bi ppm	Mo ppm	Be ppm	Pb ppm	Zn ppm	Cu ppm	Ag ppm	Ni ppm	Co ppm	Sb ppm	Au ppm	Ta ppm	V ppm	Fe %
110.40		520			25	-10		40	10	-10	5	45			0.12			
115.50		170			35	-10		60	15	-10	8	25			-0.10			
118.10		460			30	-10		45	10	10	6	35			-0.10			
120.35		260			30	-10		55	10	-10	6	30			-0.10			
123.50		200	-1	-50	(-1) 35	(-3)-10	-1	60	15	-10	(-0.1) 6	25	-5	-30	-0.10	-100	-10	
126.60		620			30	-10		60	20	-10	6	35			-0.10			
129.90		520			30	-10		55	15	-10	6	30			-0.10			
133.00		130			35	-10		45	-10	-10	6	55			-0.10			
136.50		380			35	-10		60	20	-10	6	25			-0.10			
139.70		660			25	-10		45	-10	-10	5	35			-0.10			
143.00		220			30	-10		70	25	-10	6	25			-0.10			
147.20		220			30	-10		60	10	-10	5	30			-0.10			
149.50		140			35	-10		60	60	-10	6	20			0.12			
153.50		280			25	-10		60	95	-10	6	20			0.12			
154.45		180			30	-10		60	30	10	6	20			0.11			
155.20		180			15	-10		25	30	-10	1	75			-0.10			
156.00		400			25	10		60	35	15	3	60			0.13			
157.00		520	1	-50	(-1) 25	(-3)-10	1	60	55	-10	(0.1) 5	90	-5	-30	-0.10	-100	20	
157.80		310			30	-10		60	45	-10	7	20			-0.10			
158.00		280			30	-10		70	60	-10	6	25			-0.10			
159.80		500			25	-10		65	55	10	6	25			0.10			
160.30		370	-1	-50	(-1) 30	(-3)-10	-1	60	15	10	(-0.1)6	35	-5	-30	-0.10	-100	10	
161.00		460			30	-10		60	15	10	6	30			-0.10			
161.80		480			30	-10		60	200	10	6	35			-0.10			
162.80		400			25	-10		75	15	15	7	70			-0.10			
163.54		300	1	-50	(-1) 25	(-3)-10	-1	50	45	15	(0.1) 6	30	-5	-30	-0.10	-100	-10	
164.30		370			30	10		60	20	10	5	25			-0.10			
165.10		1150			30	-10		40	25	20	5	85						
166.00		1200			35	-10		30	20	15	6	75						
167.00		700	-1	-50	(-1) 45	(-3)-10	-1	30	15	15	(0.1) 6	40	5	-30	(-3)	-100	20	
167.70		900			30	-10		30	15	20	4	75						
168.50		960			40	-10		40	15	15	6	55						
169.30		780			40	-10		30	15	10	6	45						
170.20	171.50	0.31%	0.01%*	-10	100	25		25	415	50	4				Nil*			3.0% *
171.50	172.80	0.39%	0.02%*	-10	110	-10		35	90	-10	3				Tr.*			6.2% *
172.80	173.60	0.25%	-0.01%*	-10	55	10		25	35	-10	2				Nil*			2.0% *
173.60	174.45	0.08%	-0.01%*	-10	25	-10		20	20	-10	-1				Nil*			0.4% *
174.80	175.70	0.28%	0.03%*	-10	95	-10		15	55	-10	3				Nil*			7.1% *
175.70	176.80	0.84%	0.03%*	20	130	-10		30	75	-10	3				Nil*			7.0% *
176.80	177.30	8.10%	0.03%*	30	1300	15		685	655	135	8				Tr.*			16.3% *
177.30	177.80	1.02%	0.02%*	15	300	-10		90	60	10	4				Tr.*			8.8% *
177.80	178.75	2.90%	0.03%*	20	735	-10		125	120	85	4				Tr.*			10.7% *
180.00		2000	3	-50	(1) -10	(5) 15	5	10	25	-10	(0.1) -1	215	-5	-30	(-3)	-100	10	
181.10		1700			-10	15		15	30	10	-1	200						
181.90		1420			-10	10		15	25	25	1	180						

221368

E.L. 7/74 MOINA - DOM 1.

Mineral Analysis data from bore core samples

Depth From	Metres To	F ppm	Sn ppm	WO_3 % or W ppm	Bi ppm	Mo ppm	Be ppm	Pb ppm	Zn ppm	Cu ppm	Ag ppm	Ni ppm	Co ppm	Sb ppm	Au ppm	Ta ppm	V ppm	Fe %
182.90		1130			10	10		25	30	30	1	205						
183.80		900	1	-50	(-1) -10	(3) 10	3	10	25	20	(0.1) -1	190	30	-30	(-3)	-100	200	
184.40		1200			-10	10		10	25	15	1	215						
185.30		940			20	10		-10	25	20	2	195						
186.05		960			15	10		-10	25	30	1	195						
186.85		1200	1	-50	(-1) 20	(+3) -10	1	-10	25	15	(-0.1) 2	185	-5	-30	(-3)	-100	30	
187.70		1600			-10	10		-10	20	25	-1	230						
188.50		2700			15	15		-10	30	65	1	250						
189.30		6600			25	15		-10	55	50	1	240						
190.10		5100	10	-50	(50) 80	(-3) 10	20	40	75	155	(0.1) 2	165	100	-30	(-3)	-100	50	
193.20		940			20	10		10	35	10	2	225						
196.60		820			15	-10		-10	35	-10	1	135						
199.77		1020			20	-10		-10	30	-10	-1	135						
202.90		310	1	-50	(-1) -10	(3) 10	-1	-10	15	-10	(0.1) -1	205	-5	-30	(-3)	-100	20	
DETECTION LIMIT			(1)	(50)	(10)	(10)	(1)	(10)	(10)	(10)	(1)	(10)	(5)	(30)	(0.10)	(100)	(10)	
ANALYSES BY WHOM AND METHOD		COMALCO FUSION SIE	AMDEL SEMI-QUANTITATIVE SPECTRO		COMALCO AAS		AMDEL SEMI-QUANT SPECTRO	COMALCO AAS					AMDEL SEMI-QUANT SPECTRO		COMALCO AAS	AMDEL SEMI-QUANT SPECTRO		MINES DEPT

NOTES: (1) Samples collected by Comalco are chips less than 20 cm long.

(2) * Analyses by Mines Department (see Collins 1975 Tas.Dept. Mines Tech. Rept.18) Comalco analyses of these intervals are of pulps supplied by Mines Dept.

(3) Figures in brackets ( ) are semi-quantatitive spectrographic analyses by Amdel

(4) Be spectrographic analyses may be too low by a factor of about 10.

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L. 7/74 "M[illegible]A", IRIS RIVER AREA

221369

Analyses of split core

Bore No. DOM 2.

Depth From	Metres To	F ppm	Sn ppm	W ppm	Bi ppm	Mo ppm	Be ppm	Pb ppm	Zn ppm	Cu ppm	Ag ppm	Ni ppm	Co ppm	Sb ppm	Au ppm	Ta ppm	V ppm	Fe %
0.00	0.40	4.20%	*.09%	145	785	20		55	175	425	4				Tr.*			* 32.6
0.40	1.06	2.42%	*.09%	110	490	10		160	155	120	3				Tr.*			* 27.3
1.06	1.30	2.70%	*.11%	340	624	10		190	245	165	4				Tr.*			* 17.7
1.30	2.66	5.60%	*.07%	240	1460	15		165	270	405	5				Tr.*			* 25.7
2.66	3.66	1.22%	*.06%	40	165	-10		80	140	45	3				Tr.*			* 10.6
3.66	4.40	.58%	*.05%	-10	145	-10		80	105	35	2				Tr.*			* 6.0
4.40	4.60	2.15%	*.04%	1820	625	30		2650	3385	490	7				Tr.*			* 15.7
4.60	5.45	.98%	*.12%	30	335	-10		120	115	65	3				Tr.*			* 11.2
5.45	5.85	1.33%	*.20%	325	220	-10		115	200	790	4				Tr.*			* 29.8
5.85	6.85	.96%	*.15%	20	480	-10		70	150	75	3				Nil*			* 31.1
6.85	7.88	.62%	*.05%	220	2240	-10		40	160	135	3				Tr.*			* 38.0
7.88	8.93	.36%	*.13%	30	1600	-10		35	115	690	2				Tr.*			* 31.4
8.93	10.07	1.05%	*.09%	30	1520	-10		105	130	970	3				Tr.*			* 24.8
10.07	11.10	.90%	*.10%	45	770	-10		135	140	710	3				Tr.*			* 23.8
11.10	12.15	.22%	*.07%	40	315	-10		50	70	345	2				Nil*			* 16.9
12.15	12.70	2.50%	*.05%	240	510	15		180	110	635	3				Nil*			* 15.8
12.70	13.82	.30%	*.04%	-10	230	-10		25	50	380	2				Tr.*			* 8.2
14.00		7500			870	-10		30	60	550	1	95						
14.80		1200			280	-10		20	35	650	1	125						
16.80		2700	20	-50	200(80)	15(3)		15	50	240	-1(0.1)	245	-5	-30	(-3)	-100	20	
17.60		860			40	20	1	20	45	60	-1	290						
18.00		760			40	20		20	35	95	-1	250						
18.50		830			30	15		30	65	55	-1	210						
19.00		1400	20	-50	40(50)	15(5)	-1	25	55	165	-1(0.1)	190	-5	-30	(-3)	-100	-10	
22.00		900			90	15		15	70	200	1	225						
25.35		500			15	20		15	60	80	-1	265						
29.11		1250			70	-10		15	115	65	-1	125						
32.30		1400	30	-50	20(10)	-10(5)	1	-10	65	360	1(0.1)	180	-5	-30	(-3)	-100	30	
36.40		460			10	10		20	95	70	-1	225						
38.55		280			-10	15		10	50	95	-1	220						
41.60		1040			-10	-10		-10	35	20	-1	90						
44.90		270	1	-50	-10(-1)	10(5)	1	10	50	140	-1(0.1)	165	5	-30	(-3)	-100	20	
48.10		560			-10	15		-10	30	45	-1	235						
DETECTION LIMIT			(1)	(50)	(10)	(10)	(1)	(10)	(10)	(10)	(10)	(10)	(5)	(30)	(0.10)	(100)	(10)	
ANALYSES BY WHOM AND METHOD		COMALCO FUSION SIE	AMDEL SEMI-QUANTITATIVE SPECTRO		COMALCO AAS	AMDEL SEMI-QUANT SPECTRO		COMALCO AAS					AMDEL SEMI-QUANT SPECTRO		COMALCO AAS	AMDEL SEMI-QUANT SPECTRO		MINES DEPT

NOTES: (1) Samples collected by Comalco are chips less than 20 cm long.

(2) * Analyses by Mines Department (see Collins 1975 Tas. Dept. Mines Tech. Rept.18) Comalco analyses of these intervals are of pulps supplied by Mines Dept.

(3) Figures in brackets ( ) are semi-quantatitive spectrographic analyses by Amdel

(4) Be spectrographic analyses may be too low by a factor of about 10

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L. 7/74 "MOINA" IRIS RIVER AREA.

221370

Analyses of split core

Bore No. DOM.3

Depth From	Metres To	F ppm	Sn ppm	W ppm	Bi ppm	Mo ppm	Be ppm	Pb ppm	Zn ppm	Cu ppm	Ag ppm	Ni ppm	Co ppm	Sb ppm	Au ppm	Ta ppm	V ppm	Fe %
10.04	11.04	1.25%	*.03%	15	215	-10		55	175	25	3				Tr.*			* 8.6
11.04	11.97	0.16%	*.02%	-10	105	-10		40	110	30	2				Tr.*			* 3.7
11.97	13.00	0.57%	*.11%	-10	135	-10		65	295	55	3				Tr.*			* 8.2
13.00	14.02	0.97%	*.01%	-10	75	-10		20	290	155	2				Nil*			* 5.6
14.02	15.00	0.15%	*.02%	15	95	-10		40	65	25	3				Nil*			* 6.1
15.00	16.18	0.10%	*.04%	-10	170	-10		55	90	25	4				Tr.*			* 18.6
16.18	17.12	0.13%	*.08%	15	145	-10		75	85	35	4				Tr.*			* 18.9
17.12	18.10	0.15%	*.04%	-10	115	-10		55	115	25	4				Tr.*			* 13.6
18.10	19.10	0.20%	*.05%	-10	95	-10		130	160	15	5				Nil*			* 10.7
19.10	20.17	0.20%	*.05%	-10	155	-10		125	120	15	4				Tr.*			* 10.6
20.17	21.25	0.14%	*.07%	-10	180	-10		165	95	65	4				Nil*			* 10.0
21.25	22.95	0.09%	*.04%	-10	60	-10		350	35	50	2				Tr.*			* 4.9
23.70		460	5	-50	10(1)	10(3)	2	20	50	15	-1 (0.1)	165	-5	-30	-3	-100	10	
24.50		1050			25	30		25	75	20	-1	365						
25.20		1020			15	10		15	50	-10	-1	195						
26.10		1290			-10	-10		-10	45	20	-1	165						
26.90		1450	5	-50	15(1)	10(3)	2	-10	55	45	-1 (0.1)	205	-5	-30	-3	-100	100	
27.70		1200			15	-10		10	45	-10	1	130						
28.50		7400			20	10		10	80	15	-1	185						
29.30		720			25	10		20	330	45	2	240						
29.95		2900	5	-50	15(1)q	15(3)	2	10	110	580	1 (0.1)	235	80	-30	-3	-100	30	
30.10		860			10	10		15	125	40	-1	170						
33.35		1700			10	25		20	65	100	-1	325						
36.65		1150			15	-10		15	40	-10	1	60						
40.00		700	1	-50	15(-1)	20(3)	2	-10	45	25	-1 (0.1)	310	5	-30	-3	-100	50	
43.30		1020			10	-10		-10	55	-10	-1	75						
46.33		1500			15	15		-10	55	25	1	240						
49.50		1000			10	25		10	45	35	ND	435						
52.80		860	1	-50	15(-1)	15(5)	2	20	120	15	-1 (0.1)	260	-5	-30	-3	-100	80	
DETECTION LIMIT			(1)	(50)	(10)	(10)	(1)	(10)	(10)	(10)	(1)	(10)	(5)	(30)	(0.10)	(100)	(10)	
ANALYSES BY WHOM & METHOD		COMALCO FUSION SIE	AMDEL SEMI-QUANT SPECTRO		COMALCO AAS		AMDEL SEMI-QUANT SPECTRO	COMALCO AAS					AMDEL SEMI-QUANT SPECTRO		COMALCO AAS	AMDEL SEMI-QUANT SPECTRO		MINES DEPT.

NOTES: (1) Samples collected by Comalco are chips less than 20 cm long.

(2) * Analyses by Mines Department (see Collins 1975 Tas. Dept. Mines Tech. Rept.18). Comalco analyses of these intervals are of pulps supplied by Mines Dept.

(3) Figures in brackets ( ) are semi-quantatitive spectrographic analyses by Amdel.

(4) Be spectrographic analyses may be too low by a factor of about 10.

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L. 7/74 "MOINA", SHEPHERD & MURPHY MINE AREA

Mineral analysis data from bore core samples

Revised PWA July 1978

Bore No.	Depth From	Feet To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au ppm	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 1A	00.00	00.50		35	-10	.25(3)	(20)	(-3)-10	-10	295	3*	0.37	(0.1) 1.1	(-3)	-30	-100	20	-5	5	
1A	5.00	5.50		30	-10	20		25	10	160		0.28	1.0							
1A	9.75	10.00		30	-10	20		15	-10	55		0.26	1.0							
1A	10.00	10.50		30	-10	25		10	-10	40		0.32	1.1							
1A	14.50	15.00		30	-10	30(1)	(10)	(-3)-10	-10	60	1*	0.16	(0.1) 0.9	(-3)	-30	-100	-10	-5	10	
1A	15.00	15.50		25	-10	15		-10	-10	120		0.15	0.5							
1A	81.00	95.00		25	100	45		500		205		0.10								
1A	95.00	95.50		30	35	45		-10	-10	145		0.11	0.7							
1A	95.50	107.50		30	140	35		75		140		0.09								
1A	107.50	108.00		15	30	20		710	-10	35		0.03	-10							
1A	108.00	109.00		35	85	80		660		80		0.10								
1A	109.00	109.50		30	70	25(1)	(100)	(-3)-10	-10	90	1*	0.09	(-0.1) 0.6	(-3)	-30	-100	10	-5	10	
1A	109.50	111.00		20	100	55		295		70		0.11								
1A	111.00	113.00		20	150	30		1000		75		0.10								
1A	113.00	113.50		40	80	35		-10	-10	145		0.14	1.3							
1A	113.50	117.00		30	200	40		10		160		0.13								
1A	117.00	117.50		20	70	25		-10	-10	155		0.15	1.1							
1A	117.50	121.00		30	130	35		80		145		0.13								
1A	121.00	127.50		45	295	30		55		120		0.13								
1A	127.50	160.00		35	25	35		10		95		0.10								
1A	160.00	160.50		55	30	15		-10	-10	300		0.12	0.6							

120 ppm — Bulked pulp Amdel XRF — To 225

160 ppm — Bulked pulp Amdel XRF — To 225

20 ppm — Bulked pulp Amdel Chemical — To 225

0.010 ppm — Bulked pulp Amdel Spectrographic — To 225

Continued...../2.

E.L. 7/74 "MOINA" -2- Mineral analysis data from bore core samples

Bore No.	Depth From	Feet To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au ppm	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 1A	160.50	162.50		80	65	25		10		290		0.16								
1A	162.50	163.00		35	15	(-1) 15	(1)	(-3) -10	10	110	5*	0.11	(-0.1) -0.5	-3	-30	-100	10	-5	80	
1A	163.00	169.00		45	-10	15		30		110		0.17								
1A	169.00	179.00		40	-10	10		15		95		0.14								
1A	179.00	179.50		50	30	15		-10	-10	130		0.09	-0.5							
1A	179.50	181.50		40	85	15		60		80		0.13								
1A	180.00	181.50		30	15	10		80	20	50		0.19								
1A	181.50	184.00		65	85	15		10		105		0.13								
1A	184.00	186.50		215	-10	15		90		280		0.17								
1A	186.50	190.00		20	-10	-10		30		35		0.14								
1A	190.00	192.00		20	-10	-10		90		20		0.30								
1A	192.00	194.75		20	-10	-10		145		30		0.21								
1A	194.75	197.00		35	-10	20		15		110		0.09								
1A	197.00	200.00		20	-10	15		15		125		0.09								
1A	200.00	202.00		40	40	-10		40		180		0.24								
1A	202.00	204.00		25	-10	15		50		55		0.08								
1A	204.00	206.00		40	190	35		10		140		0.34								
1A	206.00	209.00		40	85	20		65		160		0.08								
1A	208.50	209.00		35	25	20		170	35	130		0.04	0.5							
1A	209.00	212.00		25	10	15		35		100		0.12								
1A	212.00	214.00		30	-10	15		-10		50		0.14								
1A	214.00	216.00		20	-10	-10		15		65		0.12								
1A	216.00	216.50		35	-10	(-1) 15	(1)	(-3) -10	-10	45	3*	0.19	(0.1) -0.5	-3	-30	-100	50	-5	80	
1A	216.50	218.00		25	-10	15		-10		80		0.22								

Bulked pulp 120 ppm To 81; Bulked pulp 160 ppm To 81; Bulked pulp, Amdel chemical 20 ppm To 81; Bulked pulp 0.010 ppm To 81; To 225 (Bulked pulp); To 225 (Bulked pulp); To 225 (Bulked pulp); To 225 (Bulked pulp)

Continued...../3.

E.L. 7/74 "MOINA" -3- Mineral analysis data from bore core samples

Bore No.	Depth From	Feet To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au ppm	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 1A	218.00	219.00		15	25	-10	To 81	25		55	To 81	0.13		To 81						
1A	219.00	225.50		35	20	20		-10		75		0.13								
1A	228.00	228.50		15	15	15		-10	-10	45		0.15	-10							
1A	260.00	260.50		35	-10	15		15	-10	80		0.05	-10							
1A	272.50	273.00		90	-10	25		1190	20	120		0.18	1.3							
1A	280.00	280.50		35	-10	(-1)10	(1)	(5)-10	25	40	1*	0.03	(0.1)-10	(-3)	-30	-100	-10	-5	100	
1A	738.25	738.33		15	-10	-10		-10	10	-10		0.05	-10							
1A	746.50	747.00		15	-10	25		-10	340	-10		0.58	-10							
1A	755.00	755.50		30	10	(5)35	(50) -10	(5) -10	30	30	1*	0.98	(0.1)-10	(-3)	-30	-100	-10	-5	50	
1A	764.00	764.50		-10	-10	15		-10	-10	10		0.13								
1A	774.50	775.00	See note (3)	500	-10	15		-10	265	30		0.35	0.7							
1A	784.33	784.83		65	-10	10		-10	40	400		0.84	-0.5							
1A	798.33	798.83		25	185	20		-10	20	20		0.56	-0.5							
1A	807.33	807.83		20	80	(3)20	(10)	(3) -10	25	-10	3*	0.32	(-0.1)-10	(-3)	-30	-100	-10	-5	50	
1A	826.00	826.50		15	930	-10		-10	-10	-10		0.58	-10							
1A	835.50	836.00		10	240	10		-10	-10	-10		0.39	-10							
1A	845.00	845.50		-10	180	-10		-10	15	-10		0.49	-10							
1A	854.00	854.50		-10	40	(-1)-10	(3)	(200)-10	-10	-10	3*	0.38	(-0.1)-10	(-3)	-30	-100	-10	-5	150	
1A	862.00	862.50		20	25	-10		-10	10	-10		0.50	-10							
1A	866.50	867.00		20	10	-10		-10	20	-10		0.42	-10							
DETECTION LIMIT				(10)	(10)	(10)	(4)	(10)	(10)	(10)		(0.02)	(2)							
METHOD OF ANALYSIS				COM-ALCO AAS	COMALCO COLORI-METRIC	COM-ALCO AAS	AMDEL XRF SCHEME B1	COMALCO AAS			AMDEL SPECTRO	COMALCO FUSION SIE	COM-ALCO AAS	AMDEL FIRE ASSAY	AMDEL SPECTROGRAPHIC SCHEME A1, A2					

NOTES:- (1) Analyses in brackets() are semi-quantitative spectrographic by Amdel.
(2) Be analyses of bulk samples are by an Amdel chemical method. Values of Be marked * may be too low by a factor of about 10.
(3) See separate page for analyses of a separate suite of granite-greisen samples after 774 ft.

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L. 7/74 "MOINA", SHEPHERD & MURPHY MINE AREA

221374

<80

Mineral analysis data from bore core samples

Revised PWA May 78.

Bore No.	Depth From	Feet To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au ppb	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 2	123.50	126.00		15	-10	20		35		85		0.10								
2	126.00	128.50		20	-10	20		-10		80		0.05								
2	128.50	130.00		10	-10	15		-10		65		0.04								
2	130.00	133.00		15	-10	20		-10		65		0.04								
2	133.00	135.00		35	-10	25		-10		90		0.05								
2	135.00	139.83		40	25	50		-10		130		0.09								
2	139.83	140.50		50	105	145		-10		560		1.70								
2	140.00	140.50		25	100	105		-10	165	615		1.42	1.3							
2	140.50	142.00		30	35	60		-10		90		2.10								
2	142.00	143.00		40	30	425		-10		140		10.60								
2	143.00	143.50		35	75	230		15	30	90	240	9.65	1.6	20						
2	143.50	145.50		35	-10	350		10		120		5.10								
2	145.50	147.00		45	-10	715		-10		65		2.00								
2	150.25	150.83		35	50	75		-10	15	65		12.20	1.4							
2	151.00	153.00		50	215	1275		10		145		10.20								
2	153.00	155.00		30	115	350		10		90		11.30								
2	154.83	155.25		25	820	1600		25	240	170		14.40	1.8							
2	155.50	157.50		45	90	750		10		105		10.20								
2	157.50	159.00		60	-10	950		10		90		7.90								
2	159.00	159.50		25	130	310		15	280	120		10.80	1.0							
2	159.50	161.50		45	70	510		10		90		11.00								
2	161.50	163.00		35	85	550		10		95		10.30								
2	163.00	163.50		30	118	430		-10	100	135	350	11.90	1.8	132						
2	163.50	165.50		40	20	390		-10		125		8.1								

25 ppm — bulked pulp — to 272 bulked pulp

1350 ppm — bulked pulp — to 272 bulked pulp

20 ppm — bulked pulp — to 272 bulked pulp

270 ppm — bulked pulp — to 272 bulked pulp

0.010 ppm — bulked pulp — to 272 bulked pulp

0.10 ppm — bulked pulp — to 272 bulked pulp

Continued...../2.

E.L. 7/74 "MOINA"

Mineral analysis data from bore core samples

Bore No.	Depth From	Depth To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au ppb	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 2	165.50	168.00		40	-10	325		10		125		6.2								
2	168.00	168.50		30	35	85		10	135	600		6.4	2.0							
2	168.50	170.50		45	70	350		-10		280		4.9								
2	170.50	172.50		40	35	135		15		295		2.2								
2	172.50	173.00		35	135	420		10	40	135		9.2	1.8							
2	173.00	175.00		35	20	280		-10		155		4.8								
2	175.00	180.00		40	20	335		-10		110		5.3								
2	180.00	180.50		35	130	805		-10	240	65		11.3	2.1							
2	180.50	182.50		45	-10	460		10		165		6.1								
2	182.50	184.00		65	60	1010		-10		95		2.1								
2	184.00	184.50		35	115	505		-10	715	2660	136	8.4	2.0	120						
2	184.50	186.50		50	40	490		-10		325		5.5								
2	186.50	188.00		40	95	370		-10		140		10.0								
2	188.00	188.50		70	65	2000		-10	55	240		7.8	1.5							
2	188.50	190.50		30	40	295		-10		130		5.6								
2	190.50	193.00		30	30	750		-10		115		2.5								
2	193.00	193.50		35	20	695		-10	25	140		5.2	1.4							
2	193.50	195.50		45	235	685		-10		125		8.7								
2	195.50	198.00		35	260	700		10		325		10.0								
2	198.00	198.50		25	55	695		20	50	180		5.8	1.0							
2	198.50	200.00		30	35	285		-10		105		4.6								
2	200.00	203.00		35	295	80		-10		145	250	10.1		40						
2	203.00	203.50		30	400	565		15	65	135		9.4	1.7							

bulked pulp — 450 ppm — to 142 — to 272

bulked pulp — 1350 ppm — to 142 — to 272

bulked pulp — 270 ppm — to 142 — to 272

bulked pulp — 0.10 ppm — to 142 — to 272

Continued...../3.

221376

E.L. 7/74 "MOINA"

Mineral analysis data from bore core samples

Bore No.	Depth From	Feet To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au ppb	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 2	203.50	205.50		30	170	320		-10		125		8.70								
2	204.00	204.50		25	380	205		15	20	180		14.00	1.4							
2	205.00	209.00		25	165	380		-10		160		8.40								
2	209.50	211.50		40	275	500		15		125		10.4								
2	211.50	214.00		35	425	450		20		125		10.6								
2	214.00	214.50		25	310	145		10	15	180		11.5	1.2							
2	214.50	216.50		40	500	560		15		430		9.7								
2	216.50	218.00		35	385	650		15		330		8.2								
2	218.00	218.50		25	510	450		25	50	320		13.0	1.4							
2	218.50	220.50		35	730	585		15		275		10.4								
2	220.50	223.00		40	375	550		15		335		8.7								
2	223.00	223.50		35	420	720		25	170	310	250	4.2	1.2	64						
2	223.50	225.50		45	350	500		20		255		9.7								
2	225.50	227.00		40	380	610		35		110		10.0								
2	227.00	227.50		25	520	730		80	460	155		11.8	1.5							
2	227.50	229.50		30	405	375		10		130		10.6								
2	229.50	232.00		40	500	600		30		90		11.1								
2	232.00	232.50		35	115	390		30	25	105		4.2	1.3							
2	232.50	234.50		55	365	860		25		85		4.8								
2	234.50	236.00		45	270	750		50		100		8.5								
2	236.00	236.50		35	465	1400		225	25	145		3.6	1.6							
2	236.50	238.50		45	260	1055		55		110		8.3								
2	238.50	241.00		30	290	365		25		90		9.1								
2	241.00	241.50		35	480	260		35	50	160	400	9.0	1.6	-50						

Pb ppm: to 142 — bulked pulp 450 ppm — to 272

Sn ppm: to 142 — bulked pulp 1350 ppm — to 272

Be ppm: to 142 — bulked pulp 270 ppm — to 272

Au ppb: to 142 — bulked pulp 0.10 ppm — to 272

Continued...../4.

E.L. 7/74 "MOINA"

Mineral analysis data from bore core samples

Bore No.	Depth From	Depth To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au ppb	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 2	241.50	243.50		40	405	1025		35		390		9.6								
2	243.50	247.50		25	1240	1075		45		110		10.5								
2	247.50	248.00		25	1100	1430		100	20	155		10.3	1.5							
2	248.00	250.00		35	350	550		40		135		10.8								
2	250.00	252.00		30	560	625		55		110		10.3								
2	252.00	252.50		35	235	560		30	80	140		12.9	1.4							
2	252.50	254.50		40	310	460		20		155		8.7								
2	254.50	256.50		40	465	900		25		90		8.6								
2	256.50	257.00		40	250	320		20	15	115		7.4	1.5							
2	257.00	259.00		30	315	300		35		120		6.3								
2	259.00	261.00		40	405	340		60		120		8.4								
2	261.00	261.50		35	800	390		20	40	135	280	7.0	1.9	108						
2	261.50	263.50		35	250	650		45		110		10.7								
2	263.50	264.50		35	655	375		55		90		9.5								
2	264.50	265.00		35	500	470		130	205	135		7.1	1.6							
2	265.00	267.00		25	530	1260		100		120		11.2								
2	267.00	269.50		35	465	650		50		90		10.8								
2	269.50	270.00		40	725	555		150	220	80		9.1	1.3							
2	270.00	272.00		30	720	400		175		70		13.4								
2	272.00	274.00		35	800	85		200		110		3.4								
2	274.00	274.50		35	285	30		20	15	55	45	13.0	1.4	48						
2	274.50	276.50		35	430	390		100		75		4.6								
2	276.50	278.00		35	325	75		85		60		2.7								
2	278.50	281.50		35	250	40		45		55		2.0								

Bulked pulp — 450 ppm — to 142

Bulked pulp — 1350 ppm — to 142

Bulked pulp — 270 ppm — to 142

Bulked pulp — 0.1 ppm — to 142

Continued...../5.

Mineral analysis data from bore core samples

Bore No.	Depth From	Depth To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au ppb	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 2	281.50	283.50		30		80		10		80		0.7								
2	283.50	284.00		45	240	65		15	15	65		1.5	1.4							
2	283.50	286.00		50	225	1050		25		80		2.8								
2	286.00	287.00		40	415	425		40		65		5.3								
2	287.00	287.50		55	3720	1425		215	115	90		1.12	1.3							
2	287.50	289.50		55	680	1700		40		60		3.9								
2	289.50	292.00		25	445	115		65		80		4.3								
2	292.00	292.50		50	145	1190	1000	15	25	135	20*	4.38	(-0.1) 1.1	-3 ppm	-30	-100	50	30	100	
2	292.50	294.00		30	305	660		25		70		1.7								
2	294.00	296.00		20	460	50		35		70		2.9								
2	296.00	296.50		35	200	210		40	20	135		1.32	1.3							
2	296.50	298.50		25	290	55		20		80		4.0								
2	298.50	301.50		25	600	95		15		70		4.0								
2	301.50	302.00		35	320	40		25	-10	75		2.8	1.4							
2	302.00	304.00		35	75	540		10		55		3.2								
2	304.00	306.00		30	175	960		30		60		3.0								
2	306.00	306.50		30	440	35		45	-10	115		4.6	1.6							
2	306.50	308.50		40	310	50		55		120		3.1								
2	308.50	311.00		30	110	30		15		70	30*	2.0								
2	311.00	311.50		50	1260	25		45	-10	125	30*	0.59	(-0.1) 1.4	-3 ppm	-30	-100	30	-5	80	
2	311.50	313.50		35	295	30	200	15		70		1.5								
2	313.50	316.00		40	70	25		15		70		0.7								
2	316.00	316.50		55	35	25		15	-10	60		0.59	1.4							
2	316.50	318.50		20	1745	30		100		75		2.2								

Pb: ← to 335 — Bulked pulp — 580 ppm — to 272 →

Sn: ← to 335 — Bulked pulp — 740 ppm — to 272 →

Be: ← to 335 — Bulked pulp — 80 ppm — to 272 →

Au: ← to 335 — Bulked pulp 0.02 ppm — to 272 →

Continued...../6.

E.L. 7/74 "MOINA" -6- Mineral analysis data from bore core samples

Bore No.	Depth From	Feet To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au ppb	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 2	318.50	320.50		20	265	45		15		120		1.9								
2	320.50	322.00		15	475	65		40		60		3.3								
2	322.00	324.00	Missing Core																	
2	324.00	324.50		15	220	20		25		30		22.78	0.5							
2	324.00	327.00	Missing Core																	
2	327.00	330.00		10	415	25		90		70		2.4								
2	330.00	333.00		-10	455	20		230		25		1.1								
2	330.00	330.50		15	40	10		180	-10	15		1.45	-10							
2	333.00	333.50		10	80	20	100	200	20	20	10*	0.50	(0.1) 0.5	-3 ppm	-30	-100	20	5	150	
2	333.50	335.00	Missing Core																	
2	335.00	335.50		15	60	-10		80	-10	10		0.39	-10							
2	338.00	338.50		20	110	30		20	35	45		1.00	0.9							
2	338.50	342.00		25	20	15		165		45		.10								
2	342.00	342.50		20	60	10		1615	-10	15		1.66	0.5							
2	342.00	345.00		70	70	370		195		170		15.40								
2	345.50	348.00		25	140	410		190		240		2.4								
2	346.50	347.00		-10	25	30	100	45	25	20	5*	0.15	(0.1) 0.5	-3 ppm	-30	-100	10	-5	80	
2	351.00	351.50		-10	15	-10		-10	-10	10		0.82	-10							
2	356.00	356.50		-10	70	20		225	20	25		1.72	0.7							
2	360.00	360.50		10	40	20		35	50	25		0.64	0.7							
2	364.00	364.50		20	20	30	100	15	20	60	10*	5.00	(0.1) 0.9	-3 ppm	-30	-100	20	-5	80	
2	368.00	368.50		-10	1260	-10		655	-10	16		1.10	-10							
2	371.00	371.50		10	55	-10		15	15	20		0.22	-10							

W ppm: 272 → to 272 — Bulked pulp

Sn ppm: 272 → to 272 — Bulked pulp

Be ppm: 272 → to 272 — Bulked pulp

Au ppb: 272 → to 272 — Bulked pulp

Continued...../7.

Bore No.	Depth From	Depth To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au ppb	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 2	375.00	375.50		20	10	-10		20	-10	20		0.11	-10							
2	378.00	378.50		-10	40	-10	100	215	10	30	50*	1.50	(0.1) 0.5	-3 ppm	-30	-100	30	-5	100	
2	382.50	383.00		10	-10	10		30	15	40		1.50	0.7							
2	387.50	388.00		-10	-10	10		-10	-10	20		00.33	-10							
2	392.00	392.50		10	40	15		30	25	25		0.34	-10							
2	396.00	396.50		15	25	10	10	15	40	10	1*	0.05	(0.1) -10	-3 ppm	-30	-100	20	5	80	
2	1000.00	1000.50		25	-10	-10		-10	40	-10		0.03	-10							
2	1007.00	1007.50		-10	-10	-10		-10	-10	-10		0.03	-10							
2	1014.00	1014.50		-10	-10	-10		-10	10	-10		0.02	-10							
2	1021.00	1021.50		15	-10	-10	1	-10	-10	-10	-1*	0.03	(0.1) -10	-3 ppm	-30	-100	10	-5	80	
2	1028.00	1028.50		15	-10	-10		-10	-10	-10		0.08	-10							
2	1036.00	1073.50		10	20	-10		-10	-10	-10		0.06	-10							
2	1040.00	1040.50		15	25	-10		-10	-10	-10		0.03	-10							
2	1048.00	1048.50		-10	-10	-10	100	-10	-10	-10	-1*	0.03	(0.1) -10	-3 ppm	-30	-100	10	-5	80	
2	1055.00	1055.50		-10	-10	10		-10	-10	-10		0.03	-10							
2	1063.00	1063.50		-10	130	-10		15	10	10		0.05	-10							
2	1073.00	1073.50		-10	-10	-10		10	-10	-10		0.03	-10							
2	1077.00	1077.50		10	-10	-10	1	15	10	-10	-1*	0.03	(0.1) -10	-3 ppm	-30	-100	10	-5	80	
2	1085.50	1086.00		10	-10	-10		-10	-10	15		0.02	-10							
2	1099.50	1100.00		10	10	-10		-10	15	-10		0.62	-10							
DETECTION LIMIT				10	5	10		10	10	5			0.3		30	100	10	5	5	
METHOD OF ANALYSIS				COM-ALCO AAS	COMALCO COLORI-METRIC	COM-ALCO AAS	AMDEL XRF SCHEME B1	COMALCO AAS			AMDEL SPECTRO	COMALCO FUSION SIE	COM-ALCO AAS	AMDEL FIRE ASSAY	AMDEL SPECTROGRAPHIC SCHEME A1,A2.					

Notes: (1) Analyses in brackets ( ) are semi-quantatitive spectrographic by Amdel.

(2) Be analyses of bulk samples are by an Amdel chemical method.
Values of Be marked * may be too low by a factor of about 10

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L.7/74 MOINA - SHEPHERD & MURPHY MINE AREA

Mineral analysis data from bore core samples

Revised PWA May 78

Bore No.	Depth From	Feet To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au ppb	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 3A	248.00	248.50		30	-10	(-10) -10	1	-10	75	110	1*		(0.1) 1.0	-3 ppm	-30	-100	100	10	80	
3A	257.50	258.00		40	70	10		-10	110	210		0.09	0.7							
3A	262.50	263.00		25	55	-10		-10	175	120		0.04	-0.5							
3A	270.50	271.00		15	-10	-10		-10	145	85		0.05	-10							
3A	275.00	275.50		20	-10	(-10) -10	3	-10	25	25	1*	0.04	(0.1) -10	-3 ppm	-30	-100	10	-5	50	
3A	285.00	285.50		25	-10	-10		40	85	-10		0.04	-10							
3A	289.00	289.50		20	-10	-10		-10	50	25		0.03	0.9							
3A	295.50	296.00		20	-10	-10		-10	20	25		0.12	-10							
3A	301.00	301.50		25	-10	(-10) -10	3	-10	-10	10	1*	0.05	(0.1) -10	-3 ppm	-30	-100	80	-5	150	
3A	313.00	313.50		15	-10	-10		-10	80	-10		0.04	-10							
3A	317.00	317.50		15	-10	-10		-10	25	15		0.05	-10							
3A	324.00	324.50		20	-10	-10		-10	30	20		0.11	-10							
3A	334.00	334.50		20	30	15	10	30	55	25	1*	0.09	(0.3) -10	-3 ppm	-30	-100	30	-5	80	
3A	338.00	338.50		40	-10	-10		40	30	45		0.11	-10							
3A	342.00	342.50		20	60	10		1615	-10	15		1.66	-0.5							
3A	346.00	346.50		50	165	575		85	110	250		6.80	1.4							
3A	348.00	350.00		30	105	420		190		640		10.00								
3A	350.00	350.50		30	85	380		25	15	210		0.04	1.6							
3A	350.50	352.00		45	135	1950		85		175		6.40								
3A	352.00	355.00		30	7000	1090		175		140		12.00								
3A	355.00	355.50		35	360	1025	550	75	910	145	310	11.50	2.3	84						
3A	355.50	357.50		20	850	445		115		140		8.70								

Bulked sample to 342

Bulked sample to 342

Bulked sample 342

Bulked sample to 342

Continued...../2.

E.L. 7/74 "MOINA" -2- Mineral analysis data from bore core samples

Bore No.	Depth From	Depth To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au ppb	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 3A	357.50	359.00		25	280	600		130		115		11.40								
3A	359.00	359.50		30	105	230	1200	60	30	140		12.00	2.3							
3A	360.00	362.00		30	70	1290		95		110		10.50								
3A	362.00	364.00		30	485	640		85		120		11.10								
3A	364.00	364.50		30	75	230	2000	180	30	150		11.90	2.3							
3A	364.50	366.50		20	95	370		75		115		12.20								
3A	366.50	369.50		20	120	200		180		140		11.90								
3A	369.50	370.00		20	40	470	2900	155	15	165		12.00	2.2							
3A	370.00	372.00		20	360	165		500		135		12.80								
3A	372.00	373.00		15	635	340		550		135		6.00								
3A	373.00	373.50		30	330	315	3200	490	195	185	370	11.30	1.6							
3A	373.50	375.50		-10	190	55		1020		95		2.50								
3A	375.50	377.00		25	105	660		50		135		12.70								
3A	377.00	377.50		30	185	55	8100	2075	35	200		2.90	1.4							
3A	377.50	380.00		10	130	50		1100		160		2.20								
3A	380.00	381.00		20	105	350		700		190		1.30								
3A	381.00	381.50		35	140	560	2800	380	265	255		1.32	1.6							
3A	381.50	383.00		20	150	400		450		190		2.00								
3A	383.00	385.00		20	120	680		320		205		1.80								
3A	385.00	385.50		35	70	310	5800	90	280	225		1.60	1.4							
3A	385.50	388.00		15	25	450		75		175		2.40								
3A	388.00	390.00		20	130	445		110		225		3.40								
3A	390.00	390.50		20	310	30	6200	500	30	215	35	2.50	1.4	16						

Bulked sample XRF Andel 1300 ppm 377.5 to 403.5

Bulked sample Andel XRF 2750 ppm 377.5 to 403.5

Bulked sample Chemical Andel 280 ppm 377.5 to 403.5

Bulked sample fire assay K4/2 Andel 0.015 ppm 377.5 to 403.5

Bulked sample 48

E.L. 7/74 "MOINA" -3- Mineral analysis data from bore core samples

Bore No.	Depth From	Depth To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au ppb	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 3A	390.50	392.00		25	165	400		75		190		2.30								
3A	392.00	393.00		25	35	145		80		160		1.60								
3A	393.00	393.50		20	110	70	3000	310	10	140		3.40	1.1							
3A	393.50	395.50		25	100	320		300		150		4.00								
3A	395.50	398.00		15	115	120		245		110		3.10								
3A	398.00	398.50		15	45	45	4100	380	-10	165		1.80	0.7							
3A	398.50	401.50		25	100	220		90		120		2.00								
3A	401.50	403.00		15	110	290		2100		110		1.00								
3A	403.00	403.50		30	190	230	8200	1730	60	155		0.88	0.5							
3A	403.50	406.00		20	290	150		130		90		12.20								
3A	406.00	407.00		20	240	290		60		85		10.20								
3A	407.00	407.50		25	1400	620	2600	165	30	100	340	11.90	1.8	36						
3A	407.50	410.00		20	380	490		160		125		9.80								
3A	410.00	412.00		25	610	1290		160		85		10.80								
3A	412.00	412.50		25	760	150	4000	200	125	70		11.30	2.3							
3A	412.50	414.00		15	290	155		70		70		9.10								
3A	414.00	417.00		20	475	470		100		80		9.80								
3A	417.00	417.50		30	320	200	2200	20	10	170		5.60	2.0							
3A	417.50	420.00		15	3000	520		85		125		8.00								
3A	420.00	420.50		30	320	145	2900	35	-10	155		7.20	2.2							
3A	420.50	422.50		15	390	240		160		90		10.20								
3A	422.50	424.00		15	745	210		340		110		8.50								
3A	424.00	424.50		20	1260	325	4400	235	-10	120	37	7.00	1.8	28						

Bulked sample XRF Amdel 1400 ppm to 478

Bulked sample XRF Amdel 403.5 to 377.5

Bulked sample XRF AMDEL 2500 ppm to 478

Bulked sample XRF Amdel 4200 ppm 403.5 to 377.5

Bulked sample Chemical Amdel 320 ppm to 478

Bulked sample 200 ppm 403.5 to 377.5

Bulked sample, fire assay K4/2 Amdel, 0.09 ppm to 478

Bulked sample 0.02 ppm 403.5 to 377.5

Continued...../4.

Bore No.	Depth Feet From	Depth Feet To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F	Ag ppm	Au ppb	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 3A	424.50	426.00		15	600	450		90		100		8.80								
3A	426.00	428.00		20	550	490		120		105		9.00								
3A	428.00	428.50		35	785	430	1800	30	10	130		9.80	1.4							
3A	428.50	430.50		20	630	810		120		135		10.00								
3A	430.50	432.50		25	510	1290		130		120		10.00								
3A	432.50	434.00		25	1290	1530		290		115		9.80								
3A	434.00	436.00		20	330	110		125		120		10.80								
3A	436.00	436.50		25	980	775	1700	75	35	140		6.60	0.8							
3A	436.50	438.00		22	800	460		110		115		11.40								
3A	438.00	439.00		25	360	190		25		90		12.20								
3A	439.00	439.50		20	260	80	3000	435	10	125	130	10.40	0.8	16						
3A	439.50	441.50		20	385	330		20		90		14.20								
3A	441.50	443.00		20	600	630		30		150		11.50								
3A	443.00	443.50		30	345	375	2700	110	15	150		11.50	0.8							
3A	443.50	445.50		15	285	270		70		105		11.60								
3A	445.50	447.50		20	450	150		55		85		11.10								
3A	447.50	448.00		35	430	430	1700	125	140	100		13.20	0.9							
3A	448.00	450.00		20	405	330		360		160		10.60								
3A	450.00	451.00		20	510	480		75		250		12.80								
3A	451.00	451.50		30	1250	410	4900	860	10	290		11.90	0.8							
3A	451.50	453.50		20	685	460		70		225		13.20								
3A	453.50	455.50		15	800	410		85		300		12.30								
3A	455.50	456.00		40	225	310	3200	215	10	435	560	10.70	0.9	12						

Bulked sample to 403.5 — 478

Bulked sample to 403.5 — 478

Bulked sample to 403.5 — 478

Bulked sample to 403.5 — 478

Continued...../5.

E.L. 7/74 "MOINA" -5- Mineral analysis data from bore core samples

Bore No.	Depth From	Depth To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au ppb	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 3A	456.00	458.00		10	315	1180		25		405		12.20								
3A	458.00	460.00		10	1815	615		210		280		10.00								
3A	460.00	460.50		30	1065	490	1300	215	10	250		10.10	1.0							
3A	460.50	463.00		10	735	622		55		220		12.90								
3A	463.00	465.00		-10	1490	295		310		130		11.50								
3A	465.00	465.50		25	445	275	1300	160	10	180		13.50	1.0							
3A	465.50	467.50		-10	1890	195		60		150		11.40								
3A	467.50	469.00		-10	675	55		60		140		12.40								
3A	469.00	469.50		20	690	40	450	130	-10	130		10.90	1.0							
3A	469.50	471.50		10	480	240		75		210		8.30								
3A	471.50	473.50		10	720	440		65		170		11.30								
3A	473.50	474.00		35	230	1060	1800	130	-10	160	250	15.00	0.5	40						
3A	474.00	476.00		15	830	460		70		220		11.00								
3A	476.00	478.00		-10	1160	160		55		200		10.50								
3A	477.00	477.50		35	105	140		25	-10	230		1.52	1.1							
3A	478.00	480.00		15	860	60		140		60		2.30								
3A	480.00	481.50		15	910	50		420		60		2.50								
3A	481.50	482.00		25	855	75		725	-10	75		4.59	0.9							
3A	482.00	484.00		15	80	35		35		65		1.80								
3A	484.00	485.00		20	25	65		30		55		1.30								
3A	485.00	485.50		20	1360	65		630	-10	60		1.60	0.7							
3A	485.50	487.50		35	95	125		30		55		1.00								
3A	487.50	490.00		40	-10	95		-10		10		0.51								

Bulked sample to 403.5 to 445

Bulked sample to 403.5 to 445

Bulked sample to 403.5 to 445

Bulked sample to 403.5 to 445

Bulked sample to 478 to 645

Bulked sample to 478 to 645

Bulked sample to 478 to 645

Bulked sample to 478 to 645

Continued...../6.

221386

E.L. 7/74 "MOINA" -6- Mineral analysis data from bore core samples

Bore No.	Depth From	Depth To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 3A	490.00	490.50		40	-10	(3)40	500	25	-10	70	10*	1.00	(-0.1) 1.0	-3ppm	-30	-100	10	-5	20	
3A	490.50	492.00		20	25	25		-10		65		0.83								
3A	492.50	497.00		20	170	45		300		70		3.5								
3A	495.00	495.50		40	-10	40		15	-10	55		1.21	1.4							
3A	495.50	497.00		20	65	25		10		55		0.77								
3A	497.00	499.00		20	20	25		10		40		0.46								
3A	499.00	499.50		40	3040	30		200	-10	50		1.00	0.9							
3A	499.50	502.00		20	10	25		-10		40		0.31								
3A	502.00	504.00		15	15	25		-10		40		0.32								
3A	504.00	504.50		40	45	35		15	-10	55		0.58	1.5							
3A	504.50	506.50		20	20	25		-10		40		0.34								
3A	506.50	509.00		15	15	25		-10		35		0.17								
3A	509.00	509.50		20	-10	(-1) 30	500	10	-10	25	1*	0.23	(-0.1) 1.0	-3ppm	-30	-100	20	-5	20	
3A	509.50	511.50		15	-10	25		-10		55		0.19								
3A	510.00	510.50		50	-10	35		-10	-10	40		0.17	1.4							
3A	510.50	512.00		25	55	30		-10		65		0.36								
3A	511.50	514.00		25	70	25		-10		50		0.19								
3A	512.50	514.00		25	485	25		60		70		0.78								
3A	514.50	517.50		20	255	25		30		45		0.37								
3A	515.00	515.50		30	-10	40		15	-10	55		1.00	1.4							
3A	517.50	519.00		25	205	25		110		55		0.74								
3A	519.50	521.50		25	55	25		10		50		1.30								
3A	520.00	520.50		40	50	45		20	-10	50		1.40	1.6							

Bulked sample XRF Amdel 190 ppm to 478 to 645

Bulked sample XRF Amdel 330 ppm to 478 to 645

Bulked sample Chemical Amdel 60 ppm to 478 to 645

Bulked sample Amdel fire assay K4/2 0.12 ppm to 478 to 645

Continued...../7.

Bore No.	Depth From	Depth To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 3A	521.50	523.00		20	25	25		20		65		3.30								
3A	522.00	522.50		45	-10	(-1) 35	200	10	-10	50	50*	0.60	(0.1) 1.0	-3 ppm	-30	-100	20	-5	10	
3A	522.50	524.50		20	405	15		-10		55		0.87								
3A	523.50	525.50		20	15	25		-10		50		0.92								
3A	525.50	528.00		25	40	30		10		60		0.55								
3A	527.00	527.50		35	-10	35		10	-10	50		0.29	1.1							
3A	527.50	529.50		30	175	55		350		65		2.9								
3A	529.50	531.00		35	75	35		15		50		1.5								
3A	531.00	531.50		30	175	40		185	-10	85		9.30	1.4							
3A	531.50	533.50		45	35	35		15		55		2.1								
3A	533.50	535.00		30	75	30		15		55		1.0								
3A	535.00	535.50		50	-10	45		15	-10	55		1.80	1.3							
3A	535.50	537.50		25	500	45		340		45		3.40								
3A	537.00	540.00		30	-10	50		60		45		4.00								
3A	540.00	540.50		55	-10	70	100	-10	-10	65	20*	4.40	(-0.1) 1.8	-3 ppm	-30	-100	10	-5	10	
3A	540.50	542.00		25	-10	45		10		40		3.30								
3A	542.50	545.00		20	-10	50		-10		40		3.60								
3A	545.00	545.50		50	-10	60		10	-10	50		2.25	1.6							
3A	545.50	547.00		25	-10	40		-10		50		2.00								
3A	547.50	550.00		25	-10	30		-10		45		2.30								
3A	550.00	550.50		45	-10	50		15	-10	50		2.40	1.7							
3A	550.50	552.50		30	-10	35		-10		55		2.50								
3A	552.50	558.00		25	25	35		50		55		3.20								

Bulked sample XRF Amdel 190 ppm to 478 to 645

Bulked sample SRF Amdel- 330 ppm to 478 to 645

Bulked sample, Chemical, Amdel 60 ppm to 478 to 645

Bulked sample Amdel fire assay K4/2 0.12 ppm to 478 to 645

Continued...../8.

E.L. 7/74 "MOINA" -8- Mineral analysis data from bore core samples

Bore No.	Depth Feet From	To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 3A	558.00	558.50		40	-10	40		-10	-10	45		1.49	1.0							
3A	558.50	560.50		25	-10	30		10		45		1.70								
3A	560.50	562.00		25	580	35		490		50		4.90								
3A	562.00	562.50		40	325	40 (1)	50	320	-10	85	20*	8.20	(-0.1) 1.5	-3 ppm	-30	-100	10	-5	10	
3A	562.50	564.50		25	3280	60		235		70		6.10								
3A	564.50	568.00		25	395	30		15		45		2.00								
3A	568.00	568.50		40	-10	35		-10	-10	40		1.00	0.8							
3A	568.50	570.00		25	270	30		175		45		2.20								
3A	570.50	572.50		25	20	35		10		55		1.80								
3A	572.50	573.00		60	80	35		10	10	50		1.00	0.9							
3A	573.00	575.00		25	-10	35		10		45		1.70								
3A	575.00	578.50		25	-10	35		10		50		1.50								
3A	578.00	578.50		50	25	60		-10	-10	65		4.40	1.3							
3A	579.00	581.00		20	105	70		610		55		4.40								
3A	581.00	582.00		25	-10	30		30		45		1.10								
3A	582.00	582.50		50	2880	35 (1)		570	-10	65	20*	4.92	(-0.1) 2.4	-3 ppm	-30	-100	20	-5	10	
3A	582.50	585.00		25	-10	30	200	10		40		1.60								
3A	585.00	587.00		25	135	40		15		75		5.80								
3A	587.00	587.50		40	50	30		10	-10	55		0.83	2.0							
3A	587.50	589.50		25	15	25		15		50		1.10								
3A	589.50	592.00		25	-10	25		10		55		1.10								
3A	592.00	592.50		40	210	35		-10	-10	65		5.00	2.4							
3A	592.50	594.00		20	50	45		15		70		4.70								

Bulked sample to 478 / to 645

Continued...../9.

E.L. 7/74 "MOINA" -9- Mineral analysis data from bore core samples

Bore No.	Depth From	Depth To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 3A	594.00	594.50		50	110	30		-10	-10	65		2.80	2.2							
3A	594.50	596.50		25	105	40		10		55		1.80								
3A	596.50	598.00		10	-10	25		10		155		6.10								
3A	598.00	598.50		25	-10	20	2000	-10	-10	30	50*	1.12	(-0.1) -0.5	-3 ppm	-30	-100	10	-5	5	
3A	598.50	600.50		-10	-10	15		15		70		1.50								
3A	600.50	603.00		15	-10	20		95		45		2.70								
3A	603.00	603.50		20	25	20		50	-10	30		2.60	0.7							
3A	603.50	605.50		15	40	130		100		125		7.30								
3A	605.50	607.00		10	-10	25		30		85		3.70								
3A	607.00	607.50		30	45	35		10	-10	65		1.42	0.6							
3A	607.50	610.00		15	10	20		15		60		1.40								
3A	612.00	614.00		10	-10	15		15		35		1.40								
3A	614.00	614.50		25	-10	15		75	10	15		1.40	-10							
3A	614.50	619.00		15	-10	20		10		45		2.60								
3A	618.50	619.00		40	-10	(-1) 20	1	25	10	60	1*	0.13	(-0.1) -0.5	-3 ppm	-30	-100	30	-5	200	
3A	619.00	621.00		15	-10	15		-10		75		1.00								
3A	621.00	613.00		10	30	15		25		50		2.50								
3A	623.00	625.00	(A)	-10	-10	15		-10		40		1.00								
3A	623.00	625.00	(B)	10	10	25		15		60		1.50								
3A	625.00	627.50		10	45	20		140		70		4.10								
3A	627.50	630.00		10	-10	15		15		45		1.70								
3A	630.00	633.00		-10	-10	15		15		30		1.40								
3A	633.00	635.00		-10	190	15		1275		30		1.90								

Bulked sample to 478 — 645

Bulked sample to 478 — 645

Bulked sample to 478 — 645

Bulked sample to 478 — 645

E.L. 7/74 "MOINA" -10- Mineral analysis data from bore core samples

Bore No.	Depth Feet From	Depth Feet To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	F %	Ag ppm	Au	Sb ppm	Ta ppm	V ppm	Co ppm	Ni ppm	Remarks
ML 3A	635.00	637.00		10	-10	20		15		35		0.64								
3A	637.00	639.00		-10	-10	20		15		90		0.92								
3A	639.00	641.50		-10	-10	20		15		60		1.40								
3A	641.50	643.50		10	205	30		15		55		0.80								
3A	643.50	645.00		15	-10	25		10		55		0.46								
3A	690.00	690.50		20	-10	20		20	45	130		0.22	-10							
3A	700.50	701.00		10	-10	10		15	-10	450		0.19	-10							
3A	711.50	712.00		10	-10	10		-10	-10	20		0.92	-10							
3A	723.00	723.50		50	-10	15(-1)	3	30	45	105	1*	0.18	(0.1) 0.6	-3 ppm	-30	-100	20	10	150	
3A	740.00	740.50		35	35	15		25	30	90		0.05	-10							
3A	749.00	749.50		20	700	15		15	65	65		0.14	-0.3							
3A	777.50	778.00		20	15	30 (5)	100	20	75	125	5*	0.44	(0.1) 0.7		-30	-100	30	50	100	
DETECTION LIMIT				10	5	10		10	10	5			0.3		30	100	10	5	5	
METHOD OF ANALYSIS				COM-ALCO AAS	COMALCO COLORI-METRIC	COM-ALCO AAS	AMDEL XRF SCHEME B1	COMALCO AAS			AMDEL SPECTRO	COMALCO FUSION SIE	COM-ALCO AAS	AMDEL FIRE ASSAY	AMDEL SPECTROGRAPHIC SCHEME A1, A2.					

Bulked sample to 478 (Pb–Sn); Bulked sample to 478 (Be–Au)

NOTES: (1) Analyses in brackets ( ) are semi-quantatitive spectrographic by Amdel

(2) Be analyses of bulk samples are by an Amdel chemical method. Values of Be marked * may be too low by a factor of about 10.

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L. 7/74 "MOINA", SHEPHERD & MURPHY MINE AREA

Mineral analysis data from bore core samples

Bore No.	Depth Metres From	To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	AAS Au ppm	Au Fire Assay ppm	Li ppm	Remarks
SMD 4	68.37	69.00		6	540	580	1150	10	130	1240	250	15.6	-1	0.10			
4	69.00	70.00		10	820	620	1200	22	150	180	700	21.0	-1	0.10	.095	75	
4	70.00	70.70		18	270	340	2200	-4	10	90	300	9.6	-1	0.10	.075		
4	70.70	71.30		14	70	190	1900	6	5	90	300	8.5	-1	-0.05			
4	71.30	72.00		8	300	330	1600	12	50	180	250	12.5	-1	0.10			
4	72.00	73.00		4	640	620	1700	38	200	140	300	21.0	-1	0.15	.070		
4	73.00	74.00		8	1000	780	1550	30	160	130	600	26.0	-1	0.15	.15		
4	74.00	75.00		16	1750	600	1800	100	240	130	300	26.5	-1	0.10			
4	75.00	76.00		-4	1750	350	1700	95	150	120	300	24.0	-1	0.05			
4	76.00	77.00		12	520	540	1650	24	130	100	300	20.0	-1	0.10			
4	77.00	78.00		8	620	450	1250	55	120	120	300	16.5	-1	0.10			
4	78.00	79.00		10	1450	580	1400	75	150	90	500	20.0	-1	0.05			
4	79.00	80.00		10	760	300	2100	65	32	80	500	17.5	-1	-0.05			
4	80.00	80.63		22	840	470	1400	90	250	80	500	20.0	-1	0.05			
4	80.63	81.66		12	600	6	220	36	10	32	200	4.1	-1	-0.05		35	
4	81.66	83.70		20	460	320	1000	26	25	50	300	7.4	-1	-0.05		20	
4	83.70	85.00		10	1600	270	1300	60	350	70	250	16.5	-1	-0.05			
4	85.00	86.00		-4	1500	320	1500	75	220	65	200	15.5	-1	0.05			
4	86.00	87.00		14	2200	410	1250	75	240	70	250	14.0	-1	0.05			
4	87.00	88.00		14	860	660	1100	32	350	190	150	12.5	-1	0.05			
4	88.00	89.00		10	700	350	980	55	210	310	150	9.9	-1	0.05			
4	89.00	90.60		10	390	230	1150	42	160	95	100	7.5	-1	0.05		110	
4	90.60	91.25		18	180	60	1450	8	48	100	500	4.4	-1	0.05			
4	91.25	92.45		24	300	75	1150	150	15	110	100	5.8	-1	0.05			
4	92.45	93.80		18	370	80	980	-4	60	110	100	5.1	-1	-0.05			
4	93.80	94.40		14	140	14	1050	6	65	60	150	4.5	-1	-0.05			
4	94.40	95.90		12	1300	200	1200	26	80	70	200	8.1	-1	0.05		60	
4	95.90	97.15		6	400	6	280	10	45	42	150	4.9	-1	-0.05			
4	97.15	98.40		10	270	-4	75	20	40	40	70	1.9	-1	-0.05		30	

cont'd 2

E.L. 7/74 "Moina"

Mineral analysis data from bore core samples

Bore No.	Depth Metres From	To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	AAS Au ppm	Au Fire Assay ppm	Remarks
SMD 4	98.40	104.00		6	740	4	310	90	18	18	100	1.6	-1	-0.05		
4	104.00	106.50		4	390	-4	30	26	18	45	70	1.8	-1	0.05		
4	106.50	107.50		18	85	-4	60	18	20	35	70	1.3	-1	-0.05		
DETECTION LIMIT				(4)	(10)	(4)	(4)	(4)			(1)		(1)	(0.05)	(.005)	
METHOD OF ANALYSIS - All by AMDEL				X-RAY FLUORESCENCE SCHEME B1					CODE C1				CODE C2	AAS CODE C3	Fire Assay Code K4/2	

Mineral analysis data from bore core samples

Bore No.	Depth From	Depth To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	AAS Au ppm	Au Fire Assay ppm	Remarks
SMD 5	38.10	39.80		34	-10	16	80	-4	28	170	30	0.45				
5	39.80	40.67		6	270	380	660	4	200	1740	250	10.5	-1	0.10	0.075	
5	40.67	42.00		12	490	580	1350	10	75	95	250	17.0				
5	42.00	43.00		14	740	700	860	16	210	100	300	16.5				
5	43.00	44.00		10	1750	320	1400	14	320	70	200	18.0	-1	0.05		
5	44.00	45.00		12	680	620	1050	34	170	65	200	17.0				
5	45.00	46.60		24	900	1100	1500	44	70	70	200	14.0				
5	46.60	47.50		10	170	44	250	-4	22	70	70	4.5				
5	47.50	48.85		6	480	28	500	-4	12	90	100	4.7	-1	0.05		
5	48.85	50.00		14	490	80	1100	-4	10	85	300	15.0				
5	50.00	51.30		65	540	820	1250	16	200	110	250	13.5				
5	51.30	52.80		20	780	760	940	8	150	200	250	12.5				
5	52.80	53.30		12	430	65	760		430	50	250	12.5	-1	-0.05	0.035	
5	53.30	54.50		18	380	140	720	16	75	60	250	9.8				
5	54.50	55.90		20	620	160	880	22	22	65	200	11.0				
5	55.90	56.35		12	760	10	320	22	20	30	100	4.5				
5	56.35	57.45		44	110	700	540	36	15	40	100	2.5				
5	57.45	57.90		12	3350	34	740	18	85	150	250	9.9				
5	57.90	58.70		8	780	180	820	38	140	90	300	10.5				
5	58.70	60.00		12	700	270	1250	34	60	60	200	15.8				
5	60.00	60.85		10	560	220	1250	14	190	75	250	14.5				
5	60.85	62.00		6	1050	8	3800	120	42	48	100	11.0				
5	62.00	72.90		6	470	6	390	30	5	48	100	5.3				
5	72.90	74.40		8	170	24	75	4			100	2.4				
DETECTION LIMIT				(4)	(10)	(4)	(4)	(4)			(1)		(1)	(0.05)	(.005)	
METHOD OF ANALYSIS - All by AMDEL				X-RAY FLUORESCENCE SCHEME B1					CODE C1				CODE C2	AAS CODE C3	Fire Assay Code K4 '2	

Bore No.	Depth Metres From	To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	Au ppm	Remarks
SMD 6	21.35	22.75		8	130	360	1600	-4	150	270	250	18.0			
6	22.75	23.50		14	580	640	1850	-4	170	310	200	19.0			
6	23.50	24.75		6	450	310	2000	-4	35	120	250	17.5			
6	24.75	26.15		12	640	400	1950	-4	95	100	250	19.4			
6	26.15	28.00		14	540	290	1050	-4	15	90	150	8.4			
6	28.00	29.15		10	720	440	2050	-4	28	130	150	16.0			
6	29.15	29.50		10	720	360	1250	6	38	130	150	28.5			
6	29.50	30.80		12	410	560	1950	-4	50	120	250	19.5			
6	30.80	32.15		14	230	290	760	6	12	50	300	8.6			
6	32.15	33.90		14	1050	880	1950	8	190	120	250	21.5			
6	33.90	34.50		16	380	390	1750	-4	10	42	200	9.2			
6	34.50	35.50		18	660	660	2100	-4	15	100	250	20.0			
6	35.50	36.40		12	700	880	3350	10	35	110	250	16.0			
6	36.40	38.00		12	480	640	2050	6	32	100	250	15.5			
6	38.00	39.00		20	640	210	2550	12	30	100	500	15.0			
6	39.00	40.00		12	860	900	1700	90	90	90	500	24.5			
6	40.00	41.00		12	1550	400	2700	80	35	60	300	21.5			
6	41.00	42.00		4	2650	680	1150	160	160	100	250	30.0			
6	42.00	43.00		6	4600	600	1300	110	50	120	250	30.0			
6	43.00	44.00		10	860	660	1200	75	48	140	250	24.5			
6	44.00	45.00		10	1100	470	1500	90	18	140	300	24.5			
6	45.00	46.00		12	600	700	1100	16	18	120	300	23.0			
6	46.00	47.00		6	980	700	1400	-4	15	150	200	20.5			
6	47.00	48.00		6	740	600	1100	16	18	160	200	20.5			
6	48.00	49.00		-4	940	600	1150	170	8	160	250	25.0			
6	49.00	50.00		8	1050	680	1050	14	18	160	300	24.5			
6	50.00	51.00		4	740	600	1050	24	32	220	150	15.5			
6	51.00	52.00		6	880	470	900	4	8	170	150	17.0			
6	52.00	53.00		-4	860	620	1000	-4	10	200	200	17.5			
6	53.00	54.00		12	840	440	1250	10	85	200	250	17.5			
6	54.00	55.00		-4	1000	390	1200	26	28	140	250	19.0			
6	55.00	56.00		10	1750	600	1650	34	85	110	200	20.5			
6	56.00	57.00		6	700	330	1700	42	28	110	200	16.0			

Mineral analysis data from bore core samples

Bore No.	Depth Metres From	To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 ppm	Ag ppm	Au ppm	Remarks
SMD 6	57.00	58.00		10	980	540	1500		8	100	200	21.5			
6	58.00	59.00		8	2050	600	1100	46	70	130	300	24.5			
6	59.00	60.00		12	740	620	1300	30	160	180	200	28.5			
6	60.00	61.00		18	2050	480	1300	110	38	140	250	28.5			
6	61.00	62.00		16	1050	450	1550	90	65	140	300	27.5			
6	62.00	63.00		10	780	450	1350	80	32	130	250	21.0			
6	63.00	64.00		16	400	410	1150	350	15	120	300	15.0			
6	64.00	65.00		8	540	600	1600	46	60	130	300	20.0			
6	65.00	66.00		6	620	340	1500	55	35	130	150	16.0			
6	66.00	67.70		22	640	370	1550	38	75	120	200	14.0			
6	67.70	69.00		6	700	12	380	28	8	120	100	7.4			
6	69.00	70.00		6	1400	30	480	150	8	80	400	9.5			
6	70.00	71.00		10	360	48	620	24	5	140	150	6.2			
6	71.00	72.00		10	900	-4	420	75	8	130	200	5.4			
6	72.00	73.00		12	1300	30	430	24	8	100	150	8.6			
6	73.00	74.00		14	400	30	520	8	15	70	150	7.8			
6	74.00	75.00		6	460	140	1300	44	20	75	200	19.0			
6	75.00	76.00		18	840	1650	1500	40	140	1700	200	19.0			
6	76.00	77.00		4	960	110	1650	40	10	140	200	16.5			
6	77.00	78.00		-4	1050	42	1600	30	18	80	200	16.5			
6	78.00	79.00		8	1100	270	1250	65	120	70	200	19.0			
6	79.00	80.00		6	1100	110	1350	44	65	80	200	14.0			
6	80.00	81.00		18	560	320	1350	55	65	4550	200	13.5			
6	81.00	82.00		75	1050	2100	1250	110	100	470	400	10.5			
6	82.00	83.05		70	820	2000	960	75	90	170	150	11.0			
6	83.05	84.00		14	660	110	1000	36	10	130	150	6.5			
6	84.00	85.00		16	920	190	2050	55	45	110	200	9.0			
6	85.00	85.80		18	350	32	240	55	38	80	150	4.9			
6	85.80	87.25		12	660	4	350	75	8	510	200	4.5			
6	87.25	88.00		38	450	36	240	180	8	75	200	4.9			
6	88.00	89.00		48	1100	8	210	32	8	90	500	5.3			
6	89.00	90.00		38	110	18	200	280	5	75	600	3.7			
6	90.00	91.00		30	1450	28	250	240	5	70	600	6.0			

E.L. 7/74 "Moina"

Mineral analysis data from bore core samples

Bore No.	Depth Metres From	To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	Au ppm	Remarks
SMD 6	91.00	91.70		24	390	40	300	230	5	70	500	4.1			
6	91.70	92.50		10	1250	65	180	80	95	90	300	9.0			
6	92.50	101.00		12	1600	8	75	320	8	75	150	5.8			
6	101.00	102.00		12	130	8	46	36	12	45	150	3.1			
6	102.00			12	55	16	5000	310	8	55	70	7.0			
DETECTION LIMIT				(4)	(10)	(4)	(4)	(4)			(1)		(1)	(0.05)	
METHOD OF ANALYSIS				X-RAY FLUORESCENCE SCHEME B1					CODE C1				CODE C2	CODE C3	

E.L. 7/74 "Moina"

Mineral analysis data from bore core samples

Bore No.	Depth Metres From	To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	Au ppm	Remarks
SMD 7	00.00	01.00		16	880	680	2750	22	38	510	100	10.5			
7	01.00	08.00		14	800	420	2350	-4	12	280	100	25.5			
7	08.00	09.62		14	880	640	1400	-4	15	220	100	28.0			
7	09.62	11.00		4	1350	640	1350	16	28	250	100	28.0			
7	11.00	12.00		6	780	420	1650	-4	12	270	250	26.5			
7	12.00	13.00		6	330	390	1900	-4	10	360	200	21.5			
7	13.00	14.00		8	580	440	1550	4	12	370	200	21.5			
7	14.00	16.00		10	840	490	1300		20	370	200	23.5			
7	16.00	17.00		4	290	340	1300	16	10	320	300	17.7			
7	17.00	18.00		6	660	260	1400	6	2	270	200	19.5			
7	18.00	19.00		18	1100	320	1550	12	5	130	300	21.5			
7	19.00	20.00		14	840	540	1600	22	85	370	300	25.0			
7	20.00	21.00		8	800	640	1500	38	10	270	250	28.0			
7	21.00	21.75		6	920	470	2200	28	5	570	300	27.0			
7	21.75	22.65		16	390	130	2400	16	5	120	250	9.9			
7	22.65	24.00		12	1000	580	1800	20	5	170	300	22.5			
7	24.00	25.00		12	1550	420	1800	80	120	9800	200	24.5			
7	25.00	26.00		16	400	940	1300	30	230	740	200	20.5			
7	26.00	27.00		8	680	920	1500	26	440	450	200	21.5			
7	27.00	28.00		8	920	580	2500	60	220	410	250	22.5			
7	28.00	29.00		12	860	460	2250	95	85	310	250	22.0			
7	29.00	30.00		4	840	520	1700	46	110	1870	200	17.0			
7	30.00	31.00		8	1850	350	1400	46	38	180	250	23.0			
7	31.00	32.00		12	500	350	1050	50	110	160	200	23.0			
7	32.00	33.00		18	2150	390	2250	48	200	190	150	19.5			
7	33.00	34.00		-4	1300	240	2300	55	8	150	150	21.5			
7	34.00	35.00		12	720	680	1450	30	32	120	250	23.5			
7	35.00	36.00		14	940	230	2850	46	40	210	150	14.0			
7	36.00	37.00		12	1100	300	1800	70	60	110	200	20.5			
7	37.00	38.00		14	1950	470	880	70	130	110	300	22.0			
7	38.00	39.00		10	4500	500	1050	230	95	110	250	23.0			
7	39.00	40.00		4	3050	620	920	290	50	120	200	22.5			
7	40.00	41.00		8	1400	520	1000	80	55	170	300	21.0			

221398

Bore No.	Depth From	Depth To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	Au ppm	Remarks
SMD 7	41.00	42.00		6	2300	560	840	55	42	120	200	24.5			
7	42.00	43.00		4	540	460	720	24	60	100	200	25.0			
7	43.00	44.00		8	1850	340	540	160	35	120	250	19.0			
7	44.00	45.35		10	660	800	780	26	42	90	250	19.5			
7	45.35	46.15		12	180	410	1050	6	70	55	150	13.5			
7	46.15	47.30		10	370	520	980	12	32	90	150	13.5			
7	47.30	48.55		10	280	640	900	12	75	110	200	15.5			
7	48.55	50.00		20	3400	170	400	55	8	110	200	8.6			
7	50.00	51.00		20	840	42	380	180	8	120	80	5.9			
7	51.00	52.00		30	230	1300	490	14	10	70	200	10.0			
7	52.00	52.75		18	95	2050	420	8	28	85	300	15.0			
7	52.75	54.00		12	290	390	720	-4	18	95	300	16.0			
7	54.00	55.00		18	350	640	720	-4	25	110	300	22.0			
7	55.00	56.00		32	480	180	480	36	8	80	200	4.8			
7	56.00	57.00		28	760	28	420	170	8	75	300	5.3			
7	57.00	58.00		24	110	8	370	42	5	60	150	2.7			
7	58.00	59.00		12	270	12	390	24	8	75	300	3.9			
7	59.00	60.00		18	1200	8	600	140	8	85	300	9.5			
7	60.00	61.00		26	45	6	310	8	8	60	150	5.3			
7	61.00	62.00		26	80	-4	680		8	50	100	3.2			
7	62.00	63.00		26	150	8	740	4	8	70	150	4.4			
7	63.00	64.00		30	160	-4	1150	-4	8	65	100	2.9			
7	64.00	65.00		24	980	4	1000	12	8	65	100	4.0			
7	65.00	65.75		10	200	-4	330	160	5	100	100	7.8			
7	65.75	66.75		14	330	12	660	12	8	95	100	3.4			
7	66.75	67.75		12	480	32	380	42	25	60	100	6.8			
7	67.75	69.00		16	360	10	350	24	5	80	100	3.9			
7	69.00	70.00		-4	420	8	190	42	8	55	100	5.5			
7	70.00	71.50		8	270	-4	180	8	10	55	100	4.5			
DETECTION LIMIT				(4)	(10)	(4)	(4)	(4)			(1)		(1)	(0.05)	
METHOD OF ANALYSIS				X-RAY FLUORESCENCE SCHEME B1					CODE C1				CODE C2	CODE C3	

cont'd 9

E.L. 7/74 "Moina"

Mineral analysis data from bore core samples

Bore No.	Depth Metres From	To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	Au ppm	Remarks
SMD 8	40.00	41.00		30	15	34	40	-4	20	70	50	0.35			
8	41.00	41.60		6	-10	8	95	4	8	220	30	0.25			
8	41.60	42.50		6	15	-4	85	4	20	430	20	0.3			
8	42.50	43.20		-4	20	4	95	-4	5	350	30	0.25			
8	43.20	44.00		50	30	32	140	-4	8	570	50	0.25			
8	44.00	45.95		170	45	140	300	-4	10	380	70	0.65			
8	45.95	46.35		12	140	540	240	-4	30	120	80	15.5			
8	46.35	47.00		16	50	480	390	4	12	120	100	6.2			
8	47.00	48.00		14	100	230	250	-4	40	110	150	4.9			
8	48.00	49.60		14	70	44	36	4	110	48	70	2.2			
8	49.60	50.50		10	15	-4	6	4	28	22	30	0.45			
8	50.50	51.70		-4	10	4	14	4	28	28	30	0.60			
8	51.70	52.50		16	-10	4	38	-4	45	60	40	0.65			
DETECTION LIMIT				(4)	(10)	(4)	(4)	(4)			(1)		(1)	(0.05)	
METHOD OF ANALYSIS				X-RAY FLUORESCENCE SCHEME B1					CODE C1				CODE C2	CODE C3	

Bore No.	Depth Metres From	To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	Au ppm	Remarks
SMD 9	91.50	92.50		10	240	130	880	6	240	250	300	14.5			221400
9	92.50	94.20		12	310	100	1700	12	320	250	250	11.0			
9	94.20	95.00		14	220	490	120	4	820	360	700	16.0			
9	95.00	96.00		14	390	330	410	6	510	310	400	11.0			
9	96.00	97.00		6	260	380	130	6	470	340	300	15.0			
9	97.00	98.00		14	420	430	100	-4	630	460	700	16.0			
9	98.00	99.00		22	280	580	140	-4	640	360	200	15.5			
9	99.00	100.00		16	400	600	540	-4	380	750	700	15.0			
9	100.00	101.00		-4	290	130	380	-4	250	230	100	6.2			
9	101.00	102.00		12	260	300	75	-4	910	290	200	14.5			
9	102.00	103.00		6	290	270	170	-4	670	300	150	8.7			
9	103.00	104.00		4	290	680	95	-4	880	200	250	7.9			
9	104.00	104.60		8	260	290	860	-4	400	250	200	9.4			
9	104.60	105.20		70	180	75	470	-4	18	1850	80	2.7			
9	105.20	106.50		22	60	12	270	8	2	180	100	1.6			
9	106.50	107.65		16	30	16	480	4	5	130	30	0.65			
9	107.65	108.40		16	15	16	760	-4	8	180	30	0.55			
9	108.40	109.00		44	10	90	860	-4	5	120	50	0.4			
9	109.00	110.00		44	25	65	860	-4	22	260	50	2.0			
9	110.00	111.00		50	25	140	960	-4	8	130	200	6.0			
9	111.00	112.00		50	25	130	960	-4	15	180	200	3.7			
9	112.00	113.15		680	130	1500	640	-4	95	230	150	10.5			
9	113.15	113.75		480	560	800	720		120	740	150	12.5			
9	113.75	115.00		55	180	160	490	-4	10	180	150	9.4			
9	115.00	116.00		50	210	80	800	12	15	210	200	8.0			
9	116.00	117.00		190	200	3400	520	-4	28	65	200	7.2			
9	117.00	118.00		640	180	7900	540	-4	65	200	150	8.1			
9	118.00	118.65		42	100	65	640	8	28	340	300	5.8			
9	118.65	122.00		80	40	65	120	-4	45	340	80	63.5			
9	122.00	123.00		75	120	42	110	-4	28	310	30	44.5			
9	123.00	125.00		10	40	160	340	-4	65	100	200	35.0			
9	125.00	126.00		12	230	6	290	14	15	130	50	3.7			
9	126.00	127.00		14	210	10	180	18	18	90	40	4.4			

cont'd 11

E.L. 7/74 "Moina" - 11 - Mineral analysis data from bore core samples

Bore No.	Depth Metres From	To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	Au ppm	Remarks
SMD 9	127.00	128.25		50	190	16	65	-4	20	150	30	2.0			
9	128.25	129.70		16	75	4	160	12	5	100	50	2.5			
DETECTION LIMIT				(4)	(10)	(4)	(4)	(4)			(1)		(1)	(0.05)	
METHOD OF ANALYSIS				X-RAY FLUORESCENCE SCHEME B1					CODE C1				CODE C2	CODE C3	

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L. 7/74 "MOI[illegible] SHEPHERD & MURPHY MINE AREA

Mineral analysis data from bore core samples

Bore No.	Depth From	Depth To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	Au ppm	Remarks
SMD 10	38.75	41.00		18	75	420	1800	-4	250	160	70	16.0			
10	41.00	42.00		60	65	1250	1900	6	190	110	200	15.0			
10	42.00	43.00		20	150	300	1450	-4	150	85	300	19.0			
10	43.00	44.30		90	220	720	1300	-4	200	100	250	15.5			
10	44.30	45.30		18	190	360	800	-4	170	110	300	21.5			
10	45.30	46.30		18	180	500	740	8	330	100	300	21.5			
10	46.30	47.55		22	330	1150	330	14	220	170	200	12.5			
10	47.55	49.00		18	140	520	780	6	220	120	300	21.5			
10	49.00	50.00		22	210	600	960	4	390	130	300	21.0			
10	50.00	50.75		18	120	580	900	-4	60	120	250	11.0			
10	50.75	51.65		24	160	360	940	-4	710	140	300	20.5			
10	51.65	52.40		24	180	250	820	-4	75	90	250	16.5			
10	52.40	53.30		24	120	410	1300	-4	170	270	250	12.5			
10	53.30	54.30		20	200	620	2000	-4	420	150	250	20.0			
10	54.30	55.30		20	160	460	2100	-4	45	110	300	21.0			
10	55.30	56.50		14	230	410	1400	-4	110	110	500	23.0			
10	56.50	58.00		12	320	660	1150	-4	410	840	300	24.0			
10	58.00	59.00		28	860	1700	960	50	290	340	300	22.0			
10	59.00	60.00		34.	1500	1450	800	36	1020	190	300	26.5			
10	60.00	61.00		24	980	780	660	16	260	380	300	20.0			
10	61.00	62.00		18	340	740	1350	6	170	530	300	20.5			
10	62.00	63.00		16	760	740	860	6	25	110	300	23.5			
10	63.00	63.75		14	350	760	1000	4	85	190	250	22.0			
10	63.75	65.00		18	360	480	640	6	180	800	250	14.0			
10	65.00	66.00		24	2250	2200	900	22	110	150	300	21.0			
10	66.00	67.00		20	430	720	1300	34	120	120	300	19.5			

Cont'd...../2.

Bore No.	Depth From	Metres To		Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	Au ppm	Remarks
SMD 10	67.00	68.00		18	360	330	2350	16	32	90	250	13.5			
10	68.00	69.00		18	620	460	3300	24	20	95	300	16.0			221403
10	69.00	70.00		16	1150	700	1450	55	40	95	500	25.9			
10	70.00	71.00		18	780	500	1250	34	40	130	500	25.5			
10	71.00	72.00		18	740	780	1550	60	55	110	500	22.5			
10	72.00	73.25		16	1450	480	1200	42	18	110	300	22.5			
10	73.25	73.60		500	1.31%	5.90%	4.10%	220	200	48	500	11.5			
10	73.60	75.00		32	900	1900	2200	26	65	90	300	15.5			
10	75.00	76.00		20	760	450	1400	24	90	90	500	17.5			
10	76.00	77.00		14	200	140	340	8	60	45	200	4.5			
10	77.00	78.00		24	100	180	920	8	8	70	250	4.4			
10	78.00	79.00		22	170	420	1050	6	10	80	70	5.0			
10	79.00	80.90		22	480	350	1050	38	60	180	500	14.5			
10	80.90	82.20		10	3200	270	2150	150	190	28	250	5.8			
10	82.20	83.35		16	640	220	1200	8	50	65	500	23.0			
10	83.35	84.40		22	180	80	460	6	50	55	200	8.8			
10	84.40	85.85		18	380	260	1050	14	120	60	300	21.5			
10	85.85	86.60		16	130	470	7000	130	710	38	50	14.0			
10	86.60	87.50		14	250	300	660	6	48	65	200	22.5			
10	87.50	88.50		12	240	390	480	-4	38	65	200	14.5			
10	88.50	89.50		18	410	320	450	42	25	40	150	8.5			
10	89.50	90.50		6	320	75	230	100	55	70	70	9.8			
10	90.50	91.50		4	280	22	100	95	8	40	70	6.6			
DETECTION LIMIT				(4)	(10)	(4)	(4)	(4)	(2)	(1)	(1)		(1)	(0.05)	
METHOD OF ANALYSIS			AMDEL	X-RAY FLUORESCENCE SHEME B1					AAS CODE C1		SPECTRO		CODE C2	CODE C3	

221404

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L. 7/74 "MOINA", SHEPHERD & MURPHY MINE AREA

Mineral analysis data from bore core samples

Bore No.	Depth Metres From	To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	Au ppm	Remarks
SMD 11	31.00	31.65		18	260	350	2150	8	180	90	200	15.6			
11	32.90	34.00		16	540	660	2300	-4	50	80	300	18.5			
11	34.00	35.00		28	740	640	2750	14	12	60	300	16.0			
11	35.00	36.00		12	250	410	1050	8	12	32	200	8.8			
11	36.00	37.00		18	800	740	1850	60	85	100	300	25.5			
11	37.00	38.00		16	330	220	2800	-4	10	35	70	6.1			
11	38.00	39.00		20	640	520	1850	42	45	95	500	23.0			
11	39.00	40.00		24	1450	580	1950	90	38	85	300	19.5			
11	40.00	41.00		20	580	540	1450	12	140	95	300	21.5			
11	41.00	42.00		26	1250	820	2400	46	350	75	200	24.0			
11	42.00	43.00		22	2400	600	2550	36	100	90	500	23.0			
11	43.00	44.00		18	1100	560	1050	44	130	120	300	25.5			
11	44.00	45.00		22	860	560	1200	26	170	130	300	24.5			
11	45.00	46.00		20	640	520	1000	18	85	130	300	21.5			
11	46.00	47.00		14	1300	450	980	18	120	130	300	22.0			
11	47.00	48.00		20	1100	540	1100	65	220	180	300	23.0			
11	48.00	49.00		16	640	350	980	26	38	120	300	20.5			
11	49.00	50.00		14	600	560	1050	50	85	860	200	12.5			
11	50.00	51.00		14	720	600	1200	14	130	210	250	17.5			
11	51.00	52.00		16	1500	500	1250	42	90	120	300	22.5			
11	52.00	53.00		16	860	440	1400	46	75	95	300	19.5			
11	53.00	54.00		18	500	360	1400	24	110	90	250	16.5			
11	54.00	55.00		22	700	700	1400	30	230	130	300	24.5			

E.L. 7/74 "Moina" -2- Mineral analysis data from bore core samples

Bore No.	Depth From	Depth To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	Au ppm	Remarks
SMD 11	56.00	57.00		20	680	500	1250	46	220	95	300	22.0			
11	57.00	58.00		20	540	410	1350	38	110	110	300	14.5			
11	58.00	59.00		18	1400	390	1150	36	170	75	300	12.5			
11	59.00	60.00		16	900	540	1250	30	200	75	250	16.5			
11	60.00	60.70		10	6500	210	1200	370	80	65	500	23.0			
11	60.70	61.70		12	1000	40	620	80	10	55	70	11.0			
11	61.70	63.10		22	640	360	1600	90	70	48	200	13.0			
11	63.10	64.00		24	1650	560	1250	65	240	60	300	9.1			
11	64.00	65.00		18	1200	240	1700	65	220	65	300	14.0			
11	65.00	66.00		18	1100	280	1000	40	160	55	300	12.5			
11	66.00	67.00		20	470	490	1350	24	220	75	150	8.7			
11	67.00	68.00		16	820	190	2100	40	150	450	200	7.2			
11	68.00	68.50		65	540	260	1000	50	190	100	70	18.5			
11	68.50	69.20		30	50	180	1200	-4	20	90	70	4.2			
11	69.20	69.65		24	720	300	1800	70	270	60	150	11.5			
11	69.65	70.85		90	1100	18	620	85	10	50	150	9.6			
11	70.85	71.85		20	1100	12	920	12	10	65	200	11.0			
11	71.85	73.00		16	2300	36	2950	220	120	65	150	9.2			
11	73.00	74.00		12	900	8	1150	38	80	65	70	8.9			
DETECTION LIMIT				(4)	(10)	(4)	(4)	(4)	(2)	(1)	(1)				
METHOD OF ANALYSIS		AMDEL		X-RAY FLUORESCENCE SCHEME B1					AAS CODE C1		SPECTRO				

221406

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L. 7/74 "MOINA", SHEPHERD & MURPHY MINE AREA

Mineral analysis data from bore core samples

Bore No.	Depth Metres From	To	Log	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	Au ppm	Remarks
SMD 12	90.50	91.50		20	15	10	16	-4	15	18	70	0.4			
12	91.50	92.90		12	10	46	190	-4	38	85	50	1.0			
12	92.90	94.00		18	170	460	390	16	240	110	500	11.9			
12	94.00	95.35		12	-10	75	34	6	18	25	50	1.5			
12	95.35	96.00		26	10	40	170	12	5	55	150	4.3			
12	96.00	97.00		22	15	130	500	8	8	50	200	10.5			
12	97.00	98.00		26	260	330	980	8	330	90	500	22.0			
12	98.00	99.00		26	170	460	700	10	180	80	300	15.5			
12	99.00	100.00		26	300	580	720	18	230	100	300	20.5			
12	100.00	101.00		24	140	580	540	6	210	85	200	13.5			
12	101.00	102.00		22	270	500	540	-4	480	110	300	19.0			
12	102.00	103.00		38	75	310	430	-4	75	110	200	10.0			
12	103.00	104.00		38	40	400	780	6	220	120	150	7.8			
12	104.00	104.80		30	270	580	960	6	80	100	200	12.5			
12	104.80	106.00		20	15	130	1050	12	20	50	50	3.4			
12	106.00	107.00		24	55	310	380	4	18	48	70	4.4			
12	107.00	108.00		90	400	2300	270	34	50	38	70	4.4			
12	108.00	109.50		20	260	230	780	24	75	45	150	7.7			
12	109.50	110.50		10	55	28	130	-4	45	25	50	1.2			
12	110.50	111.50		10	75	22	38	8	38	22	50	1.8			
DETECTION LIMIT				(4)	(10)	(4)	(4)	(4)	(2)	(1)	(1)				
METHOD OF ANALYSIS			AMDEL	X-RAY FLUORESCENCE SCHEME B1					AAS CODE C1		SPECTRO				

Devonport

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L. 7/74 "MOINA", SHEPHERD & MURPHY MINE AREA.

Mineral analysis data from bore core samples

Bore No.	Depth m From	Depth m To	Log	F %	Sn ppm	W ppm	Bi ppm	Mo ppm	Be ppm	Cu ppm	Pb ppm	Zn ppm	Cd ppm	In ppm	Ge ppm	Au ppm	Sc ppm	Ce ppm	Y ppm	Ta ppm	Remarks
SMD 13	7.00	9.00		-0.1	-4	-10	-10	-4	2	30	8	25	-1	4	-2	-0.05	5	-300	10	-100	
13	9.00	11.00		-0.1	-4	20	-10	4	3	42	32	100	-1	-4	-2	-0.05	10	-300	50	-100	
13	35.00	36.15		0.2	50	10	12	-4	40	60	25	270	1	4	-2	-0.05	5	-300	30	-100	
13	36.15	36.95		0.3	240	-10	240	-4	10	1080	22	250	-1	12	-2	4.5	5	-300	30	-100	
13	36.95	37.50		0.2	210	-10	-10	-4	10	140	70	130	-1	4	2	-0.05	10	-300	50	-100	
13	80.00	81.00		0.1	-4	15	-10	-4	3	10	18	420	-1	-4	-2	-0.05	10	-300	50	-100	
13	81.00	82.00		0.1	6	15	-10	-4	5	12	8	1420	2	-4	-2	-0.05	10	-300	50	-100	
13	82.00	83.00		0.1	-4	-10	-10	-4	5	2	8	410	-1	6	-2	-0.05	10	-300	20	-100	
13	83.00	83.70		0.1	-4	-10	-10	-4	10	12	8	290	-1	-4	-2	-0.05	15	-300	50	-100	
13	83.70	84.70		0.2	600	25	15	-4	30	8	18	130	-1	8	-2	-0.05	10	-300	20	-100	
13	84.70	85.50		0.3	490	25	18	-4	30	8	15	140	-1	6	-2	-0.05	5	-300	20	-100	
13	85.50	86.30		0.1	500	85	60	8	30	12	42	260	-1	4	-2	0.10	10	-300	30	-100	
13	86.30	87.35		0.6	720	55	1100	48	30	42	100	820	3	16	4	1.05	5	-300	20	-100	
13	87.35	88.50		0.1	470	10	650	-4	30	570	110	6.3%	230	48	3	0.30	10	-300	30	-100	
13	88.50	89.10		-0.1	360	-20	900	-4	50	640	110	13.8%	620	42	-2	0.50	10	-300	30	-100	
13	89.10	90.00		0.1	190	-50	660	-4	70	860	45	27.5%	1340	90	-2	0.80	5	-300	30	-100	
13	90.00	91.00		0.2	330	-20	1950	-4	40	760	38	14.5%	700	70	-2	1.35	5	-300	20	-100	
13	91.00	92.00		0.1	460	-20	1880	4	40	160	55	5.25%	210	32	5	0.80	15	-300	30	-100	
13	92.00	93.00		-0.1	430	-20	1720	-4	40	360	50	12.5%	490	46	-2	1.30	10	-300	30	-100	
13	93.00	94.00		-0.1	500	-20	1190	-4	10	460	55	11.5%	440	50	-2	0.65	10	-300	30	-100	
13	94.00	95.00		-0.1	520	-10	2100	4	20	340	85	8.5%	380	42	4	1.20	10	-300	20	-100	
13	95.00	96.00		-0.1	360	-20	1890	-4	30	470	55	13.0%	600	26	2	1.80	10	-300	20	-100	
13	96.00	97.00		-0.1	490	-10	250	-4	20	470	25	5.4%	210	20	2	0.35	15	-300	50	-100	
13	97.00	98.25		0.3	380	10	150	-4	80	18	28	4300	15	8	2	0.15	10	-300	10	-100	
13	98.25	99.25		0.6	500	20	90	-4	100	28	18	1.31%	40	10	-2	0.10	10	-300	20	-100	
13	99.25	100.25		0.2	370	-20	470	-4	70	380	38	7.75%	260	55	-2	0.50	5	-300	10	-100	
13	100.25	101.25		0.1	340	-20	320	4	70	910	28	10.4%	390	30	-2	0.30	10	-300	30	-100	
13	101.25	102.25		0.5	390	-10	42	10	30	580	21	7.0%	250	36	-2	0.15	10	-300	30	-100	

Mineral analysis data from bore core samples

Bore No.	Depth From	Depth To	Log	F %	Sn ppm	W ppm	Bi ppm	Mo ppm	Be ppm	Cu ppm	Pb ppm	Zn ppm	Cd ppm	In ppm	Ge ppm	Au ppm	Sc ppm	Ce ppm	Y ppm	Ta ppm	Remarks
SMD 13	102.25	103.25		0.6	440	-10	120	-4	50	310	25	4.65%	160	60	-2	-0.05	10	-300	30	-100	
13	103.25	104.00		0.7	500	10	220	-4	80	55	42	1.63%	55	36	2	-0.05	15	-300	30	-100	
13	104.00	105.00		0.5	560	15	25	-4	50	18	20	230	-1	10	2	-0.05	10	-300	20	-100	
13	105.00	106.00		0.5	580	20	40	-4	80	10	18	220	-1	14	2	-0.05	10	-300	50	-100	
13	106.00	107.00		0.5	520	30	38	4	100	10	18	220	-1	22	-2	-0.05	10	-300	50	-100	
13	107.00	108.00		0.3	490	20	22	-4	70	8	25	150	-1	10	4	-0.05	10	-300	30	-100	
13	108.00	109.00		0.5	580	10	-10	6	50	8	30	120	-1	-4	-2	0.10	10	-300	30	-100	
13	109.00	110.00		0.7	400	35	15	-4	100	8	25	160	-1	10	4	-0.05	20	-300	50	-100	
13	110.00	111.00		0.3	380	25	30	4	50	8	30	120	-1	10	-2	-0.05	10	-300	30	-100	
13	111.00	112.00		0.2	150	40	320	6	80	15	60	180	-1	-4	3	-0.05	20	-300	30	-100	
13	112.00	113.00		5.6	220	120	70	-4	200	18	22	280	-1	-4	-2	0.15	5	-300	30	-100	
13	113.00	114.00		0.5	240	20	55	8	200	8	38	190	-1	-4	-2	-0.05	10	-300	30	-100	
13	114.00	115.00		0.7	310	-10	28	-4	100	8	28	170	-1	-4	-2	-0.05	10	-300	30	-100	
13	115.00	116.00		0.2	210	-10	-10	4	70	8	25	160	-1	14	-2	0.05	10	-300	30	-100	
13	116.00	117.25		0.4	270	40	-10	-4	50	8	30	240	-1	4	-2	-0.05	10	-300	30	-100	
13	117.25	118.25		0.8	200	40	15	-4	70	8	15	160	-1	-4	2	0.10	5	-300	30	-100	
13	118.25	119.30		1.0	280	20	85	28	70	10	38	160	-1	4	-2	0.10	10	-300	50	-100	
13	119.30	120.30		1.0	100	-10	140	10	70	5	28	110	-1	-4	2	0.10	5	-300	50	-100	
13	120.30	121.30		2.1	140	150	18	16	70	2	15	150	-1	12	2	-0.05	5	-300	50	-100	
13	121.30	122.35		1.1	180	25	-10	34	70	5	12	120	-1	10	-2	-0.05	5	-300	50	-100	
13	122.35	123.12		0.8	250	25	-10	12	70	8	15	220	-1	8	-2	-0.05	10	-300	50	-100	
13	123.12	124.00		5.0	180	620	-10	70	500	15	18	250	-1	-4	5	-0.05	20	-300	70	-100	
13	124.00	131.00		0.4	10	30	-10	6	30	35	22	90	-1	-4	-2	0.05	20	-300	50	-100	
13	131.00	132.00		0.7	40	360	-10	6	50	60	18	65	-1	8	-2	-0.05	10	-300	50	-100	
13	132.00	133.00		1.7	8	1850	12	42	1000	32	25	140	-1	-4	-2	-0.05	20	-300	70	-100	
13	133.00	134.00		1.1	55	70	95	32	300	130	22	140	-1	6	3	0.05	20	-300	70	-100	
13	176.00	177.00		-0.1	16	15	-10	120	50	22	-5	42	-1	6	-2	-0.05	5	-300	20	-100	
13	177.00	178.00		-0.1	44	35	-10	8	30	42	60	130	1	-4	-2	-0.05	5	-300	30	-100	
	DETECTION LIMIT			(0.1)	(4)	(10)	(10)	(4)	(1)	(2)	(5)	(1)	(1)	(4)	(2)	(0.05)	(3)	(300)	(10)	(100)	
	ALL ANALYSES BY AMDEL METHOD			E 3 SIE	B 1 XRF	B 1 XRF	C 1 AAS	B 1 XRF	A 1 SPECTRO SEMI-QUANT-ITATIVE	C 1 AAS	C 1 AAS	C 1 AAS	C 1 AAS	B 1 XRF	B 1 XRF	C 3 AAS	A 1 SPECTRO	A 1 SEMI-QUANTITATIVE	A 1	A 1	

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L. 7/74 "MOINA", SHEPHERD & MURPHY MINE AREA

Mineral analysis data from bore core samples

Bore No.	Depth Metres From	To	Log	CaF_2 %	Sn ppm	W ppm	Bi ppm	Mo ppm	Be ppm	Cu ppm	Pb ppm	Zn ppm	Cd ppm	Sc ppm	Y ppm	Remarks
SMD 15	33.00	35.00		17.5	1750	210	500	-3	200					3	10	
15	35.00	38.12		23.4	1200	450	500	10	250					5	20	
15	38.12	40.00		20.3	1900	370	700	10	300					3	10	
15	40.00	42.00		18.7	1650	540	700	5	300					3	10	
15	42.00	43.60		5.8	660	180	250	5	100					5	30	
15	43.60	45.00		22.2	1850	640	700	5	300					5	20	
15	45.00	47.00		23.8	1550	800	500	5	500					3	10	
15	47.00	49.00		21.8	2350	660	500	10	300					5	10	
15	49.00	51.00		18.3	2200	370	500	-3	200					5	20	
15	51.00	53.00		18.9	3600	1450	700	30	250					5	20	
15	53.00	55.00		19.7	2050	680	500	50	300					5	20	
15	55.00	57.00		18.9	2000	860	500	50	300					5	20	
15	57.00	59.00		25.4	1500	1200	500	80	400					5	20	
15	59.00	61.00		24.5	1250	1400	500	70	250					3	10	
15	61.00	63.00		21.6	1300	820	300	70	300					5	10	
15	63.00	65.00		18.1	1650	780	400	10	300					5	10	
15	65.00	67.00		19.3	1400	780	500	10	250					3	10	
15	67.00	69.00		16.4	1350	760	300	70	300					5	20	
15	69.00	71.00		18.9	1150	760	700	70	200					5	10	
15	71.00	73.00		18.9	1250	1150	500	30	300					5	10	
15	73.00	75.00		17.7	1450	1850	300	100	300					5	10	
15	75.00	77.00		25.5	2100	1200	700	10	300					3	-10	
15	77.00	79.00		24.9	1600	760	500	5	500					3	10	
15	79.00	81.00		22.8	1350	620	700	20	250					5	20	
15	81.00	83.00		16.4	1350	560	500	30	300					5	20	
15	83.00	85.00		18.1	1550	1450	700	80	300					8	30	
15	85.00	86.30		20.5	1700	860	500	70	300					3	20	
15	86.30	88.00		6.6	480	1250	70	10	150					3	20	
15	88.00	90.00		8.2	880	360	200	80	150					5	30	
15	90.00	92.00		19.7	1750	7500	300	80	200					5	30	
15	92.00	94.00		14.4	1350	820	300	100	300					5	20	
15	94.00	96.00		15.8	1100	6200	200	150	150					10	30	
15	96.00	98.00		10.8	1000	720	200	70	150					5	30	
15	98.00	100.70		13.6	1200	1400	300	70	300					5	20	
15	100.70	103.00		9.7	360	3650	200	150	250					20	70	
15	103.00	105.00		10.3	460	370	200	- 3	200					10	30	
15	105.00	107.00		6.7	470	110	70	10	300					5	20	

Continued...../2.

E.L. 7/74 "MOINA" -2- Mineral analysis data from bore core samples

Bore No.	Depth Metres From	Depth Metres To	Log	CaF_2 %	Sn ppm	W ppm	Bi ppm	Mo ppm	Be ppm	Cu ppm	Pb ppm	Zn ppm	Cd ppm	Sc ppm	Y ppm	Remarks
SMD 15	107.00	109.25		9.0	360	120	70	10	150					15	30	
15	109.25	111.00		3.5	90	90	5	10	200					10	50	
15	111.00	113.00		2.0	34	75	5	70	100					10	50	
15	113.00	115.00		1.5	26	25	10	30	80					10	50	
15	115.00	116.12		2.3	60	45	-1	10	100					3	30	
	DETECTION LIMIT				(4)	(10)	(1)	(3)	(1)					(3)	(10)	
	ALL ANALYSES BY AMDEL - METHOD			E 3 SIE	B 1 XRF	B 1 XRF	A1/2 SPECTRO SEMI-QUANTITATIVE	A1/2	A1/2					A1/2 SPECTRO SEMI-QUANTITATIVE	A1/2	

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L. 7/74 "MOINA", SHEPHERD & MURPHY MINE AREA

Mineral analysis data from bore core samples

Bore No.	Depth Metres From	To	Log	CaF_2 %	Sn ppm	W ppm	Bi ppm	Mo ppm	Be ppm	Cu ppm	Pb ppm	Zn ppm	Cd ppm	Sc ppm	Y ppm	Remarks
SMD																
16	26.27	28.10		2.9	490	140	800	-3	80			3.80%		5	20	
16	28 0	30.22		1.6	1050	15	70	3	50			4650		5	20	
16	30.22	32.00		5.6	1550	140	20	10	70			190		-3	10	
16	32.00	34.00		6.2	380	450	30	80	70			320		10	30	
16	34.00	36.00		5.6	960	330	50	50	80			130		10	30	
16	36.00	38.00		4.7	1200	440	70	10	70			750		5	30	
16	38.00	40.00		5.4	1250	540	800	80	80			4.30%		5	20	
16	40.00	42.35		5.3	760	1050	300	30	70			4.10%		3	10	
16	42.35	44.20		22.6	1400	540	200	5	200			530		3	10	
16	44.20	46.20		20.5	1650	580	500	10	300			300		5	10	
16	46.20	48.20		21.4	1500	940	300	10	300			230		5	30	
16	48.20	50.20		23.0	1050	1850	700	30	250			220		5	20	
16	50.20	52.20		4.5	720	450	50	10	100			150		5	10	
16	52.20	54.20		1.1	1550	75	100	3	70			410		5	30	
16	54.20	55.70		1.0	920	20	70	-3	80			360		5	30	
16	55.70	57.20		7.1	410	170	50	70	80			940		5	30	
16	57.20	59.20		15.3	800	820	100	30	100			1120		5	20	
16	59.20	61.20		17.1	1850	1800	2000	20	150			270		5	20	
16	61.20	63.20		16.2	1200	3600	400	70	150			280		3	-10	
16	63.20	65.50		14.8	1700	2100	500	150	150			2.85%		3	20	
16	65.50	68.10		2.9	700	170	400	20	80			6.60%		3	20	
16	68.10	70.00		7.2	190	1950	1000	80	80			11.70%		3	30	
16	70.00	72.25		1.5	740	340	700	80	50			2.90%		3	20	
16	72.55	76.05		10.7	520	260	300	50	80			1.22%		3	20	
16	76.05	78.05		13.6	520	1250	300	70	400			1580		5	30	
16	78.05	81.60		4.2	430	110	200	50	150			600		10	30	
16	81.60	82.35		22.4	240	370	150	70	150			320		5	20	
16	82.35	85.10		21.4	370	6500	300	10	200			300		10	30	
16	85.10	87.10		7.6	560	80	700	70	200			5400		10	30	
16	87.10	88.70		6.7	520	2050	700	200	150			1.50%		10	30	
16	88.70	91.00		14.4	280	1.27%	700	200	150			630		10	30	
16	91.00	93.00		3.7	470	100	1000	80	200			200		10	30	
16	93.00	95.00		3.0	560	55	300	10	70			150		5	30	
16	95.00	97.00		5.0	760	85	200	200	100			130		5	30	
16	97.00	99.00		4.9	740	100	500	10	200			190		5	20	
16	99.00	101.70		4.4	340	470	300	20	150			15.40%		3	-10	
16	101.70	104.27		2.4	780	40	200	-3	200			5.70%		3	20	
16	104.27	106.00		24.7	1000	430	700	10	200			850		3	-10	
16	106.00	108.00		21.8	1700	450	500	3	250			450		3	-10	
16	108.00	110.00		22.8	1900	600	700	20	500			260		3	10	
16	110.00	112.00		21.4	660	350	500	10	200			200		3	10	
16	112.00	114.00		21.6	820	600	700	20	300			390		3	10	

E.L. 7/74 "MOINA" -2- Mineral analysis data from bore core samples

Bore No.	Depth From	Depth To	Log	CaF_2 %	Sn ppm	W ppm	Bi ppm	Mo ppm	Be ppm	Cu ppm	Pb ppm	Zn ppm	Cd ppm	Sc ppm	Y ppm	Remarks
SMD 16	114.00	116.00		17.3	1650	540	800	20	300			430		3	10	
16	116.00	118.00		18.9	2000	620	500	10	250			800		3	10	
16	118.00	120.50		19.7	1750	480	300	3	400			340		3	10	
16	120.50	123.00		4.3	1300	140	800	3	100			230		5	30	
16	123.00	125.43		2.6	720	55	500	-3	100			2520		10	30	
16	125.43	128.75		21.8	2650	1150	500	70	200			300		5	20	
16	128.75	131.00		5.3	720	220	300	-3	200			90		5	30	
16	131.00	133.00		2.5	410	1050	100	70	150			55		5	30	
16	133.00	135.00		0.7	430	20	30	-3	70			60		5	30	
16	135.00	137.00		4.4	350	360	30	80	80			75		10	30	
16	154.00	156.00		11.9	260	260	20	30	80			90		15	70	
DETECTION LIMIT					(4)	(10)	(1)	(3)	(1)					(3)	(10)	
ALL ANALYSES BY AMDEL - METHOD				E 3 SIE	B 1 XRF	B 1 XRF	A1/2 SPECTRO SEMI-QUANTITATIVE	A1/2	A1/2			Cl AAS OR Fl/1 AAS for % Zn.		A1/2 SPECTRO SEMI-QUANTITATIVE	A1/2	

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L. 7/74 "MOINA", SHEPHERD & MURPHY MINE AREA

Mineral analysis data from bore core samples

Bore No.	Depth Metres From	To	CaF_2 %	Sn ppm	W ppm	Bi ppm	Mo ppm	Be ppm	Cu ppm	Pb ppm	Zn ppm	Cd ppm	Sc ppm	Y ppm	Remarks
SMD 17	26.50	28.00	11.9	360	140	700	5	200					5	20	
17	28.00	29.00	11.9	340	280	500	3	250					10	30	
17	29.00	30.00	11.7	340	290	700	5	200					10	30	
17	30.00	31.00	13.1	280	250	500	5	200					5	20	
17	31.00	32.00	14.0	420	270	500	5	300					10	50	
17	32.00	33.00	12.7	250	160	500	3	300					5	50	
17	33.00	34.00	14.7	310	280	500	10	300					10	50	
17	34.00	35.15	4.4	110	140	200	30	80					15	70	
17	35.15	37.00	5.8	110	110	100	-3	100					3	10	
17	37.00	38.90	9.3	170	130	300	5	150					5	30	
17	38.90	39.90	18.1	300	250	500	-3	250					5	10	
17	39.90	40.90	12.9	350	260	500	5	150					10	30	
17	40.90	41.90	12.3	280	270	300	5	150					5	20	
17	41.90	43.90	7.2	200	170	100	-3	150					5	30	
17	43.90	45.90	3.6	65	50	50	-3	100					5	50	
17	45.90	47.90	0.9	24	20	5	5	50					20	100	
17	47.90	49.90	1.5	40	30	30	-3	100					10	50	
DETECTION LIMIT				(4)	(10)	(1)	(3)	(1)					(3)	(10)	
ALL ANALYSES BY AMDEL - METHOD			E 3 SIE	B 1 XRF	B 1 XRF	A1/2 SPECTRO SEMI-QUANTITATIVE		A1/2					A1/2 SPECTRO SEMI-QUANTITATIVE	A1/2	

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L. 7/74 "MOINA", SHEPHERD & MURPHY MINE AREA

Analyses of split core.

Bore No.	Depth From	Metres To		CaF_2 %	Sn ppm	W ppm	Bi ppm	Mo ppm	Be ppm	Zn ppm	Ag ppm	Ge ppm	Sc ppm	Y ppm	Sr ppm	In ppm	Au ppm	Remarks
SMD 21	105.70	108.00		2.05	220	-10	70	-3	100	500	0.2	2	5	30	-50	-10	-0.05	
	108.00	110.00		0.80	210	20	70	-3	20	300	0.1	-1	5	50	-50	-10	-0.05	
	110.00	112.50		0.45	370	-10	150	-3	30	200	0.5	-1	3	30	-50	-10	0.10	
	112.50	114.50		3.30	650	35	150	30	150	3000	0.2	1	5	30	-50	-10	-0.05	
	114.50	116.50		6.20	600	20	150	30	100	100	0.1	1	10	70	-50	-10	-0.05	
	116.50	118.50		6.20	610	240	300	50	100	30	-0.1	3	10	70	100	-10	0.05	
	118.50	120.50		6.80	470	20	50	10	100	100	-0.1	-1	5	50	-50	-10	-0.05	
	120.50	122.10		6.20	960	340	300	10	150	80	0.1	2	15	70	50	-10	-0.05	
	122.10	124.50		16.40	1050	1050	500	100	200	200	-0.1	5	5	50	100	-10	-0.05	
DETECTION LIMIT					(4)	(10)	(1)	(3)	(1)	(20)	(0.1)	(1)	(3)	(10)	(50)	(10)		
ALL ANALYSES BY AMDEL - METHOD.				E 3 SIE	B 1 XRF	B 1 XRF	A1/2 SPECTRO	A1/2 SEMI -	A1/2 QUANTITATIVE	A1/2	A1/2	A1/2	A1/2	A1/2	A1/2	A1/2	C 3 AAS	

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L. 7/74 "MOINA", SHEPHERD & MURPHY MINE AREA,

Analyses of split core

Bore No.	Depth From	Metres To		CaF_2 %	Sn ppm	W ppm	Bi ppm	Mo ppm	Be ppm	Zn ppm	Ag ppm	Ge ppm	Sc ppm	Y ppm	Sr ppm	In ppm	Au ppm	Remarks
SMD 22	69.40	71.40		.40	240	-10	30	3	50	300	-0.1	2	7	50	-50	-10	-0.05	
	71.40	74.00		.30	620	65	300	-3	30	700	2.0	2	5	30	50	-10	-0.05	
	78.40	80.40		.20	260	-10	30	3	20	300	-0.1	1	7	30	-50	-10	-0.05	
	80.40	82.40		.15	350	55	30	-3	10	100	-0.1	1	5	30	-50	-10	-0.05	
	82.40	84.40		.20	80	-10	5	20	20	200	-0.1	-1	10	80	-50	-10	-0.05	
	84.40	86.40		.15	80	-10	250	-3	30	1000	0.5	-1	5	50	-50	-10	0.10	
	86.40	87.85		1.20	95	15	50	-3	50	800	0.5	-1	3	30	50	-10	-0.05	
	87.85	89.70		0.70	28	-10	15	-3	50	500	-0.1	-1	3	10	200	-10	-0.05	
	89.70	90.70		10.00	290	220	500	20	300	5000	0.7	8	5	80	80	-10		
DETECTION LIMIT					(4)	(10)	(1)	(3)	(1)	(20)	(0.1)	(1)	(3)	(10)	(50)	(10)		
ALL ANALYSES BY AMDEL - METHOD.				E 3 SIE	B 1 XRF	B 1 XRF	A1/2	A1/2	A1/2	A1/2	A1/2	A1/2	A1/2	A1/2	A1/2	A1/2	C 3 AAS	
										SPECTRO SEMI-QUANTITATIVE								

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L. 7/74 "MOINA", SHEPHERD & MURPHY MINE AREA.

Analyses of split core

Bore No.	Depth From	Metres To		CaF_2 %	Sn ppm	W ppm	Bi ppm	Mo ppm	Be ppm	Zn ppm	Ag ppm	Ge ppm	Sc ppm	Y ppm	Sr ppm	In ppm	Au ppm	Remarks
SMD 23	6.00	8.00		15.0	560	320	500	3	200	300	0.2	20	3	30	50	-10	0.05	
	8.00	10.00		11.9	670	310	200	3	300	300	-0.1	10	3	20	-50	-10	-0.05	
	10.00	12.00		11.9	580	290	300	-3	300	300	-0.1	15	7	30	80	-10	-0.05	
	12.00	13.60		4.1	620	50	200	-3	200	300	-0.1	15	7	50	80	-10	-0.05	
	13.60	15.65		14.0	730	370	300	3	300	200	0.1	15	5	20	80	-10	0.05	
	15.65	18.80		14.2	600	160	300	-3	250	80	-0.1	5	7	70	80	-10	-0.05	
	18.80	20.60		10.7	95	140	150	-3	200	80	-0.1	1	3	70	-50	-10	-0.05	
	20.60	22.00		10.0	530	150	300	-3	300	300	-0.1	7	10	50	50	-10	-0.05	
	22.00	24.00		10.7	490	210	80	-3	300	100	-0.1	1	7	30	50	-10	-0.05	
	24.00	27.00		4.1	160	120	200	-3	100	200	-0.1	-1	7	70	100	-10	-0.05	
	27.00	28.50		2.25	34	65	300	-3	70	100	-0.1	-1	3	30	-50	-10	0.05	
DETECTION LIMIT					(4)	(10)	(1)	(3)	(1)	(20)	(0.1)	(1)	(3)	(10)	(50)	(10)		
ALL ANALYSES BY AMDEL - METHOD.				E 3 SIE	B 1 XRF	B 1 XRF	Al/2	Al/2	Al/2	Al/2	Al/2	Al/2	Al/2	Al/2	Al/2	Al/2	C 3 AAS	
										SPECTRO SEMI-QUANTITATIVE								

APPENDIX 26

RECOVERY LOGS, DIAMOND DRILLING, MOINA.

NOTE:

In most cases adequate compensation for gaps along the core trays has not been made by the field assistant. Thus there are many listed instances of core recoveries exceeding 100%.

PROJECT: MOINA HOLE NO: ML 1A RECOVERY LOG 221418

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0.0	5' 0"	5' 0"	4' 8"	93.33
5' 0"	10' 0"	5' 0"	4' 9"	95
10' 0"	15' 0"	5' 0"	4' 7"	91.66
15' 0"	18' 0"	3' 0"	2' 9"	91.66
18' 0"	95' 0"	77' 0"	2' 0"	2.59
95' 0"	108' 0"	13' 0"	1' 0"	7.69
108' 0"	109' 0"	1' 0"	1' 0"	100
109' 0"	110' 0"	2' 0"	1' 6"	75
110' 0"	113' 0"	2' 0"	1' 6"	75
113' 0"	117' 0"	4' 0"	1' 0"	25
117' 0"	121' 0"	4' 0"	0' 8"	16.66
121' 0"	127' 0"	8' 0"	3' 6"	43.75
127' 0"	160' 0"	33' 0"	4' 3"	12.87
160' 0"	162' 0"	2' 0"	1' 6"	75
162' 0"	179' 0"	17' 0"	3' 0"	17.64
179' 0"	185' 6"	6' 6"	5' 6"	83.33
185' 6"	187' 6"	2' 0"	2' 0"	100
187' 6"	190' 0"	2' 6"	2' 6"	100
190' 0"	194' 9"	4' 9"	4' 9"	100
194' 9"	200' 0"	5' 3"	4' 0"	76.19
200' 0"	201' 0"	1' 0"	0' 9"	75
201' 0"	202' 0"	1' 0"	1' 0"	100
202' 0"	204' 0"	2' 0"	2' 0"	100
204' 0"	206' 0"	2' 3"	1' 0"	44.44
206' 3"	207' 9"	1' 6"	1' 0"	66.66
207' 9"	211' 4"	3' 7"	4' 6"	125.58
211' 4"	214' 5"	3' 1"	2' 1"	67.56
214' 5"	216'10"	2' 5"	1' 5"	58.61
216'10"	219' 0"	2' 2"	1' 4"	61.53
219' 0"	225' 6"	6' 6"	0' 8"	10.25
225' 6"	226' 6"	1' 0"	1' 9"	175
226' 6"	227' 0"	0' 6"	1' 7"	316.66
227' 0"	229' 5"	2' 5"	2'10"	117.24
229' 5"	231' 6"	2' 1"	0' 7"	27.99
231' 6"	236' 9"	5' 3"	0' 9"	14.28
236' 9"	252' 0"	15' 9"	4' 9"	30.15
252' 0"	261' 0"	9' 0"	2' 2"	24.07
261' 0"	265' 0"	4' 0"	3' 2"	79.16
265' 0"	268' 0"	3' 0"	4' 4"	144.44
268' 0"	270' 0"	2' 0"	2' 0"	100
270' 0"	280' 0"	10' 0"	13' 4"	133.33
280' 0"	284' 0"	4' 0"	5' 0"	125
284' 0"	286' 6"	2' 6"	3' 0"	120
286' 6"	293' 9"	7' 3"	8' 4"	114
293' 9"	304' 9"	11' 0"	1' 2"	10.60
304' 9"	310' 0"	5' 3"	4' 9"	90.47
310' 0"	317' 2"	7' 2"	9' 3"	129.07
317' 2"	327' 8"	10' 6"	10' 9"	102.38
327' 8"	332' 8"	5' 0"	5' 0"	100
332' 8"	338' 0"	5' 4"	5'10"	109.37
338' 0"	344' 0"	6' 0"	3'10"	63.88
344' 0"	351' 6"	7' 6"	10' 2"	135.55
351' 6"	360' 9"	9' 3"	9' 3"	100
360' 9"	366' 0"	5' 3"	6' 2"	117.45
366' 0"	368' 5"	2' 5"	2' 6"	103.45
368' 5"	375' 0"	6' 5"	2' 5"	37.66
375' 0"	381' 0"	6' 0"	8' 0"	133.33

PROJECT: MOINA HOLE NO: ML 1A RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
381' 0"	384' 0"	3' 0"	3' 1"	102.77
384' 0"	388' 8"	4' 8"	5' 7"	119.64
388' 8"	391' 2"	2' 4"	3' 3"	139.28
391' 2"	401' 4"	10' 2"	8' 6"	83.60
401' 4"	414' 0"	12' 8"	6' 3"	49.34
414' 0"	418' 4"	4' 4"	8' 0"	184.61
418' 4"	421' 4"	3' 0"	3' 2"	105.55
421' 4"	426' 0"	4' 8"	6' 0"	128.57
426' 0"	428' 0"	2' 0"	1' 6"	75
428' 0"	438' 0"	10' 0"	10' 8"	106.66
438' 0"	443' 0"	5' 0"	4'11"	98.33
443' 0"	444' 0"	1' 0"	1' 4"	133.33
444' 0"	447' 0"	3' 0"	3' 2"	105.55
447' 0"	454' 0"	7' 0"	6' 7"	94.04
454' 0"	463' 0"	9' 0"	9' 0"	100
463' 0"	468' 0"	5' 0"	5' 0"	100
468' 0"	473' 0"	5' 0"	7' 0"	140
473' 0"	481' 0"	8' 0"	9' 5"	117.70
481' 0"	488' 0"	7' 0"	6' 4"	90.47
488' 0"	490' 0"	2' 0"	2' 8"	133.33
490' 0"	492' 0"	2' 0"	2' 0"	100
492' 0"	500' 0"	8' 0"	5' 0"	62.5
500' 0"	503' 6"	3' 6"	3' 0"	85.71
503' 6"	506' 0"	2' 6"	3' 0"	120
506' 0"	510' 9"	4' 9"	2' 0"	42.10
510' 9"	516'10"	6' 1"	7' 8"	126.02
516'10"	520'10"	4' 0"	2' 8"	66.66
520'10"	527' 8"	6'10"	9' 6"	139.02
527' 8"	531' 0"	3' 4"	4' 4"	130
531' 0"	539' 3"	8' 3"	6' 4"	76.76
539' 3"	543' 0"	3' 9"	3'10"	102.22
543' 0"	545' 5"	2' 5"	2' 4"	96.55
545' 5"	550' 0"	4' 7"	2' 1"	45.45
550' 0"	551' 0"	1' 0"	2' 2"	216
551' 0"	557' 0"	6' 0"	2' 4"	38.88
557' 0"	566' 0"	9' 0"	5' 8"	62.96
566' 0"	569' 0"	3' 0"	4' 4"	144.44
569' 0"	570' 6"	1' 6"	1' 0"	66.66
570' 6"	572' 6"	2' 0"	2' 0"	100
572' 6"	575' 6"	3' 0"	1' 8"	55.55
575' 6"	576' 6"	1' 0"	0' 6"	50
576' 6"	583' 6"	7' 0"	7' 8"	109.52
583' 6"	593' 6"	10' 0"	11' 0"	110
593' 6"	599' 0"	5' 6"	5' 0"	90.90
599' 0"	601' 9"	2' 9"	1' 8"	60.60
601' 9"	603' 9"	2' 0"	2' 3"	112.50
603' 9"	604' 6"	0' 9"	0' 3"	33.33
604' 6"	614' 0"	9' 6"	10' 7"	111.40
614' 0"	620' 0"	6' 0"	7' 0"	116.66'
620' 0"	624' 0"	4' 0"	3' 4"	83.33
624' 0"	626' 0"	no core	no core	no core
626' 0"	627' 6"	1' 6"	4' 3"	283.33
627' 6"	635' 0"	7' 6"	10' 0"	133.33
635' 0"	640' 0"	5' 0"	4' 4"	86.66
640' 0"	643' 0"	3' 0"	2' 8"	88.88
643' 0"	646' 0"	3' 0"	3' 0"	100
646' 0"	653' 0"	7' 0"	7' 0"	100

PROJECT: MOINA HOLE NO: ML 1A RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
653' 0"	663' 0"	10' 0"	10' 2"	101.66
663' 0"	668' 0"	5' 0"	5' 0"	100
668' 0"	671' 6"	3' 6"	3' 6"	100
671' 6"	673' 0"	1' 6"	1' 6"	100
673' 0"	680' 0"	5' 0"	8' 5"	168.33
680' 0"	683' 6"	3' 6"	1' 6"	42.85
683' 6"	684' 0"	0' 6"	0'10"	166.66
684' 0"	690' 9"	6' 9"	4' 4"	64.19
690' 9"	697' 0"	6' 3"	7' 0"	112
697' 0"	705' 9"	8' 9"	9' 0"	102.85
705' 9"	713' 9"	8' 0"	7' 4"	91.66
713' 9"	719' 7"	5'10"	5' 2"	88.57
719' 7"	725' 9"	6' 2"	4' 2"	67.56
725' 9"	726' 0"	0' 3"	0' 3"	100
726' 0"	728' 0"	2' 0"	2' 0"	100
728' 0"	730' 0"	2' 0"	1' 7"	79.16
730' 0"	737' 0"	7' 0"	7' 3"	103.57
737' 0"	739' 0"	2' 0"	1' 7"	79.16
739' 0"	744' 0"	5' 0"	1' 4"	26.66
744' 0"	754' 0"	10' 0"	8'10"	88.33
754' 0"	756' 0"	2' 0"	0'11"	45.83
756' 0"	760' 0"	4' 0"	4'11"	122.91
760' 0"	768' 0"	8' 0"	7' 0"	87.50
768' 0"	774' 0"	6' 0"	6' 0"	100
774' 0"	774' 1"	0' 1"	0' 1"	100
774' 1"	774' 3"	0' 2"	0' 1"	50
774' 3"	774' 4"	0' 1"	0' 1"	100
774' 4"	783' 0"	8' 8"	6' 0"	69.23
783' 0"	784' 4"	1' 4"	1' 4"	100
784' 4"	793' 0"	8' 8"	8' 8"	100
793' 0"	804' 4"	11' 4"	10'10"	95.58
804' 0"	811' 0"	7' 0"	5' 2"	73.80

221421

PROJECT: MOINA HOLE NO: ML 2 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0' 0"	18' 0"	18' 0"	4' 6"	25.
18' 0"	30' 0"	12' 0"	6' 8"	55.50
30' 0"	45' 0"	15' 0"	6' 7"	43.88
45' 0"	47' 0"	2' 0"	1' 8"	83.33
47' 0"	60' 2"	12'10"	5' 5"	42.20
60' 2"	62' 0"	1'10"	2' 3"	122.72
62' 0"	68' 0"	6' 0"	5' 8"	94.33
68' 0"	71' 0"	3' 0"	6' 8"	222.22
71' 0"	80' 0"	9' 0"	5'10"	64.81
80' 0"	93' 0"	13' 0"	10' 6"	80.76
93' 0"	96' 0"	3' 0"	3' 7"	119.44
96' 0"	103' 0"	7' 0"	8' 9"	125
103' 0"	105' 0"	2' 0"	2' 1"	104.16
105' 0"	118' 0"	13' 0"	8' 4"	64.10
118' 0"	122' 0"	4' 0"	8'11"	222.91
122' 0"	123' 6"	1' 6"	1' 7"	105.55
123' 6"	133' 0"	9' 6"	7' 1"	74.56
133' 0"	133' 6"	0' 6"	0' 5"	83,32
133' 6"	134' 0"	0' 6"	0' 1"	16.66
134' 0"	139' 6"	5' 6"	3' 3"	59.09
139' 6"	139'10"	0' 4"	0' 3"	75.00
139'10"	140' 4"	0' 6"	0' 6"	100
140' 4"	143' 0"	2' 8"	2' 5"	90.62
143' 0"	150' 4"	7' 4"	4' 2"	56.81
150' 4"	157'10"	7' 3"	4'11"	67.81
157'10"	160' 0"	2' 2"	2' 4"	107.69
160' 0"	163' 0"	3' 0"	6' 4"	211.11
163' 0"	167'10"	4'10"	3' 5"	70.68
167'10"	177'10"	10' 0"	8'11"	89.16
177'10"	182' 0"	4' 2"	3' 2"	57.75
182' 0"	184' 0"	2' 0"	1' 5"	70.83
184' 0"	190' 0"	8' 0"	2' 0"	25
190' 0"	200' 0"	10' 0"	12' 8"	126.66
200' 0"	203' 0"	3' 0"	1' 8"	55.55
203' 0"	210' 0"	7' 0"	5' 3"	75
210' 0"	213' 7"	3' 7"	3' 0"	83.72
213' 7"	223' 6"	9'11"	2' 9"	27.73
223' 6"	230' 0"	6' 6"	6' 4"	97.43
230' 0"	240' 0"	10' 0"	10' 0"	100
240' 0"	241' 0"	1' 0"	0' 6"	50
241' 0"	248' 0"	7' 0"	5'10"	83.33
248' 0"	250' 0"	2' 0"	0' 9"	37.5
250' 0"	260' 0"	10' 0"	9' 0"	90
260' 0"	261' 0"	1' 0"	0' 6"	50
261' 0"	266' 0"	5' 0"	5' 6"	110
266' 0"	275' 0"	9' 0"	9' 2"	101.85
275' 0"	276' 0"	1' 0"	1' 8"	166.66
276' 0"	278' 0"	2' 0"	2' 0"	100
278' 0"	282' 0"	4' 0"	2' 7"	64.58
282' 0"	285' 6"	3' 6"	3' 6"	100
285' 6"	287' 0"	1' 6"	2' 3"	150
287' 0"	294' 0"	7' 0"	7' 4"	104.76
294' 0"	296' 0"	2' 0"	1' 8"	83.33
296' 0"	300' 0"	4' 0"	3' 8"	91.66
300' 0"	304' 0"	4' 0"	3' 8"	91.66
304' 0"	314' 0"	10' 0"	5' 0"	50

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221422

PROJECT: MOINA HOLE NO: ML 2 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
314' 0"	316' 0"	2' 0"	5' 4"	266.66
316' 0"	324' 0"	8' 0"	6' 2"	77.08
324' 0"	330' 0"	6' 0"	3' 0"	50
330' 0"	333' 0"	3' 0"	2' 0"	66.66
333' 0"	335' 0"	2' 0"	0' 5"	20.83
335' 0"	337' 0"	2' 0"	3'10"	191.66
337' 0"	340' 0"	3' 0"	2' 6"	83.33
340' 0"	344' 0"	4' 0"	4' 3"	106.25
344' 0"	345' 0"	1' 0"	1' 7"	158.00
345' 0"	348' 0"	3' 0"	3' 1"	102.77
348' 0"	351' 0"	3' 0"	2' 7"	86.11
351' 0"	357' 0"	6' 0"	4' 8"	77.77
357' 0"	360' 0"	3' 0"	1'10"	61.11
360' 0"	368' 0"	8' 0"	6' 4"	79.16
368' 0"	400' 0"	Box lost		
400' 0"	408' 0"	8' 0"	7'10"	97.91
408' 0"	414' 6"	6' 6"	5' 6"	84.61
414' 6"	417' 0"	2' 6"	3' 0"	120
417' 0"	419' 6"	2' 6"	2' 6"	100
419' 6"	425' 6"	6' 0"	7' 0"	116.66
425' 6"	432' 6"	7' 0"	7' 0"	100
432' 6"	435' 0"	2' 6"	1' 4"	53.33
435' 0"	437' 0"	2' 0"	3' 0"	150
437' 0"	441' 0"	4' 0"	3' 6"	87.5
441' 0"	442' 0"	1' 0"	0' 9"	75.0
442' 0"	446' 6"	4' 6"	4' 1"	90.74
446' 6"	450' 0"	3' 6"	2' 8"	76.18
450' 0"	451' 0"	1' 0"	0' 9"	75.00
451' 0"	454' 6"	3' 6"	3' 6"	100
454' 6"	463' 6"	9' 0"	7' 2"	79.62
463' 6"	467' 0"	4' 0"	3' 0"	75
467' 0"	468' 0"	1' 0"	1' 0"	100
468' 0"	469' 6"	1' 6"	1' 2"	77.77
469' 6"	473' 0"	3' 6"	8' 6"	242.85
473' 0"	481' 6"	8' 6"	4' 3"	50
481' 6"	486' 0"	4' 6"	4' 3"	94.44
486' 0"	491' 6"	5' 6"	5' 2"	93.93
491' 6"	493' 0"	1' 6"	1' 4"	88 86
493' 0"	497' 0"	4' 0"	4'10"	120.83
497' 0"	500' 6"	3' 6"	4'10"	138.09
500' 6"	501' 0"	0' 6"	0' 6"	100
501' 0"	504' 6"	3' 6"	3' 5"	97.61
504' 6"	511' 0"	6' 6"	5' 9"	88.46
511' 0"	517' 0"	6' 0"	5' 9"	95.83
517' 0"	518' 0"	1' 0"	2' 0"	200
518' 0"	536' 6"	core lost		
536' 6"	543' 6"	7' 0"	8' 6"	121.42
543' 6"	545' 6"	2' 0"	1' 4"	66.66
545' 6"	554' 0"	9' 6"	3' 6"	36.84
554' 0"	558' 0"	4' 0"	3' 4"	83.33
558' 0"	563' 0"	5' 0"	5' 0"	100
563' 0"	564' 0"	1' 0"	0'11"	91.66
564' 0"	569' 0"	5' 0"	5' 4"	106.66
569' 0"	572' 0"	3' 0"	1' 0"	33.33
572' 0"	574' 6"	2' 6"	2' 7"	103.33

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221423

PROJECT: MOINA HOLE NO: ML 2 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
574' 6"	577' 0"	2' 6"	1'10"	73.33
577' 0"	581' 6"	4' 6"	4' 5"	98.14
581' 6"	588' 6"	7' 0"	8' 0"	114.28
588' 6"	590' 6"	2' 0"	2' 0"	100
590' 6"	594' 0"	3' 6"	3' 6"	100
594' 0"	596' 6"	2' 6"	2' 6"	100
596' 6"	600' 0"	3' 6"	2' 9"	78.57
600' 0"	605' 0"	5' 0"	5' 5"	108.33
605' 0"	612' 0"	7' 0"	6' 0"	85.71
612' 0"	613' 0"	1' 0"	1' 0"	100
613' 0"	618' 0"	5' 0"	5' 3"	105
618' 0"	623' 0"	5' 0"	6' 0"	120
623' 0"	626' 6"	3' 6"	4' 6"	128.57
626' 6"	628' 0"	1' 6"	1' 6"	100
628' 0"	634' 0"	6' 0"	7' 3"	120.83
634' 0"	639' 0"	5' 0"	4' 2"	104.16
639' 0"	644' 0"	5' 0"	5' 6"	110
644' 0"	645' 0"	1' 0"	1' 0"	100
645' 0"	649' 0"	4' 0"	3' 7"	89.58
649' 0"	654' 0"	5' 0"	4' 5"	88.33
654' 0"	659' 0"	5' 0"	5'10"	116.66
659' 0"	660' 0"	1' 0"	0' 7"	58.33
660' 0"	663' 0"	3' 0"	2' 4"	77.77
663' 0"	665' 0"	2' 0"	2' 1"	104.16
665' 0"	668' 0"	3' 0"	3' 8"	122.22
668' 0"	672' 0"	4' 0"	4' 5"	110.41
672' 0"	675' 0"	3' 0"	2' 2"	72.22
675' 0"	676' 0"	1' 0"	1' 0"	100
676' 0"	680' 0"	4' 0"	3' 0"	75
680' 0"	681' 0"	1' 0"	0' 9"	75
681' 0"	683' 6"	2' 6"	2' 8"	106.66
683' 6"	684' 0"	0' 6"	0' 4"	66.66
684' 0"	685' 6"	1' 6"	1' 7"	105.55
685' 6"	689' 6"	4' 0"	3' 8"	91.66
689' 6"	690' 0"	0' 6"	0' 6"	100
690' 0"	692' 0"	2' 0"	1' 9"	87.5
692' 0"	694' 6"	2' 6"	1'10"	73.33
694' 6"	696' 6"	2' 6"	2' 8"	106.66
696' 6"	697' 6"	1' 0"	1' 5"	141.66
697' 6"	700' 0"	2' 6"	2' 1"	83.33
700' 0"	701' 0"	1' 0"	0' 9"	75
701' 0"	703' 0"	2' 0"	2' 6"	125
703' 0"	706' 6"	3' 6"	3' 8"	104.76
706' 6"	709' 6"	3' 0"	3' 3"	108.33
709' 6"	711' 6"	2' 0"	2' 2"	108.33
711' 6"	716' 0"	4' 6"	4' 5"	98.14
716' 0"	717' 6"	1' 6"	1' 9"	116.66
717' 6"	721' 6"	4' 0"	3' 3"	81.25
721' 6"	724' 6"	3' 0"	3' 2"	105.55
724' 6"	729' 6"	5' 6"	5' 0"	90.90
729' 6"	730' 0"	0' 6"	0' 6"	100
730' 0"	734' 0"	4' 0"	3' 1"	77.08
734' 0"	738' 0"	4' 0"	4' 3"	106.25
738' 0"	742' 0"	4' 0"	4' 5"	110.41

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221424

PROJECT: MOINA HOLE NO: ML 2 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
742' 0"	748' 0"	6' 0"	6' 1"	101.38
748' 0"	751' 0"	3' 0"	3' 0"	100
751' 0"	751' 6"	0' 6"	0' 6"	100
751' 6"	754' 0"	2' 6"	3' 3"	130
754' 0"	756' 6"	2' 6"	1'10"	73.33
756' 6"	763' 0"	6' 6"	6' 9"	103.84
763' 0"	767' 0"	4' 0"	4' 3"	106.25
767' 0"	790' 0"	23' 0"	4' 6"	19.56
790' 0"	794' 6"	4' 6"	4' 4"	96.29
794' 6"	797' 0"	2' 6"	3' 0"	120
797' 0"	803' 0"	6' 0"	3'10"	63.88
803' 0"	804' 6"	1' 6"	1' 6"	100
804' 6"	806' 6"	2' 0"	2' 0"	100
806' 6"	807' 6"	1' 0"	0' 9"	75
807' 6"	809' 0"	1' 6"	1' 8"	111.10
809' 0"	813' 0"	4' 0"	3' 8"	91.66
813' 0"	816' 6"	3' 6"	3' 5"	95.23
816' 6"	819' 0"	2' 6"	2' 6"	100
819' 0"	822' 0"	3' 0"	2' 6"	83.33
822' 0"	824' 0"	2' 0"	1' 3"	62.5
824' 0"	829' 0"	5' 0"	5' 0"	100
829' 0"	835' 0"	6' 0"	7' 6"	125
835' 0"	837' 6"	2' 6"	3' 0"	120
837' 6"	841' 0"	3' 6"	4' 0"	114.28
841' 0"	843' 6"	2' 6"	4' 0"	160
843' 6"	848' 6"	5' 0"	4' 0"	80
848' 6"	850' 6"	2' 0"	2'10"	141.66
850' 6"	852' 0"	1' 6"	2' 0"	133.33
852' 0"	856' 0"	4' 0"	4' 0"	100
856' 0"	862' 0"	6' 0"	11' 2"	186.11
862' 0"	871' 0"	9' 0"	8' 8"	96.29
871' 0"	884' 0"	13' 0"	8' 0"	61.15
884' 0"	891' 0"	7' 0"	7' 9"	110.71
891' 0"	893' 0"	2' 0"	3' 0"	150
893' 0"	897' 0"	4' 0"	5' 0"	125
897' 0"	902' 6"	5' 6"	4'10"	87.87
902' 6"	910' 6"	8' 0"	8' 7"	107.29
910' 6"	913' 0"	2' 6"	2' 7"	103.33
913' 0"	919' 0"	6' 0"	5'10"	97.22
919' 0"	925' 6"	6' 6"	6' 9"	103.84
925' 6"	930' 6"	5' 0"	4'10"	96.66
930' 6"	935' 6"	5' 0"	4'10"	96.66
935' 6"	938'00"	2' 6"	2' 4"	93.33
938' 0"	940' 0"	2' 0"	1'10"	91.66

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221425

PROJECT: MOINA HOLE NO: ML 2 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
940' 0"	944' 0"	4' 0"	4' 0"	100
944' 0"	946' 0"	2' 0"	2' 0"	100
946' 0"	951' 6"	5' 6"	5' 8"	103.02
951' 6"	953' 6"	2' 0"	2' 0"	100
953' 6"	961' 6"	8' 0"	8' 0"	100
961' 6"	965' 6"	4' 0"	4' 4"	108.33
965' 6"	971' 6"	6' 0"	5' 4"	88.88
971' 6"	975' 6"	4' 0"	3' 7"	89.58
975' 6"	979' 6"	4' 0"	3' 4"	83.32
979' 6"	984' 6"	5' 0"	5' 8"	113.33
984' 6"	986' 6"	2' 0"	2' 2"	108.33
986' 6"	989' 0"	2' 6"	2' 5"	96.66
989' 0"	992' 6"	3' 6"	2' 6"	71.42
992' 6"	995' 0"	2' 6"	4' 4"	173.33
995' 0"	1002' 6"	7' 6"	8' 5"	112.22
1002' 6"	1006' 0"	3' 6"	3' 3"	92.85
1006' 0"	1008' 6"	2' 6"	2' 2"	86.66
1008' 6"	1010' 6"	2' 6"	2' 6"	100
1010' 6"	1014' 0"	3' 6"	3' 7"	102.38
1014' 0"	1017' 6"	3' 6"	2' 7"	73.80
1017' 6"	1024' 0"	6' 6"	9' 1"	139.74
1024' 0"	1026' 0"	2' 0"	2' 2"	108.33
1026' 0"	1031' 6"	5' 6"	7' 1"	128.78
1031' 6"	1033' 0"	1' 6"	1' 1"	72.22
1033' 0"	1040' 0"	7' 0"	7' 7"	108.33
1040' 0"	1046' 6"	2' 6"	2' 9"	110
1046' 6"	1049' 0"	2' 6"	3' 0"	120
1049' 0"	1054' 0"	5' 0"	5' 0"	100
1054' 0"	1055' 0"	1' 0"	1' 3"	125
1055' 0"	1058' 0"	3' 0"	4' 0"	133.33
1058' 0"	1100' 0"	42' 0"	14' 0"	core lost partly jumbled

332

221426

PROJECT: MOINA HOLE NO: ML 3 A RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0' 0"	20' 0"	20' 0"	6' 2"	30.83
20' 0"	30' 0"	10' 0"	4' 3"	42.50
30' 0"	35' 0"	5' 0"	3' 5"	68.33
35' 0"	40' 0"	5' 0"	3' 6"	70
40' 0"	57' 0"	17' 0"	5' 1"	29.90
57' 0"	63' 6"	6' 6"	6' 8"	102.56
63' 6"	68' 6"	5' 0"	5' 7"	111.66
68' 6"	72' 6"	2' 0"	4' 8"	233.33
72' 6"	76' 6"	4' 0"	4' 2"	104.16
76' 6"	79' 0"	2' 6"	3' 4"	133.32
79' 0"	79' 6"	0' 6"	2' 3"	450.00
79' 6"	83' 0"	3' 6"	4' 6"	128.57
83' 0"	84' 0"	1' 0"	1' 2"	116.66
84' 0"	89' 0"	5' 0"	4' 9"	95
89' 0"	94' 0"	5' 0"	4' 3"	85
94' 0"	99' 0"	5' 0"	5' 7"	111.66
99' 0"	101' 0"	2' 0"	1'10"	91.66
101' 0"	109' 0"	8' 0"	8' 8"	108.33
109' 0"	114' 6"	5' 6"	4' 8"	84.84
114' 6"	120' 6"	6' 0"	7' 7"	126.38
120' 6"	124' 0"	3' 6"	4' 3"	121.42
124' 0"	127' 0"	3' 0"	3' 5"	113.88
127' 0"	129' 0"	2' 0"	2' 8"	133.33
129' 0"	131' 0"	2' 0"	2' 7"	129.16
131' 0"	133' 0"	2' 0"	1' 6"	75
133' 0"	137' 0"	4' 0"	4' 2"	104.16
137' 0"	140' 0"	3' 0"	2' 9"	91.66
140' 0"	146' 6"	6' 6"	6'10"	105.12
146' 6"	153' 0"	6' 3"	6' 9"	108
153' 0"	160' 0"	7' 0"	6' 2"	88.09
160' 0"	168' 0"	8' 0"	9' 6"	118.75
168' 0"	171' 0"	3' 0"	2' 9"	91.66
171' 0"	177' 0"	6' 0"	5' 4"	88.88
177' 0"	179' 0"	2' 0"	2'11"	145.83
179' 0"	185' 0"	6' 0"	5' 4"	88.88
185' 0"	186' 0"	1' 0"	4' 5"	441.66
186' 0"	190' 0"	4' 0"	4' 1"	102.08
190' 0"	193' 0"	3' 0"	2' 5"	80.55
193' 0"	201' 0"	8' 0"	1' 1"	13:54
201' 0"	206' 0"	5' 0"	5' 0"	100
206' 0"	212' 0"	6' 0"	6'10"	113.88
212' 0"	218' 0"	6' 0"	6' 7"	109.72
218' 0"	223' 0"	5' 0"	5' 9"	115
223' 0"	233' 0"	10' 0"	9'10"	98.33
233' 0"	243' 0"	10' 0"	10' 0"	100
243' 0"	252' 0"	9' 0"	5' 4"	59.25
252' 0"	254' 0"	2' 0"	1' 4"	66.65
254' 0"	256' 0"	2' 0"	3'10"	191.66
256' 0"	263' 0"	7' 0"	9'10"	140.47
263' 0"	273' 0"	10' 0"	6' 4"	63.33
273' 0"	275' 0"	2' 0"	3' 4"	166.65
275' 0"	285' 0"	10' 0"	4' 0"	40
285' 0"	291' 0"	6' 0"	2' 6"	41.66
291' 0"	293' 0"	2' 0"	7' 3"	362.50

333

221427

PROJECT: MOINA HOLE NO: ML 3 A RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
293' 0"	294' 6"	1' 6"	2' 6"	66.66
294' 0"	296' 0"	1' 6"	1'10"	122.22
296' 0"	299' 0"	3' 0"	2' 4"	77.76
299' 0"	310' 0"	11' 0"	2' 4"	21.20
310' 0"	311' 0"	1' 0"	1' 5"	141.66
311' 0"	316' 0"	5' 0"	4' 2"	83.33
316' 0"	319' 0"	3' 0"	1' 7"	52.77
319' 0"	338' 0"			
338' 0"	340' 0"	2' 0"	2' 0"	100
340' 0"	346' 0"	6' 0"	3' 0"	50
346' 0"	352' 6"	6' 6"	6' 3"	96.15
352' 6"	359' 0"	6' 6"	6' 7"	101.28
359' 0"	362' 0"	3' 0"	2' 6"	83.33
362' 0"	371' 0"	9' 0"	7' 8"	85.17
371' 0"	377' 0"	6' 0"	6' 0"	100
377' 0"	385' 0"	8' 0"	8' 6"	106.25
385' 0"	393' 0"	8' 0"	10' 0"	125
393' 0"	397' 0"	4' 0"	4' 6"	112.50
397' 0"	402' 0"	5' 0"	3' 9"	75
402' 0"	404' 0"	2' 0"	1' 3"	62.50
404' 0"	408' 0"	4' 0"	3'10"	95.83
408' 0"	409' 0"	1' 0"	1' 7"	158.33
409' 0"	412' 0"	3' 0"	2' 1"	69.44
412' 0"	414' 0"	2' 0"	2' 2"	108.33
414' 0"	421' 0"	7' 0"	5' 0"	71.42
421' 0"	424' 0"	3' 0"	3' 9"	125
424' 0"	426' 0"	2' 0"	1' 3"	65
426' 0"	429' 0"	7' 0"	4' 2"	59.52
429' 0"	432' 0"	3' 0"	3' 1"	102.77
432' 0"	435' 0"	3' 0"	3'10"	127.77
435' 0"	438' 0"	3' 0"	4' 0"	133.33
438' 0"	442' 0"	4' 0"	3' 5"	85.41
442' 0"	445' 0"	3' 0"	3' 8"	122.2
445' 0"	450' 0"	5' 0"	5' 0"	100
450' 0"	454' 6"	4' 6"	4' 7"	101.85
454' 6"	457' 0"	2' 6"	3' 3"	130
457' 0"	460' 0"	3' 0"	2' 9"	91.66
460' 0"	462' 0"	2' 0"	1' 6"	75
462' 0"	471' 6"	9' 6"	9. 6"	100
471' 6"	476' 0"	5' 6"	4'11"	89.39
476' 0"	478' 0"	2' 0"	2' 2"	108.33
478' 0"	480' 0"	2' 0"	1' 6"	75
480' 0"	482' 0"	2' 0"	2' 0"	100
482' 0"	484' 0"	2' 0"	2' 0"	100
484' 0"	491' 0"	7' 0"	6' 8"	95.23
491' 0"	497' 0"	6' 0"	3' 1"	51.38
497' 0"	504' 0"	7' 0"	8' 0"	114.28
504' 0"	505' 0"	1' 0"	1' 0"	100
505' 0"	510' 0"	5' 0"	8' 0"	160
510' 0"	513' 0"	3' 0"	8' 1"	269.44
513' 0"	516' 0"	3' 0"	2' 3"	75
516' 0"	521' 0"	5' 0"	4' 0"	80
521' 0"	523' 0"	2' 0"	2' 7"	129.16
523' 0"	528' 0"	5' 0"	5' 7"	111.66

221428

PROJECT: MOINA HOLE NO: ML 3 A RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
528' 0"	534' 0"	6' 0"		
534' 0"	546' 0"	12' 0"	10' 7"	88.19
546' 0"	548' 0"	2' 0"	2' 3"	112.5
548' 0"	552' 0"	4' 0"	2' 9"	68.75
552' 0"	562' 0"	10' 0"	6' 2"	61.66
562' 0"	572' 0"	10' 0"	8' 5"	84.16
572' 0"	580' 0"	8' 0"	6' 5"	80.20
580' 0"	581' 0"	1' 0"	1' 3"	125
581' 0"	585' 0"	5' 0"	4' 2"	83.33
585' 0"	594' 0"	9' 0"	8' 7"	95.37
594' 0"	603' 0"	9' 0"	9' 4"	103.70
603' 0"	607' 0"	4' 0"	4' 2"	104.16
607' 0"	614' 0"	7' 0"	4' 3"	60.71
614' 0"	619' 0"	5' 0"	5' 9"	115.00
619' 0"	623' 0"	4' 0"	3' 6"	42.85
623' 0"	625' 0"	2' 0"	2' 8"	133.30
625' 0"	627' 0"	2' 0"	3' 5"	170.83
627' 0"	633' 0"	6' 0"	4' 8"	77.77
633' 0"	641' 0"	8' 0"	8'10"	110.41
641' 0"	645' 0"	4' 0"	5' 2"	129.16
645' 0"	647' 0"	2' 0"	2'10"	141.66
647' 0"	650' 0"	3' 0"	3' 5"	113.88
650' 0"	656' 0"	6' 0"	6' 8"	111.11
656' 0"	663' 0"	7' 0"	5' 6"	78.57
663' 0"	667' 0"	4' 0"	5' 6"	137.5
667' 0"	670' 0"	3' 0"	3' 9"	125
670' 0"	674' 0"	4' 0"	3'10"	95.83
674' 0"	678' 0"	4' 0"	4' 4"	108.32
678' 0"	681' 0"	3' 0"	2'11"	97.22
681' 0"	684' 0"	3' 0"	5' 4"	177.77
684' 0"	687' 0"	3' 0"	3' 9"	125
687' 0"	690' 0"	3' 0"	2' 2"	72.22
690' 0"	695' 0"	5' 0"	6' 8"	133.33
695' 0"	697' 0"	2' 0"	3'10"	191.66
697' 0"	700' 0"	3' 0"	4'10"	161.11
700' 0"	703' 0"	3' 0"	3' 9"	125
703' 0"	711' 0"	8' 0"	5' 5"	677.70
711' 0"	716' 0"	5' 0"	4'11"	98.33
716' 0"	717' 0"	1' 0"	1'11"	191.66
717' 0"	723' 0"	6' 0"	3' 3"	54.16
723' 0"	734' 0"	11' 0"	2'10"	25.75
734' 0"	738' 0"	4' 0"	4' 2"	104.16
738' 0"	745' 0"	7' 0"	6'11"	98.80
745' 0"	747' 0"	2' 0"	2' 8"	133
747' 0"	753' 0"	6' 0"	4'10"	80.55
753' 0"	757' 0"	4' 0"	5' 4"	133.32
757' 0"	760' 0"	3' 0"	6' 3"	208.33
760' 0"	763' 0"	3' 0"	2' 9"	91.66
763' 0"	767' 0"	4' 0"	4' 9"	118.75
767' 0"	769' 0"	2' 0"	4' 5"	220.83
769' 0"	770' 0"	1' 0"	1' 8"	166.66
770' 0"	771' 0"	1' 0"	2' 1"	208.33
771' 0"	780' 0"	9' 0"	8' 3"	91.66
780' 0"	782' 0"	2' 0"	2' 4"	116.65

335

221429

PROJECT: MOINA HOLE NO: ML 3 A RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
782' 0"	784' 0"	2' 0"	2' 7"	129.16
784' 0"	787' 0"	3' 0"	4' 8"	155.55
787' 0"	792' 0"	5' 0"	5' 3"	105
792' 0"	796' 0"	4' 0"	7'11"	197.91
796' 0"	801' 6"	5' 6"	5'11"	107.57
801' 6"	806' 0"	4' 6"	7'10"	174.07
806' 0"	809' 0"	3' 0"	4'10"	161.11
809' 0"	812' 0"	3' 0"	2' 3"	75
812' 0"	818' 6"	6' 6"	7' 6"	115.38
818' 6"	823' 0"	4' 6"	5' 4"	118.51
823' 0"	827' 0"	4' 0"	5' 8"	141.66
827' 0"	830' 0"	3' 0"	2'11"	97.22
830' 0"	834' 0"	4' 0"	3' 6"	87.5
834' 0"	836' 0"	2' 0"	2' 4"	116.66
836' 0"	842' 0"	6' 0"	4' 0"	66.66
842' 0"	845' 0"	3' 0"	4' 1"	136.11
845' 0"	847' 0"	2' 0"	1' 2"	58.33
847' 0"	851' 0"	4' 0"	3'10"	95.83
851' 0"	854' 0"	3' 0"	5' 1"	169.44

336

221430

PROJECT: MOINA HOLE NO: SMD 4 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0	8.00	8.00	4.30	53.7
8.00	10.00	2.00	1.75	87.5
10.00	15.23	5.25	5.32	101.7
15.23	17.26	2.03	2.03	100
17.26	19.60	2.34	2.34	100
19.60	20.70	1.10	1.05	95.4
20.70	22.50	1.80	1.90	105.5
22.50	23.23	0.73	0.60	82.1
23.23	23.70	0.47	0.50	106.3
23.70	24.68	0.98	0.40	40.8
24.68	27.00	2.32	1.50	64.6
27.00	30.00	3.00	2.90	96.6
30.00	30.56	0.56	0.32	57.1
30.56	34.31	3.75	0.50	13.3
34.31	35.85	1.54	0.90	58.4
35.85	37.00	1.15	0.75	65.2
37.00	37.50	0.50	0.40	80
37.50	40.00	2.50	2.50	100
40.00	43.53	3.53	3.45	97.7
43.53	45.84	2.31	0.95	41.1
45.84	46.00	0.16	0.70	437.5
46.00	46.77	0.77	0.50	64.9
46.77	47.00	0.23	0.40	173.9
47.00	50.00	3.00	2.00	66.6
50.00	52.55	2.55	2.50	98
52.55	54.53	1.98	1.80	90.9
54.53	55.55	1.02	0.55	53.9
55.55	56.00	0.45	1.00	222.2
56.00	59.00	3.00	3.00	100
59.00	59.93	0.93	0.90	96.7
59.93	62.00	2.07	1.75	94.2
62.00	65.00	3.00	3.00	100
65.00	66.90	1.90	1.55	81.5
66.90	68.00	1.10	1.30	118.1
68.00	68.37	0.37	0.28	75.6
68.37	71.00	2.63	2.72	103.4
71.00	74.00	3.00	1.98	66
74.00	77.00	3.00	2.94	98
77.00	77.27	0.27	0.25	92.5
77.27	80.00	2.73	2.75	100.7
80.00	81.36	1.36	1.34	98.5
81.36	81.66	0.30	0.32	106.6
81.66	83.00	1.34	0.88	65.6
83.00	86.00	3.00	2.55	85
86.00	89.00	3.00	3.00	100
89.00	91.25	2.25	2.70	120
91.25	92.00	0.75	0.58	77.3
92.00	95.00	3.00	3.10	103.3
95.00	97.75	2.75	2.75	100
97.75	100.88	3.13	3.20	102.2
100.88	103.00	2.12	3.25	153.3
103.00	105.25	2.25	2.65	117.7
105.25	108.25	3.00	2.90	96.6

337

221431

PROJECT: MOINA HOLE NO: SMD 5 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0	9.00	9.00	1.65	18.3
9.00	12.00	3.00	1.30	43.3
12.00	14.00	2.00	2.35	100
14.00	16.00	2.00	1.60	80
16.00	17.00	1.00	0.75	75
17.00	19.00	2.00	1.55	77.5
19.00	20.00	1.00	0.80	80
20.00	22.25	2.25	1.40	62.2
22.25	23.00	0.75	0.90	120
23.00	24.29	1.29	0.85	65.8
24.29	26.00	1.71	1.72	105
26.00	27.33	1.33	1.30	97.7
27.33	29.00	1.67	1.78	106.5
29.00	32.75	3.75	2.80	74.6
32.75	36.00	3.25	2.63	80.9
36.00	39.00	3.00	2.80	93
39.00	42.00	3.00	3.00	100
42.00	45.00	3.00	3.00	100
45.00	47.25	2.25	2.20	97.7
47.25	50.50	3.25	3.18	97.8
50.50	51.00	0.50	0.50	100
51.00	53.85	2.85	2.70	94.7
53.85	57.00	3.15	3.12	99
57.00	60.00	3.00	3.05	101.6
60.00	63.00	3.00	2.87	95.6
63.00	66.00	3.00	3.35	111.6
66.00	68.00	2.00	2.20	110
68.00	69.00	1.00	1.15	115
69.00	72.00	3.00	3.00	100
72.00	75.00	3.00	3.00	100
75.00	78.00	3.00	3.05	101.6
78.00	80.79	2.79	2.45	87.8
80.79	81.15	0.36	1.25	347.2

221432

PROJECT: MOINA HOLE NO: SMD6 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0	5.50	5.50	0.50	90.9
5.50	7.00	1.50	1.60	106.6
7.00	8.75	1.75	1.30	74.2
8.75	10.25	1.50	1.32	88
10.25	12.25	2.00	1.85	92.5
12.25	14.00	1.75	1.70	97.14
14.00	17.00	3.00	3.00	100
17.00	20.00	3.00	2.85	95
20.00	23.00	3.00	2.25	75
23.00	24.50	1.50	1.15	76.6
24.50	26.00	1.50	1.65	110
26.00	29.00	3.00	2.55	75
29.00	30.00	1.00	1.15	115
30.00	31.55	1.55	1.81	116.7
31.55	35.00	3.45	3.15	91.3
35.00	38.00	3.00	3.00	100
38.00	41.00	3.00	3.00	100
41.00	44.00	3.00	3.00	100
44.00	47.00	3.00	3.00	100
47.00	50.00	3.00	3.00	100
50.00	51.50	1.50	1.50	100
51.50	53.00	1.50	1.55	103
53.00	56.00	3.00	3.00	100
56.00	59.00	3.00	3.00	100
59.00	62.00	3.00	3.00	100
62.00	65.00	3.00	2.90	96.6
65.00	68.00	3.00	3.00	100
68.00	71.00	3.00	3.00	100
71.00	74.00	3.00	3.00	100
74.00	77.00	3.00	3.05	101
77.00	80.00	3.00	3.00	100
80.00	83.00	3.00	3.00	100
83.00	86.00	3.00	2.85	95
86.00	87.00	1.00	0.92	92
87.00	89.00	2.00	2.30	115
89.00	92.00	3.00	3.10	103.3
92.00	95.00	3.00	2.90	96.6
95.00	98.00	3.00	2.85	95
98.00	99.50	1.50	1.30	86.6
99.50	102.50	3.00	3.10	103.3

221433

PROJECT: MOINA HOLE NO: SMD 7 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0	1.00	1.00	1.00	100
1.00	8.00	7.00	0.95	13.5
8.00	9.62	1.62	0.60	37.0
9.62	11.00	1.38	0.75	54.3
11.00	13.00	2.00	1.90	95
13.00	14.00	1.00	1.00	100
14.00	17.00	3.00	2.05	68.3
17.00	20.00	3.00	3.00	100
20.00	23.00	3.00	2.95	98.3
23.00	26.00	3.00	3.05	101
26.00	28.50	2.50	2.45	98
28.50	31.50	3.00	3.00	100
31.50	35.00	3.50	3.40	97.1
35.00	38.00	3.00	2.95	98.3
38.00	41.00	3.00	2.90	96.6
41.00	44.00	3.00	3.00	100
44.00	47.00	3.00	2.92	97.3
47.00	50.00	3.00	3.00	100
50.00	53.00	3.00	3.00	100
53.00	56.00	3.00	2.98	99.3
56.00	59.00	3.00	2.95	98.3
59.00	62.00	3.00	3.00	100
62.00	65.00	3.00	3.00	100
65.00	67.00	2.00	2.20	110
67.00	70.00	3.00	3.00	100
70.00	71.50	1.50	1.25	83.3

340

221434

PROJECT: MOINA HOLE NO: SMD 8 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0	3.00	3.00	1.95	65
3.00	9.00	6.00	4.80	80
9.00	11.50	2.50	2.00	80
11.50	14.00	2.50	2.80	112
14.00	17.00	3.00	2.90	96.6
17.00	26.00	9.00	3.80	42.2
26.00	27.00	1.00	0.10	10
27.00	28.00	1.00	0.50	50
28.00	30.00	2.00	2.00	100
30.00	31.00	1.00	1.00	100
31.00	32.50	1.50	0.35	23.3
32.50	34.00	1.50	0.95	63.3
34.00	37.00	3.00	3.00	100
37.00	40.00	3.00	2.25	75
40.00	42.50	2.50	1.75	70
42.50	44.00	1.50	1.70	113
44.00	46.00	2.00	1.10	55
46.00	48.00	2.00	2.20	110
48.00	49.50	1.50	1.35	90
49.50	52.00	2.50	2.35	94
52.00	53.00	1.00	1.00	100
53.00	55.00	2.00	2.00	100
55.00	60.80	5.80	5.80	100

341

221435

PROJECT: MOINA HOLE NO:SMD 9 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0	12.00	12.00	1.25	10.4
12.00	17.00	5.00	5.30	106
17.00	20.00	3.00	2.85	95
20.00	23.00	3.00	2.90	96.6
23.00	26.00	3.00	3.05	101
26.00	29.00	3.00	3.00	100
29.00	35.00	6.00	5.80	96.6
35.00	38.00	3.00	3.00	100
38.00	41.00	3.00	3.00	100
41.00	44.00	3.00	2.95	98.3
44.00	47.00	3.00	2.95	98.3
47.00	50.00	3.00	3.00	100
50.00	53.00	3.00	3.00	100
53.00	56.00	3.00	2.95	98.3
56.00	59.00	3.00	2.95	98.3
59.00	62.00	3.00	3.00	100
62.00	65.00	3.00	3.00	100
65.00	68.00	3.00	2.95	98.3
68.00	71.00	3.00	3.00	100
71.00	74.00	3.00	2.85	98.3
74.00	76.50	2.50	2.40	96
76.50	79.50	3.00	3.05	101.6
79.50	82.50	3.00	3.00	100
82.50	85.50	3.00	2.95	98.3
85.50	88.50	3.00	3.00	100
88.50	91.50	3.00	3.00	100
91.50	92.50	1.00	1.20	120
92.50	95.00	2.50	1.95	78
95.00	98.00	3.00	2.90	96.6
98.00	101.00	3.00	3.00	100
101.00	104.00	3.00	2.70	90
104.00	107.00	3.00	3.00	100
107.00	109.00	2.00	2.20	110
109.00	110.00	1.00	1.00	100
110.00	113.00	3.00	3.00	100
113.00	116.00	3.00	3.00	100
116.00	122.00	6.00	4.20	70
122.00	122.50	0.50	0.32	64
122.50	123.00	0.50	0.45	90
123.00	125.00	2.00	1.50	95
125.00	128.00	3.00	3.00	100
128.00	129.00	1.00	1.00	100

342

221436

PROJECT: MOINA HOLE NO: SMD 10 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0	6.00	6.00	0.25	4
6.00	6.50	0.50	0.40	80
6.50	7.00	0.50	0.40	80
7.00	7.50	0.50	0.25	50
7.50	8.00	0.50	0.25	50
8.00	8.50	0.50	0.35	70
8.50	9.50	1.00	0.45	45
9.50	11.20	1.70	0.95	56
11.20	12.50	1.30	1.00	77
12.50	13.50	1.00	1.00	100
13.50	15.50	2.00	2.00	100
15.50	17.25	1.75	1.80	103
17.25	19.40	2.15	2.10	98
19.40	20.00	0.60	0.60	100
20.00	23.00	3.00	3.00	100
23.00	26.00	3.00	3.00	100
26.00	27.00	1.00	1.00	100
27.00	29.00	2.00	0.95	48
29.00	32.00	3.00	2.70	90
32.00	34.00	2.00	1.20	60
34.00	34.50	0.50	0.40	80
34.50	36.50	core lost		
36.50	37.00	0.50	0.35	70
37.00	38.75	1.75	1.10	63
38.75	41.00	2.25	1.20	53
41.00	45.00	4.00	3.92	98
45.00	47.25	2.25	2.18	97
47.25	50.25	3.00	3.00	100
50.25	53.00	2.75	2.75	100
53.00	56.00	3.00	3.20	107
56.00	59.00	3.00	3.00	100
59.00	60.25	1.25	1.15	92
60.25	62.00	1.75	1.75	100
62.00	65.00	3.00	2.92	97
65.00	68.00	3.00	3.00	100
68.00	71.00	3.00	3.00	100
71.00	73.50	2.50	2.45	98
73.50	76.50	3.00	3.00	100
76.50	79.00	2.50	2.40	96
79.00	80.50	1.50	0.95	63
80.50	83.00	2.50	2.50	100
83.00	85.00	2.00	2.00	100
85.00	88.50	3.50	3.50	100
88.50	91.25	2.75	2.75	100
91.25	93.00	1.75	1.50	86
93.00	94.00	1.00	1.00	100
94.00	95.00	1.00	1.00	100
95.04	97.00	2.00	1.85	93
97.00	98.00	1.00	0.85	85
98.00	99.00	1.00	0.90	90
99.00	101.00	2.00	2.00	100
101.00	102.00	1.00	1.00	100
102.00	103.00	1.00	1.00	100
103.00	104.00	1.00	0.85	85
104.00	105.00	1.00	0.95	95

221437

PROJECT: MOINA HOLE NO: SMD 10 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
105.00	107.00	2.00	1.80	90
107.00	108.25	1.25	1.20	96
108.25	109.75	1.50	1.25	83
109.75	111.00	1.25	1.15	92
111.00	112.00	1.00	0.90	90
112.00	113.00	1.00	0.95	95
113.00	114.00	1.00	0.95	95
114.00	116.75	2.75	2.70	98

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221438

PROJECT: MOINA HOLE NO: SMD 11 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0	6.00	6.00	0.00	0
6.00	7.00	1.00	0.35	35
7.00	8.00	1.00	0.33	33
8.00	9.00	1.00	0.25	25
9.00	9.50	0.50	0.26	52
9.50	10.25	0.75	0.36	48
10.25	11.25	1.00	0.18	18
11.25	12.00	0.75	0.27	36
12.00	14.25	2.25	0.40	18
14.25	15.00	0.75	0.55	73
15.00	17.25	2.25	0.14	6
17.25	20.25	3.00	1.96	65
20.25	21.50	1.25	1.22	98
21.50	23.25	1.75	1.40	80
23.25	26.25	3.00	2.42	81
26.25	29.50	3.25	0.90	28
29.50	30.00	0.50	0.50	100
30.00	30.50	0.50	0.50	100
30.50	31.00	0.50	0.50	100
31.00	32.20	1.20	0.64	53
32.20	35.20	3.00	2.94	98
35.20	38.20	3.00	2.75	92
38.20	41.20	3.00	3.00	100
41.20	44.20	3.00	3.00	100
44.20	47.20	3.00	2.95	98
47.20	50.20	3.00	2.96	98
50.20	53.20	3.00	3.00	100
53.20	56.20	3.00	3.00	100
56.20	59.20	3.00	2.98	99
59.20	62.20	3.00	3.00	100
62.20	65.20	3.00	3.00	100
65.20	68.00	2.80	2.68	96
68.00	69.50	1.50	1.50	100
69.50	71.20	1.70	1.70	100
71.20	74.20	3.00	3.00	100
74.20	77.20	3.00	3.00	100
77.20	78.00	0.80	0.75	94
78.00	78.50	0.50	0.40	80
78.50	79.00	0.50	0.50	100
79.00	79.50	0.50	0.42	84
79.50	80.50	1.00	0.85	85
80.50	82.00	1.50	1.50	100
82.00	83.20	1.20	1.15	96
83.20	85.00	1.80	1.70	94
85.00	85.30	0.30	0.30	100
85.30	86.20	0.90	0.90	100
86.20	88.00	1.80	1.80	100
88.00	89.20	1.20	1.18	98
89.20	90.25	1.05	1.02	97
90.25	92.25	2.00	2.00	100
92.25	92.50	0.25	0.25	100
92.50	94.00	1.50	1.50	100
94.00	95.00	1.00	0.95	95
95.00	96.00	1.00	0.88	88
96.00	98.00	2.00	2.00	100

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221439

PROJECT: MOINA HOLE NO: SMD 11 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
98.00	101.00	3.00	3.00	100
101.00	102.00	1.00	0.98	98
102.00	104.20	2.20	2.20	100
104.20	107.20	3.00	3.00	100
107.20	110.20	3.00	2.92	97
110.20	112.25	2.05	2.05	100
112.25	113.20	0.95	0.90	95
113.20	114.50	1.30	0.70	54
114.50	117.20	2.70	2.00	74
117.20	119.20	2.00	1.80	90
119.20	120.00	0.80	0.78	98

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221440

PROJECT: MOINA HOLE NO: SMD 12 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0	3.00	3.00	0.75	25
3.00	6.00	3.00	0.28	9.33
6.00	8.00	2.00	2.00	100
8.00	11.00	3.00	3.00	100
11.00	12.50	1.50	1.50	100
12.50	13.00	0.50	0.35	70
13.00	14.00	1.00	0.80	80
14.00	15.00	1.00	0.86	86
15.00	17.00	2.00	2.00	100
17.00	17.50	0.50	0.50	100
17.50	20.00	2.50	2.20	88
20.00	23.00	3.00	3.00	100
23.00	26.00	3.00	1.70	56.6
26.00	27.00	1.00	0.20	20
27.00	27.25	0.25	0.16	64
27.25	34.50	7.25	2.20	30.3
34.50	37.25	2.75	2.70	98.1
37.25	39.25	2.00	1.10	55
39.25	41.00	1.75	1.40	80
41.00	42.00	1.00	0.76	76
42.00	43.10	1.10	0.58	52.7
43.10	44.00	0.90	0.48	53.3
44.00	45.00	1.00	0.78	78
45.00	46.00	1.00	0.26	26
46.00	48.25	2.25	1.10	48.8
48.25	49.00	0.75	0.18	24
49.00	50.00	1.00	0.12	12
50.00	52.00	2.00	1.75	87.5
52.00	53.00	1.00	0.80	80
53.00	54.50	1.50	1.50	100
54.50	56.00	1.50	1.46	97.3
56.00	59.00	3.00	3.00	100
59.00	62.00	3.00	1.75	58.3
62.00	65.00	3.00	3.00	100
65.00	68.00	3.00	3.00	100
68.00	71.00	3.00	3.00	100
71.00	75.00	4.00	4.00	100
75.00	78.00	3.00	3.00	100
78.00	84.00	6.00	6.00	100
84.00	87.00	3.00	3.00	100
87.00	89.30	2.30	2.30	100
89.30	90.20	0.90	0.90	100
90.20	93.00	2.80	2.70	96.4
93.00	96.00	3.00	3.00	100
96.00	99.00	3.00	3.00	100
99.00	102.00	3.00	3.00	100
102.00	105.00	3.00	3.00	100
105.00	108.00	3.00	3.00	100
108.00	111.00	3.00	3.00	100
111.00	114.00	3.00	3.00	100
114.00	117.00	3.00	3.00	100
117.00	120.00	3.00	3.00	100
120.00	123.00	3.00	3.00	100
123.00	124.25	1.25	1.25	100

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221441

PROJECT: MOINA HOLE NO: SMD 13 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0	3.00	3.00	2.45	81.6
3.00	4.00	1.00	0.80	80
4.00	5.80	1.80	0.80	44.4
5.80	7.00	1.20	1.20	100
7.00	9.80	2.80	1.40	50
9.80	10.50	0.70	0.65	92.8
10.50	11.00	0.50	0.72	144
11.00	11.50	0.50	0.42	84
11.50	13.00	1.50	1.65	110
13.00	13.61	0.61	0.63	103.2
13.61	14.60	0.99	1.04	105
14.60	16.58	1.98	2.40	121.2
16.58	16.90	0.32	0.34	106.2
16.90	17.45	0.55	0.64	116.3
17.45	18.12	0.67	0.64	95.5
18.12	18.90	0.78	0.84	107.6
18.90	19.45	0.55	0.68	123.6
19.45	19.90	0.45	0.44	97.7
19.90	20.75	0.85	0.35	41.1
20.75	21.50	0.75	1.00	133.3
21.50	22.50	1.00	1.00	100
22.50	23.00	0.50	0.90	180
23.00	23.90	0.90	0.85	94.4
23.90	24.48	0.58	0.76	131
24.48	24.90	0.42	0.35	83.3
24.90	25.25	0.35	0.30	85.7
25.25	25.50	0.25	0.25	100
25.50	26.00	0.50	0.55	110
26.00	27.65	1.65	0.55	33.3
27.65	28.35	0.70	1.44	205.7
28.35	29.25	0.90	1.00	111.1
29.25	30.30	1.05	1.35	128.5
30.30	31.50	1.20	1.25	104.1
31.50	33.41	1.91	2.40	125.6
33.41	33.70	0.29	0.30	103.4
33.70	34.85	1.15	1.20	104.3
34.85	37.15	2.30	2.40	104.3
37.15	37.50	0.35	0.45	128.5
37.50	38.00	0.50	0.60	120
38.00	38.25	0.25	0.25	100
38.25	39.14	0.89	0.60	67.4
39.14	39.40	0.26	0.30	115.3
39.40	40.00	0.60	0.25	41.6
40.00	40.75	0.75	0.55	73.3
40.75	41.25	0.50	0.46	92
41.25	41.60	0.35	0.16	45.7
41.60	42.33	0.73	0.72	98.6
42.33	43.00	0.67	0.68	101.4
43.00	43.50	0.50	0.46	92
43.50	43.90	0.40	0.10	25
43.90	44.75	0.85	0.80	94.1
44.75	45.33	0.58	0.55	94.8
45.33	46.00	0.67	0.94	140.2
46.00	46.32	0.32	0.30	93.7

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221442

PROJECT: MOINA HOLE NO:SMD 13 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
46.32	47.25	0.93	1.05	112.9
47.25	48.34	1.09	1.24	113.7
48.34	48.50	0.16	0.15	93.7
48.50	48.90	0.40	0.14	35
48.90	50.25	1.35	0.20	14.8
50.25	50.75	0.50	0.65	130
50.75	51.70	0.95	1.00	105.2
51.70	52.25	0.55	0.60	109.0
52.25	52.60	0.35	0.35	100
52.60	52.85	0.25	0.26	104
52.85	53.25	0.40	0.35	87.5
53.25	54.10	0.85	0.85	100
54.10	54.25	0.15	0.23	166.6
54.25	54.50	0.25	0.27	108
54.50	55.30	0.80	1.00	125
55.30	55.45	0.15	0.15	100
55.45	55.75	0.30	0.12	40
55.75	55.85	0.10	0.17	170
55.85	57.43	1.58	1.80	113
57.43	58.00	0.57	0.55	96.4
58.00	58.20	0.20	0.20	100
58.20	59.25	1.05	1.10	104
59.25	60.07	0.82	0.90	91
60.07	60.83	0.76	0.68	89.4
60.83	61.10	0.27	0.20	74
61.10	61.75	0.65	0.82	126
61.75	62.03	0.28	0.15	53.5
62.03	63.53	1.50	1.40	93.3
63.53	64.75	1.22	1.20	93.7
64.75	66.30	1.55	1.70	109.6
66.30	67.00	0.70	0.62	88.5
67.00	67.50	0.50	0.61	122
67.50	68.00	0.50	0.50	100
68.00	68.25	0.25	0.30	120
68.25	68.75	0.50	0.41	82
68.75	69.32	0.57	0.83	145
69.32	69.75	0.43	0.35	81.3
69.75	71.25	1.50	1.62	108
71.25	72.50	1.25	1.30	104
72.50	73.25	0.75	0.80	106.6
73.25	73.64	0.39	0.27	69.2
73.64	74.00	0.36	0.26	72.2
74.00	74.50	0.50	0.45	90
74.50	74.90	0.40	0.10	25
74.90	75.60	0.70	0.10	14.2
75.60	77.00	1.40	0.12	8.3
77.00	77.65	0.65	0.60	92.3
77.65	77.75	0.10	0.12	120
77.75	78.42	0.67	0.10	14.9
78.42	79.00	0.58	0.40	68.9
79.00	79.60	0.60	0.28	46.6
79.60	80.00	0.40	0.48	120
80.00	80.70	0.70	0.20	28.5
80.70	81.00	0.30	0.63	210

221443

PROJECT: MOINA HOLE NO: SMD 13 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
81.00	81.38	0.38	0.25	65.7
81.38	82.00	0.62	0.35	56.4
82.00	82.30	0.30	0.36	120
82.30	82.75	0.45	0.40	88
82.75	83.00	0.25	0.25	100
83.00	83.50	0.50	0.48	96
83.50	84.25	0.75	0.76	101
84.25	84.75	0.50	0.60	120
84.75	85.50	0.75	0.80	106.6
85.50	85.90	0.40	0.30	75
85.90	86.75	0.85	0.85	100
86.75	88.00	1.25	1.35	108
88.00	89.25	1.25	1.20	96
89.25	90.00	0.75	0.80	106.6
90.00	92.25	2.25	2.15	95.5
92.25	95.25	3.00	3.00	100
95.25	98.25	3.00	3.00	100
98.25	100.25	2.00	2.00	100
100.25	101.25	1.00	1.00	100
101.25	104.25	3.00	3.00	100
104.25	107.07	2.82	2.85	101
107.07	107.88	0.81	0.72	88.8
107.88	108.10	0.22	0.20	90.9
108.10	109.00	0.90	0.90	100
109.00	110.25	1.25	1.20	96
110.25	111.00	0.75	0.50	66.6
111.00	112.00	1.00	1.00	100
112.00	112.50	0.50	0.25	50
112.50	113.25	0.75	0.65	86.6
113.25	115.10	1.85	1.80	97.2
115.10	116.00	0.90	0.85	94.4
116.00	116.25	0.25	0.20	80
116.25	119.05	2.80	2.75	98.2
119.05	122.15	3.10	3.05	98.3
122.15	123.12	0.97	0.90	92.7
123.12	124.00	0.88	0.90	97.7
124.00	124.80	0.80	0.75	93.7
124.80	125.45	0.65	0.90	138
125.45	126.40	0.95	1.00	105.2
126.40	127.25	0.85	0.85	100
127.25	127.75	0.50	0.60	120
127.75	129.75	2.00	2.00	100
129.75	131.00	1.25	1.25	100
131.00	131.25	0.25	0.25	100
131.25	134.25	3.00	3.00	100
134.25	135.00	0.75	0.70	107
135.00	136.00	1.00	1.00	100
136.00	136.50	0.50	0.45	90
136.50	137.25	0.75	0.85	89.4
137.25	138.20	0.95	0.80	84.2
138.20	140.25	2.05	1.00	48.7
140.25	143.25	3.00	3.25	108.3
143.25	145.25	2.00	2.10	105
145.25	147.15	1.90	2.10	110.5
147.15	149.25	2.10	2.12	100.9

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221444

PROJECT: MOINA HOLE NO: SMD 13 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
149.25	150.25	1.00	0.92	92
150.25	150.50	0.25	0.34	136
150.50	152.25	1.75	1.95	111
152.25	155.25	3.00	3.05	101.6
155.25	156.25	1.00	1.05	105
156.25	157.50	1.25	1.55	124
157.50	160.25	2.75	2.85	103
160.25	161.25	1.00	0.95	95
161.25	162.15	0.90	1.04	115
162.15	162.60	0.45	0.50	125
162.60	163.75	1.15	0.70	60.8
163.75	164.30	0.55	0.45	81.8
164.30	165.50	1.20	0.93	77.5
165.50	166.70	1.20	0.50	41.6
166.70	168.00	1.30	0.65	50
168.00	169.00	1.00	1.00	100
169.00	169.75	0.75	0.90	120
169.75	170.00	0.25	0.20	80
170.00	170.84	0.84	0.70	83.3
170.84	171.10	0.26	0.25	96.1
171.10	171.34	0.24	0.12	50
171.34	171.80	0.46	0.27	58.6
171.80	172.10	0.30	0.40	133
172.10	172.40	0.30	0.15	50
172.40	173.25	0.85	0.95	111
173.25	176.25	3.00	3.20	106
176.25	178.00	1.75	2.15	124
178.00	179.25	1.25	1.30	104
179.25	180.40	1.15	1.10	95.6
180.40	182.25	1.85	0.80	43.2

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221445

PROJECT: MOINA HOLE NO: CMD 14 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0	3.00	3.00	0.25	8.0
3.00	4.50	1.50	0.15	10.0
4.50	6.00	1.50	0.25	16.6
6.00	7.50	1.50	0.10	6.0
7.50	10.32	2.84	2.20	77.4
10.32	11.00	0.64	0.64	100
11.00	12.55	1.55	1.30	83.8
12.55	14.15	1.60	1.60	100
14.15	17.00	2.85	2.80	98.2
17.00	19.00	2.00	2.00	100
19.00	20.17	1.17	0.95	81.1
20.17	23.17	3.00	3.00	100
23.17	26.17	3.00	3.00	100
26.17	29.17	3.00	3.00	100
29.17	32.17	3.00	3.00	100
32.17	35.17	3.00	3.00	100
35.17	37.92	2.75	2.75	100
37.92	41.10	3.18	3.00	94.3
41.10	44.10	3.00	3.00	100
44.10	47.17	3.07	3.00	97.7
47.17	50.17	3.00	2.95	98.3
50.17	53.17	3.00	2.90	96.6
53.17	56.17	3.00	3.00	100
56.17	59.17	3.00	3.00	100
59.17	62.17	3.00	3.00	100
62.17	65.17	3.00	3.00	100
65.17	68.17	3.00	2.90	96.6
68.17	71.00	2.83	2.75	97.1
71.00	74.17	3.17	3.17	100
74.17	76.35	2.17	2.17	100
76.35	79.50	3.15	3.15	100
79.50	82.64	3.14	3.14	100
82.64	85.76	3.12	3.08	98.7
85.76	88.95	3.19	3.08	96.5
88.95	92.05	3.10	3.10	100
92.05	95.17	3.12	3.10	99.3
95.17	98.17	3.00	3.00	100
98.17	101.17	3.00	3.00	100
101.17	104.17	3.00	3.00	100
104.17	107.17	3.00	3.00	100
107.17	110.17	3.00	3.00	100
110.17	113.17	3.00	3.00	100
113.17	116.17	3.00	3.00	100
116.17	119.17	3.00	3.00	100
119.17	112.17	3.00	3.00	100
122.17	125.17	3.00	3.00	100
125.17	128.17	3.00	3.00	100
128.17	131.17	3.00	3.00	100
131.17	134.17	3.00	3.00	100
134.17	137.17	3.00	3.00	100
137.17	140.17	3.00	3.00	100
140.17	143.17	3.00	3.00	100
143.17	146.17	3.00	3.00	100
146.17	149.17	3.00	3.00	100
149.17	152.17	3.00	3.00	100
152.17	155.17	3.00	3.00	100
155.17	158.17	3.00	3.00	100

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221446

PROJECT: MOINA HOLE NO: SMD 14 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
158.17	161.17	3.00	3.00	100
161.17	164.17	3.00	3.00	100
164.17	167.17	3.00	3.00	100
167.17	170.17	3.00	3.00	100
170.17	173.17	3.00	3.00	100
173.17	176.17	3.00	3.00	100
176.17	179.17	3.00	3.00	100
179.17	182.17	3.00	3.00	100
182.17	185.17	3.00	3.00	100
185.17	188.17	3.00	3.00	100
188.17	191.17	3.00	3.00	100
191.17	194.17	3.00	3.00	100

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221447

PROJECT: MOINA HOLE NO: SMD 15 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0	2.00	2.00	1.30	65
2.00	6.00	4.00	2.00	50
6.00	7.25	1.25	1.30	104
7.25	8.12	0.87	0.60	68.9
8.12	8.50	0.38	0.50	131.5
8.50	9.00	0.50	0.70	140
9.00	10.00	1.00	0.85	85
10.00	11.00	1.00	1.15	115
11.00	13.15	2.50	2.10	84
13.15	16.50	3.35	3.45	102.9
16.50	17.00	0.50	0.50	100
17.00	20.10	3.10	3.35	108
20.10	23.10	3.00	1.45	48.3
23.10	25.50	2.40	1.25	52
25.50	28.50	3.00	no core taken	
28.50	33.00	4.50	1.65	36.6
33.00	34.15	1.15	0.75	65.2
34.15	35.12	0.97	1.12	115.4
35.12	38.12	3.00	2.70	90.0
38.12	41.12	3.00	3.00	100
41.12	42.67	1.55	1.48	95.4
42.67	44.12	1.45	1.50	96.6
44.12	47.12	3.00	3.00	100
47.12	49.35	2.23	2.05	91.9
49.35	50.12	0.77	0.75	97.4
50.12	53.12	3.00	2.85	95.0
53.12	56.12	3.00	3.00	100
56.12	59.00	2.88	2.88	100
59.00	62.12	3.12	3.00	96.1
62.12	65.12	3.00	3.00	100
65.12	68.12	3.00	3.00	100
68.12	71.12	3.00	3.00	100
71.12	74.12	3.00	2.90	96.6
74.12	77.12	3.00	3.00	100
77.12	80.12	3.00	3.00	100
80.12	83.12	3.00	2.85	95.0
83.12	84.22	1.10	1.10	100
84.22	86.12	1.90	1.90	100
86.12	88.00	1.88	1.75	93
88.00	89.12	1.12	1.20	107.1
89.12	92.12	3.00	3.00	100
92.12	95.12	3.00	3.00	100
95.12	96.62	1.50	1.50	100
96.62	98.12	1.50	1.50	100
98.12	101.12	3.00	2.90	96.6
101.12	104.12	3.00	2.90	96.6
104.12	107.12	3.00	3.00	100
107.12	108.75	1.63	1.63	100
108.75	109.25	0.50	0.50	100
109.25	111.90	2.65	2.55	96.2
111.90	113.12	1.22	1.22	100
113.12	116.12	3.00	3.20	106.6

221448

PROJECT: MOINA HOLE NO: SMD 16 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0	3.75	3.75	2.40	64
3.75	5.95	2.20	1.00	45.4
5.95	6.75	0.80	1.00	125
6.75	7.75	1.00	1.00	100
7.75	8.85	1.10	1.05	95.4
8.85	9.50	0.65	1.10	169.2
9.50	10.25	0.75	0.80	106.6
10.25	10.85	0.60	0.82	136.6
10.85	12.25	1.35	1.15	85.1
12.25	13.20	0.95	1.00	105.2
13.20	14.00	0.80	0.68	85.0
14.00	14.25	0.25	0.30	120
14.25	15.50	1.25	0.50	40
15.50	16.00	0.50	0.55	110
16.00	17.00	1.00	1.00	100
17.00	18.00	1.00	1.00	100
18.00	18.70	0.70	0.70	100
18.70	20.30	1.60	1.30	81.2
20.30	20.72	0.42	0.42	100
20.72	23.00	2.28	2.85	125
23.00	24.33	1.33	0.88	66.1
24.33	26.27	1.94	0.32	16.4
26.27	29.27	3.00	2.50	83.3
29.27	30.37	1.10	1.13	102.7
30.37	32.18	1.81	1.62	89.5
32.18	33.55	1.17	1.40	119.6
33.55	34.55	1.00	0.95	95.0
34.55	35.27	0.72	0.65	65.4
35.27	38.27	3.00	2.95	98.3
38.27	41.27	3.00	3.05	101.6
41.27	42.70	1.43	1.15	80.4
42.70	44.27	1.57	1.57	100
44.27	46.00	1.73	1.80	104.
46.00	47.27	1.27	1.25	98.4
47.27	49.94	2.67	2.70	101.1
49.94	50.78	0.84	0.84	100
50.78	51.73	0.95	0.80	84.2
51.73	53.27	1.54	1.52	98.7
53.27	54.26	0.99	0.96	96.9
54.26	55.60	1.34	1.20	89.5
55.60	56.28	0.68	0.68	100
56.28	57.20	0.92	0.92	100
57.20	58.25	1.05	0.87	82.8
58.25	58.75	0.50	0.30	60
58.75	59.27	0.52	0.60	115.3
59.27	60.70	1.43	1.50	104.8
60.70	61.75	1.05	1.10	104.7
61.75	63.53	1.78	1.78	100
63.53	64.60	1.07	1.07	100
64.60	65.86	1.26	1.20	95.2
65.86	67.25	1.39	1.60	115.1
67.25	68.27	1.02	0.75	73.5
68.27	70.25	1.98	2.35	118.6
70.25	71.27	1.02	1.00	98.0
71.27	72.25	0.98	0.72	73.4

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221449

PROJECT: MOINA HOLE NO:SMD 16 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
72.25	73.50	1.25	0.25	20
73.50	74.27	0.77	0.10	12.9
74.27	75.41	1.14	0.40	35.0
75.41	76.05	0.64	0.64	100
76.05	76.45	0.40	0.40	100
76.45	77.27	0.82	0.82	100
77.27	77.92	0.65	0.55	84.6
77.92	78.50	0.56	0.07	12.0
78.50	79.25	0.75	0.65	86.6
79.25	80.25	1.00	0.22	22.0
80.25	80.54	0.29	0.20	68.9
80.54	81.00	0.46	0.16	34.7
81.00	82.00	1.00	0.46	46
82.00	82.50	0.50	0.42	84
82.50	83.27	0.77	1.00	129.8
83.27	84.25	0.98	1.00	102
84.25	84.75	0.50	0.55	110
84.75	85.50	0.75	0.80	106.6
85.50	86.25	0.75	0.72	96.0
86.25	86.90	0.65	0.65	100
86.90	87.75	0.85	0.95	111.7
87.75	89.27	1.52	1.60	105.2
89.27	90.50	0.73	1.40	191.7
90.50	91.60	1.10	1.10	100
91.60	92.27	0.67	0.70	104.4
92.27	93.48	1.21	1.21	100
93.48	94.30	0.82	0.90	109.7
94.30	95.10	0.80	0.80	100
95.10	96.60	1.50	1.10	73.3
96.60	97.30	0.70	0.80	114.2
97.30	98.11	0.81	0.45	55.5
98.11	99.00	0.89	0.89	100
99.00	101.27	2.27	2.27	100
101.27	102.22	0.95	1.10	115.7
102.22	103.25	1.03	1.00	97.0
103.25	104.27	1.02	1.10	107.8
104.27	106.25	1.98	2.10	106
106.25	107.27	1.02	1.20	117.6
107.27	109.68	2.41	2.45	101.6
109.68	112.78	3.10	3.05	98.3
112.78	115.88	3.10	3.10	100
115.88	119.09	3.21	3.10	96.5
119.09	122.27	3.18	3.07	96.5
122.27	124.84	2.57	2.55	99.2
124.84	125.43	0.55	0.55	100
125.43	128.27	2.84	2.84	100
128.27	131.27	3.00	2.87	95.6
131.27	134.27	3.00	3.00	100
134.27	137.27	3.00	3.00	100
137.27	140.27	3.00	3.00	100
140.27	141.70	1.43	1.50	104.8
141.70	143.27	1.57	1.57	100
143.27	146.27	3.00	3.00	100
146.27	149.27	3.00	3.00	100
149.27	152.27	3.00	3.00	100
152.27	155.27	3.00	3.00	100
155.27	158.27	3.00	3.00	100

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221450

PROJECT: MOINA HOLE NO: SMD 16 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
158.27	161.10	2.83	2.75	97.1
161.10	164.25	3.15	3.10	98.4
164.25	164.82	0.57	0.35	61.4
164.82	167.27	2.45	2.45	100
167.27	170.27	3.00	3.00	100
170.27	171.27	1.00	0.70	70

221451

PROJECT: MOINA HOLE NO: SMD 17 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0	20.15	20.15	0.80	3.90
20.15	21.65	1.50	1.70	113.3
21.65	23.15	2.15	1.50	69.7
23.15	24.20	1.05	1.05	100
24.20	26.15	1.95	1.70	87.1
26.15	29.15	3.00	3.00	100
29.15	32.15	3.00	3.00	100
32.15	35.15	3.00	3.00	100
35.15	36.85	1.70	1.70	100
36.85	38.90	2.05	2.05	100
38.90	41.15	2.25	2.30	102.2
41.15	44.15	3.00	3.00	100
44.15	47.15	3.00	3.00	100
47.15	48.00	1.00	0.77	77
48.00	48.33	0.33	0.10	30.3
48.33	50.15	1.82	2.00	109.8
50.15	53.15	3.00	3.00	100
53.15	56.07	2.92	2.90	99.3
56.07	59.15	3.06	3.10	101.3
59.15	60.60	1.05	1.25	119.
60.60	62.15	1.55	1.60	103.2
62.15	63.75	1.60	1.60	100
63.75	65.15	1.40	1.40	100
65.15	68.15	3.00	3.00	100
68.15	71.15	3.00	2.97	99
71.15	74.15	3.00	3.00	100

221452

PROJECT: MOINA HOLE NO: SMD 20 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0	4.00	4.00	.85	21.3
4.00	6.00	2.00	.10	5.0
6.00	10.00	4.00	.25	6.3
10.00	12.00	2.00	.35	17.5
12.00	13.30	1.30	.15	11.5
13.30	15.00	1.70	.60	35.3
15.00	16.30	1.30	.90	69.2
16.30	18.00	1.70	.60	35.3
18.00	19.30	1.30	1.30	80.8
19.30	21.00	1.70	1.20	70.6
21.00	22.30	1.30	.70	53.8
22.30	24.00	1.70	1.15	67.6
24.00	26.25	2.25	1.75	77.8
26.25	27.90	1.65	.40	24.2
27.90	29.25	1.35	1.00	74.1
29.25	30.42	1.17	.95	81.2
30.42	31.50	1.08	.85	78.7
31.50	32.25	.75	.01	1.3
32.25	34.00	1.75	1.75	100.0
34.00	35.25	1.25	1.30	104.0
35.25	36.60	1.35	1.15	85.2
36.60	38.25	1.65	1.30	78.8
38.25	41.25	3.00	.10	3.3
41.25	41.80	.55	1.00	181.8
41.80	44.00	2.20	.60	27.3
44.00	46.25	2.25	.80	35.5
46.25	48.00	1.75	.55	31.4
48.00	49.25	1.25	.15	12.0
49.25	51.00	1.75	.95	54.3
51.00	52.25	1.25	No Core	0.0
52.25	53.63	1.38	.20	14.5
53.63	55.25	1.62	.05	3.1
55.25	57.00	1.75	.20	11.4
57.00	62.25	5.25	No Core	0.0
62.25	65.25	3.00	.05	1.7
65.25	68.25	3.00	1.35	45.0
68.25	69.20	.95	.25	26.3
69.20	69.40	.20	.20	100.0
69.40	70.92	1.52	1.65	108.5
70.92	74.00	3.08	3.20	104.0
74.00	75.50	1.50	1.50	100.0
75.50	77.25	1.75	1.65	94.3
77.25	80.25	3.00	3.00	100.0
80.25	82.40	2.15	2.28	106.0
82.40	83.25	.85	.90	105.9
83.25	86.25	3.00	3.05	101.7
86.25	89.25	3.00	3.00	100.0
89.25	92.25	3.00	3.00	100.0
92.25	94.75	2.50	2.55	102.0

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221453

PROJECT: MOINA HOLE NO: SMD 21 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0.0	6.0	6.00	.60	60
6.00	9.00	3.00	2.80	93
9.00	10.50	1.50	.30	20.0
10.50	13.00	2.50	.55	22.6
13.00	16.00	3.00	3.00	100
16.00	17.25	1.25	1.10	38
17.25	20.25	3.00	3.00	100
20.25	23.25	3.00	3.00	100
23.25	26.25	3.00	3.00	100
26.25	29.25	3.00	2.95	98.33
29.25	32.25	3.00	3.00	100
32.25	35.25	3.00	3.00	100
35.25	38.25	3.00	2.90	96.66
38.25	41.25	3.00	3.00	100
41.25	44.25	3.00	3.00	100
44.25	47.25	3.00	3.00	100
47.25	50.25	3.00	3.00	100
50.25	53.25	3.00	3.00	100
53.25	56.25	3.00	2.95	98.33
56.25	59.25	3.00	3.00	100
59.25	62.25	3.00	2.95	98.33
62.25	65.25	3.00	3.00	100
65.25	67.25	2.00	2.00	100
67.25	68.25	1.00	1.05	105
68.25	71.25	3.00	2.95	98.33
71.25	74.25	3.00	3.00	100
74.25	77.25	3.00	2.95	98.33
77.25	80.25	3.00	3.00	100
80.25	83.25	3.00	3.00	100
83.25	86.25	3.00	2.90	96.66
86.25	89.25	3.00	2.95	98.33
89.25	92.00	2.75	2.70	98.18
92.00	95.12	3.12	2.90	92.94
95.12	98.25	3.13	3.05	97.44
98.25	101.25	3.00	3.00	100
101.25	104.25	3.00	3.00	100
104.25	107.25	3.00	3.00	100
107.25	110.25	3.00	2.95	98.32
110.25	113.25	3.00	3.00	100
113.25	116.25	3.00	3.00	100
116.25	119.25	3.00	3.00	100
119.25	122.25	3.00	3.00	100
122.25	125.25	3.00	3.00	100
125.25	126.85	1.60	1.60	100
126.85	128.25	1.40	1.45	103.57
128.25	131.25	3.00	2.95	98.33
131.25	134.25	3.00	3.00	100
134.25	137.25	3.00	3.00	100
137.25	140.25	3.00	3.00	100
140.25	143.25	3.00	2.95	98.33
143.25	146.25	3.00	3.00	100
146.25	149.25	3.00	3.00	100
149.25	150.78	1.58	1.50	98.03
150.78	152.25	1.57	1.55	98.72
152.25	155.25	3.00	3.00	100

360

221454

PROJECT: MOINA HOLE NO: SMD 22 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0.0	21.30	21.30	1.60	7.51
21.30	23.10	1.80	1.65	91.66
23.10	25.00	1.90	1.45	76.31
25.00	26.80	1.80	1.50	83.33
26.80	29.25	2.45	caves 1.55	63.26
29.25	31.00	1.75	1.25	71.42
31.00	32.80	1.80	.25	13.88
32.80	34.00	1.20	.90	75.00
34.00	36.00	2.00	.80	40.00
36.00	37.00	1.00	.70	70.00
37.00	39.00	2.00	.45	22.50
39.00	40.00	1.00	.30	30.00
40.00	41.50	1.50	.35	23.33
41.50	43.00	1.50	.60	40.00
43.00	45.00	2.00	1.70	85.00
45.00	45.50	.50	.60	120
45.50	48.00	2.50	1.20	48.00
48.00	49.00	1.00	.60	60.00
49.00	49.30	.30	.10	33.33
49.30	51.50	2.20	1.40	63.63
51.50	52.10	.60	.40	66.66
52.10	53.00	.90	.85	94.44
53.00	53.50	.50	.60	120
53.50	55.70	2.20	2.75	125
55.70	58.00	2.30	1.85	80.43
58.00	59.00	1.00	.90	90.
59.00	62.00	3.00	2.75	91.66
62.00	65.00	3.00	2.90	96.66
65.00	68.00	3.00	3.00	100
68.00	71.00	3.00	3.00	100
71.00	74.00	3.00	2.90	96.66
74.00	77.00	3.00	2.90	96.66
77.00	80.00	3.00	3.00	100
80.00	83.00	3.00	3.00	100
83.00	86.00	3.00	3.00	100
86.00	89.00	3.00	3.00	100
89.00	91.30	2.30	2.25	97.82
91.30	92.35	1.05	1.05	100
92.35	93.45	1.10	1.10	100
93.45	95.00	1.55	1.65	106.45
95.00	98.00	3.00	2.95	98.33
98.00	101.00	3.00	3.00	100
101.00	104.00	3.00	3.00	100
104.00	107.00	3.00	2.95	98.33
107.00	110.00	3.00	3.00	100.00

221455

PROJECT: MOINA HOLE NO: SMD 23 RECOVERY LOG

FROM	TO	ADVANCED INTERVAL	RECOVERY	RECOVERED %
0	2.50	2.50	.40	16.00
2.50	4.00	1.50	.20	13.33
4.00	6.00	2.00	1.00	50.00
6.00	7.80	1.80	1.90	105.00
7.80	8.10	.30	.40	133.33
8.10	11.10	3.00	3.00	100.00
11.10	14.10	3.00	3.00	100.00
14.10	17.10	3.00	3.00	100.00
17.10	20.10	3.00	3.00	100.00
20.10	23.10	3.00	3.00	100.00
23.10	26.10	3.00	2.60	86.66
26.10	27.00	.90	.65	72.00
27.00	28.30	1.30	1.00	76.92
28.30	29.10	.80	.70	87.50
29.10	32.00	2.90	2.80	96.55
32.00	33.70	1.70	1.70	100.00
33.70	34.25	.55	.60	109.09
34.25	35.10	.85	.95	111.76
35.10	37.00	1.90	2.20	115.78

APPENDIX 27

SEDIMENTOLOGICAL LOGS BY P.W. STAINTON

A. PRELIMINARY

P.W. Stainton conducted an investigation into the lithological reasons (if any) controlling the main wrigglite body .

Cores from eleven diamond drill holes over the deposit were sliced longitudinally and logged lithologically, taking special note of sedimentary and organo-sedimentary structures, visible (X20 magn.) mineralogy, and other features that would shed light on the depositional history of the Gordon Limestone at Moina.

Graphical logs are presented as the following drawings:-

ML	2	TAS-76-54 A
SMD	4	B
	5	C
	6	D
	7	E
	9	F
	10	G
	12	H
	14	I
	15	J
	17	K

B. CONCLUSIONS OF P.W. STAINTON.

An interpreted depositional environment is shown on drawing TAS-76-58, and Stainton's conclusions are:

"(a) Stratigraphy at Moina confirms that the Gordon Limestone Formation is a transgressive unit, conformable on Moina Sandstone Formation.

(b) The basal part of the Gordon River Limestone Formation contains a biofacies - algal stromatolite bioherm - build up, 60 m. thick near Shepherd and Murphy mine.

(c) The definition of carbonate lithologic types has been complicated by metasomatic (and to a lesser degree, metamorphic) alteration. However, in the writer's opinion, the following is a confident interpretation of the original lithology:

(i) calc-silicate: originally marl.
(ii) coarse garnetiferous rock: originally dolarenite,dolomite
(iii) Wollastonite hornfels: originally lime mud.

(d) The definition of 'wrigglite' must be clarified, as the wriggly texture (defined by P.W. Askins) may also be attributed to the natural sedimentary texture displayed by stromatolites.

(e) The writer suggests that the 'wrigglite', as found at Moina, can also, and realistically, be defined as:

'Metasomatised (fluorite, garnet, tin tungsten) algal stromatolite....'

(f) Lithologic correlation within the algal unit is not possible (a common feature of bioherms), but once recognised, such a unit must be regarded as a 'special' biofacies within the carbonate regime.

(g) A realistic datum horizon would be the base of the algal unit. By using this horizon as datum, the build-up of the algal bioherm is demonstrated, (see drawing TAS-76-58)

(h) Although unconfirmed, it can be interpreted that the bioherm is developed over (and probably confined to) a 'palaeo-swell'. This may have extended offshore from the coast, or, a reef development just offshore.

(i) The transitional beds between Moina sandstone and the algal stromatolites were (prior to metasomatism) composed of coarse, sometimes vuggy dolomite, calcareous siltstone and marls.

(j) The iron rich sulphides contained in the algal stromatolite unit encountered in SMD 9, SMD 5, and outcropping near DOM 2, may be interpreted as syngenetic (pyrite), remobilised during metasomatism.

The location of the above occurrences implies small, localised algal swamps with limited current disturbance and circulation, and anoxic (reducing) conditions. Such an environment is regarded as a concentrating point for uranium as well as other base metal (Cu, Pb, Zn, Sn) sulphides, adjacent to a hinterland suitably enriched in and supplying those elements.

(k) Dolomitic and stromatolitic facies generally indicate coastal/near-shore environment, and should be regarded as excellent indicators for possible syngenetic sulphide (and uranium) deposits, in an adjacent and lateral facies deposited under anoxic conditions."

C. COMMENTS.

The author (P.W. Askins) does not believe that the wrigglite is a replacement of algal stromatolite for the following reasons:-

(a) no unreplaced stromatolites with similar textures to wrigglite have been found;

(b) where partial replacement textures are observed in ML 2, SMD 5, SMD 12 the wrigglite is as veinlet-like replacement of limestone similar to that illustrated in Fig.8;

(c) evidence for predominantly structural control rather than lithological control of the wrigglite is abundant, as has been discussed elsewhere;

(d) the layering in wrigglite is very complex; unconformable relationships between the layers are common, and such geometry is unknown in stromatolites;

(e) selective replacement of pre-existing stromatolite layering by rhythmic layers of fluorite, magnetite and calc-silicate minerals over such a large scale is an unlikely, exceedingly specialized, process .

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APPENDIX 28.

221459

AMDEL. CHEMICAL EXTRACTION OF FLUORIDE FROM WRIGGLITE.

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221460

amdel
The Australian Mineral Development Laboratories

Flemington Street, Frewville, South Australia 5063
Phone Adelaide 79 1662, telex AA82520

Please address all correspondence to Frewville
In reply quote: CM 570/2702
3/1/4/0

9 December 1975

The General Manager,
Comalco Limited,
GPO Box 2773 Y,
MELBOURNE. Vic. 3001

Attention: Mr A.H. Bartlett

Dear Sir,

Chemical Treatment of Fluorite Ore

Following his visit to Comalco on 28 October 1975 Dr Keith Henley advised us of your enquiry regarding chemical extraction of fluoride from low grade fluorite ores which are difficult to treat by flotation.

For some time, we have been interested in possible chemical methods for treating lower-grade fluorite ores and concentrates, to produce hydrogen fluoride or other fluorine chemicals, and have been doing preliminary experimental work at our own expense to examine some possible treatment methods.

One process in particular has given promising results. This comprises roasting with a reagent at about 400°C to permit extraction of fluoride in a form that can be converted to aluminium fluoride by known technology. Fluoride extraction of 83 to 89% was achieved in five laboratory experiments, from an ore that has not given acceptable flotation results at Amdel or elsewhere.

The process has several advantages:

(a) the reaction temperature is relatively low,

(b) the reagent is fairly cheap,

(c) the fluoride is readily recoverable,

(d) the technology for conversion to aluminium fluoride is already established at industrial scale.

We see our preliminary test results as promising and would be glad to continue development of the roasting step for a company with a suitable fluorite ore. We are therefore taking the liberty of submitting to you a proposal for examining experimentally the behaviour of your Moina and Mt Garnet fluorites in this process.

To: Comalco Limited 9 December 1975

The technical programme would comprise a laboratory experimental study of the dependence of fluoride recovery from the two ores on the level of reagent addition, on fineness of grind and on reaction temperature and time. A larger (1 kg) scale test would be included.

The estimated fee for this programme is $5,000. We realise that your company may wish to know more of the technical details of the process before committing itself to supporting this proposed work. If so, we suggest a round-table discussion here in Adelaide, and would then ask you to sign an agreement to maintain confidentiality until you are actually supporting the programme at Amdel. We trust you will understand our position, as we have invested our own, very limited, resources until now.

If you accept our proposal, the project would be carried out under the Standard Conditions of Contract which are included and form part of this Proposal.

If the experimental results are sufficiently promising we can make a preliminary estimate of capital and operating costs as Stage II. We look forward to receiving your reply.

Yours faithfully,
The Australian Mineral Development Laboratories

RR Wilmshurst

for F.R. Hartley
Director.

RA:cah
Encl.

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221462

THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES

ADELAIDE, SOUTH AUSTRALIA

CONDITIONS OF CONTRACT

1. **Co-operation between Amdel and Client.** Subject to the terms and conditions of this Agreement, all information obtained by Amdel through work on the project will be made available to the Client during normal hours, and Amdel will communicate to the Client all information pertinent to the project as it progresses. An authorised representative of the Client may confer with the appropriate members of the Amdel staff at mutually agreed times for the purpose of providing such assistance as will make it possible for Amdel to achieve the objectives of the project.
   Amdel welcomes the co-operation of the Client in providing such information, assistance and co-operation as is necessary for Amdel to achieve the objectives of the project.

2. **Variation of the Project.** The terms of reference of the project may be varied by mutual agreement expressed in writing between Amdel and the Client, and such agreement will include variations to the date of completion of the project and the overall cost of the project.

3. **Reports.** Amdel agrees to supply the client with copies of reports of its findings and progress during the term of the project at monthly or such other intervals as Amdel and the Client agree.

4. **Confidential Nature of Project.** Amdel agrees that the Amdel staff shall not, without the consent of the Client, disclose to members of the Amdel Council or any other party any confidential information received from the Client or any report or information relevant to the project.

5. **Conflict of Interest.** Amdel will endeavour to avoid at all times any conflict of interest between its Client and any third party with regard to work done on the project.

6. **Patents.** Amdel covenants that each of its servants and employees has undertaken to assign to Amdel all inventions produced by the servant or employee arising from the course of his employment. It is mutually agreed between Amdel and the Client
   a. That where the objective of the project is the making of an invention Amdel will, if the Client agrees, make application for Letters Patent and assign any grant resulting therefrom to the Client at the cost of the Client
   b. that where the project is of such a nature that a patentable idea will or is likely to emerge the rights attaching thereto shall vest in Amdel who agrees to negotiate with the Client in relation to it
   c. and that where a patentable idea emerges coincidentally from any project or series of projects all rights to it remain the property of Amdel.

   It is further agreed between Amdel and the Client that any expense for filing assignment prosecution and recording relating to such Letters Patent shall be borne by each of the parties in proportion to the interest each party has in the Letters Patent but such expenses shall not in any event constitute part of the principal sum payable by the Client for the project.

7. **Publicity, Advertising and Legal Actions.** The Client agrees that he will not use or allow to be used any information arising from the project in association with the name of The Australian Mineral Development Laboratories, its officers, servants, agents, employees and independent contractors in any advertising, promotion or publicity, or any legal proceedings or litigation unless and until the same shall have been submitted to and received the written authority of The Australian Mineral Development Laboratories.

8. **Right to Terminate Contract.** This Agreement shall be terminated on the completion of the project or at such other time and under such conditions as Amdel and the Client mutually agree in writing.

9. **Relationship between Amdel and Client.** Nothing herein contained shall constitute Amdel or its servants, employees or staff members as agents or employees or servants of the Client, but the relationship between Amdel and the Client shall be and remain that of independent contractors.

10. **Default by Amdel.** In the event of any default by Amdel in carrying out the terms and conditions of this Agreement (not being default caused by the Client), the Client may at his discretion give notice in writing of the default to Amdel requiring that the default be remedied within 30 days or such further time as specified after the date of service of the notice. If Amdel does not remedy the default within the time specified in the notice the Client may (except in the circumstances set forth in Clause E (12)) terminate the Agreement. The Client shall not be entitled to recover from Amdel the costs and expenses incurred by reason of the default and any losses or damages incurred by reason of the dependence by the Client on the fulfilment of this Agreement or any consequential or incidental losses or damages or liability to any third party.

11. **Workmanship.** Amdel covenants that its servants, agents and employees will carry out the work required to be done on the project to the best of their skill and ability. If it is discovered during the project or after its completion that Amdel, its servants, agents or employees have made any error or omitted to do anything which ought to have been done according to the proposal, whether negligently or otherwise, Amdel will promptly do such work done in error or omitted to have been done at its own cost and expense and Amdel shall have no other liability whether for negligent act, default or omission or breach of contract.

12. **Default by Client.** In the event of any default by the Client in carrying out the terms and conditions of this Agreement Amdel may in its discretion and without prejudice to any other rights which it may have to terminate this Agreement give the Client written notice of such default requesting him to remedy the default within 30 days after the date of service of the notice or such further time as specified. Should the Client fail or refuse to remedy fully the default within the time specified Amdel may in its discretion terminate the Agreement and upon such determination Amdel shall be entitled to recover from the Client all its costs expenses and reasonable charges incurred up to the date of termination and any costs incurred in anticipation of the continuance of the project or the cancellation of any arrangements made, orders placed or the return or resale of equipment and materials made in anticipation of the continuance of the project.

13. **Submission of Notices.** Any notice required to be given hereunder shall be sufficiently given by either party to the other by registered post or in the ordinary course of the post, whichever is the later, to its address specified herein and such notice shall be deemed to have been given the day after the posting of such notice at any post office in Australia.

14. **Payment.** The Client agrees to pay the invoices rendered by Amdel from time to time and on completion of the project within 30 days after posting by prepaid post. Amdel reserves the right to discontinue work on the project if the Client fails to pay any of the invoices rendered by Amdel within the time specified.

15. **Law Applicable to Contract.** These conditions shall be construed in accordance with the laws in force in the State of South Australia.

16. **Arbitration.** Any dispute which may arise as to the interpretation of these conditions shall be referred to an arbitrator to be appointed by agreement of both parties or, in default of agreement, by the Auditor-General for the time being of the State of South Australia and such arbitrator shall act pursuant to the provisions of the Arbitration Act, 1891-1935.

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EL 7/74 -MOINA-
BENEFICIATION

221463

COMALCO LIMITED

COMALCO

MEMORANDUM

Ref	JCN:pt	Date	15th January, 1976.
To	A.H. BARTLETT	From	J.C. NIXON
At	I.A.C.	At	'95'
Copies	-	Subject	FLUORITE SKARN

As discussed, I visited Amel on 13 January and discussed the substance of the Amdel letter dated 9 December 1975 with John Gooden, officer-in-charge of the Chemical Engineering Section.

Amdel had a change of mind since the letter was written and declined to give details of the process. All I could ascertain was that it involved a calcination, as in a rotary kiln, at 400°c for 1 hour at temperature, followed by a leach. The fluoride would end up in a solution that should be substantially free from iron and silica.

I handed Gooden a copy of an analysis of the material but they were not able to predict the behaviour of all of the minor elements. It was thought that tin would remain in the residue after leaching.

Amdel suggested that a 100 g. or preferably 200 g. sample should be forwarded for a few preliminary experiments. Analysis of major elements would be conducted on the product and a sample would be forwarded to us for determination of the minor elements.

The cost of this work would be $750. It would not matter that the sample was not representative of the deposit, provided it was typical of the type of ore.

J.C. Nixon

J.C. Nixon

221464

COMMONWEALTH ALUMINIUM CORPORATION LIMITED

DESPATCH and LABORATORY WORK REQUISITION NOTE

REFERENCE No. AB/T/MOINA/— No: OLM : 80

TO: AMDEL (ATTN. J. GOODEN) FROM: A.H. BARTLETT

AT: FREWVILLE, S.A. 5063. AT: 95 COLLINS ST.,

DATE: 29TH MARCH, 1976 MELBOURNE. TEL: 654 4211

Per: Consignment No.:

No. of Items:

Category: ~~EXPLORATION~~, R. & D., ~~PRODUCTION, EXTERNAL~~

Claimable Under I. R. & D. Grant: YES/~~NO~~

Sample Numbers	Sample Type	Work Required
MOINA / 1	CRUSHED ROCK.	LOW GRADE FLUORITE ORE (~20% CaF_2 FROM MOINA, TAS.. — "FLUORITE MAGNETITE SKARN
MOINA / 2	"	
		AIM: TO DETERMINE WHETHER OR NOT "ACID GRADE" FLUORITE CAN BE EXTRACTED CHEMICALLY FROM THE ORE SAMPLES PROVIDED. METHOD: DETAILS CONFIDENTIAL TO AMDEL UNTIL LARGER SCALE TESTS UNDERTAKEN. BUT, UNDERSTOOD TO INVOLVE : (i). LOW TEMP. HEAT TREATMENT ~ 400°C. (ii) LEACH. RESULTS: FULL ANALYSES AND COMMENTS TO BE MADE AS SO RECOMMENDATIONS FOR FUTURE WORK SINCE ORE CONTAINS UP TO 0.5% Sn IN FINELY DIVIDED FORM, ADVICE IS ALSO NEEDED ON THE BEHAVIOR AND EXTRACTION OF THIS MINERAL.
		TOTAL NUMBER OF SAMPLES:

SEND RESULTS TO: (1) A.H. BARTLETT (2) DR. J.C. NIXON (3) W. HAND

CHARGE TO: COMALCO SIGNATURE: A.H. Bartlett

THIS COPY TO THE LABORATORY WITH SAMPLES.

571 -

c.c. W. H. Andrews
J. A. Eady
J. C. Nixon
C. Ong
Cairns Office
Devonport Office

EL 7/74, MOINA, BENEFICIATION

Hd. GW.

221465

30th March 1976

Mr. F. R. Hartley,
Director,
Australian Mineral Development
Laboratories,
Flemington Street,
FRESWVILLE, S.A. 5063

Attention: Mr. J. Gooden

Dear Sir,

Chemical Treatment of Fluorite Ore

Further to your letters of 9th and 15th December, 1975 and discussions with Dr. J. C. Nixon and W. H. Andrews of Comalco, we are forwarding to you two samples of "fluorite magnetite skarn" / low grade fluorite ore (about 20% CaF_2) from Moina area in Tasmania. The samples are not representative of the whole ore body, much more drilling will be required before truly representative samples can be made available.

The analyses of the two samples are as follows:-

Analysis	Moina 1	Moina 2
CaF_2	25.2 %	20.5 %
CaO	5.1 %	2.4 %
$CaCO_3$	1.1 %	0.9 %
$MgCO_3$	0.2 %	0.2 %
$FeCO_3$ (?)	0.7 %	0.8 %
Fe_2O_3	29.9 %	34.1 %
SiO_2	20.0 %	22.9 %
TiO_2	0.1 %	0.1 %
Al_2O_3	10.2 %	9.5 %
Zn	700 ppm	1060 ppm
Pb	15 ppm	15 ppm
Cu	115 ppm	70 ppm
Bi	605 ppm	325 ppm
Mo	50 ppm	110 ppm
Sb	55 ppm	45 ppm
W	1270 ppm	1190 ppm
Au	0.1 ppm	0.1 ppm
Sub-total	92.78	91.67
L.O.I.	~7 %	~8 %

cont'd ....

These samples were not analysed for tin, however, from analysis of other samples we believe this "fluorite magnetite skarn" may contain up to 0.5% total Sn in a very finely divided form. Tin determinations by X.R.F. and Na_2O_2 fusion and nitrous oxide flame A.A.S. gave values almost twice those determined by ammonium iodide leach / A.A.S..

Not so all Moina samples have been analysed by XRF none by iodide leach CW

At this stage we are prepared to commit to a maximum of $750 and wish Amdel to determine whether or not "acid grade" fluorite can be extracted chemically from the samples provided.

We appreciate that Amdel has been doing work at its own expense and thank you for inviting us to participate. Initially it must be to a maximum of $750 since like Amdel, we are also on a very tight budget. We also understand that the details of the test method(s) should remain confidential to Amdel until Comalco commits itself to larger scale tests. However, this is a "chicken and egg" situation as we will need considerable detail before agreeing to spending $5000+ on an expanded programme. So that we can assess the desirability of having more research done, we must request that you give us:-

Full analyses of all fractions.
Comments on behaviour of samples.
Recommendations for future work.
Behaviour of tin.
Return all pulps / fractions to us on completion of this initial work.

Separately, we have sent to Amdel two samples:-

Labelled: "Moina 1" and "Moina 2"
Laboratory work requisition note: O.L.M. 80
Reference "number": AB/T/Moina/-
From: A. H. Bartlett
Exploration Department
Comalco Limited.

Thank you for your help, we shall look forward to receiving the results of your tests shortly.

Yours sincerely,

AHB.

A. H. Bartlett,
Manager - Exploration.

JMS

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EL7/74 MOINA BENEFICIATION

221467

amdel
The Australian Mineral Development Laboratories
Flemington Street, Frewville, South Australia 5063
Phone Adelaide 79 1662, telex AA82520

Winner of Award for Outstanding Export Achievement 1975

Please address all correspondence to Frewville,
In reply quote: CM3/1/4/0

6 July 1976

The Manager - Exploration,
Comalco Limited,
GPO Box 2773Y,
MELBOURNE. Vic. 3001

Attention: Mr A.H. Bartlett

REPORT: CM2873/76

YOUR REFERENCE:	Letter of 30 March, 1976 and Requisition Note No.OLM:80
MATERIAL:	Fluorite
LOCALITY:	Moina, Tasmania
IDENTIFICATION:	Moina 1
DATE RECEIVED:	2 April 1976
WORK REQUIRED:	Investigate behaviour in fluoride extraction process

Investigation and Report by: R.J. Allen and D.J. Ellis

Officer in Charge,
Chemical Metallurgy Section: J.E.A. Gooden

for F.R. Hartley
Director

cah

FLUORIDE EXTRACTION FROM FLUORITE

1. INTRODUCTION

In response to a letter from Amdel, Dr J.C. Nixon visited Thebarton for a brief discussion on 13 January 1976. Mr J. Gooden explained that Amdel had done laboratory work at its own expense on a novel process for fluorine extraction from difficult ores. Before divulging details of the process and discussing possible Comalco support of further development, it was suggested that Amdel test, for minimum cost, the response of a sample of Comalco's Moina fluorite ore to the Amdel process. A firmer basis would then exist to discuss Comalco support of further development of the process.

Mr A. Bartlett, Exploration Manager of Comalco Limited then supplied two samples of siliceous fluorite ores from the Moina area of Tasmania. Only the first sample, Moina 1, could be tested within the money approved for the work. The behaviour of tin, present in the ore up to a level of 0.5%, was also of interest.

2. MATERIAL EXAMINED

The ore sample supplied was split into halves and one half ground to pass a 100 mesh BSS screen. An assay sample was riffled from the minus 100 mesh ore and analysed chemically for fluorine, silica and tin. The results are tabulated below, along with those reported by the client. A more comprehensive analysis provided by Comalco Limited is included as Table 1.

Component	percent by weight	
	Comalco	Amdel
F	12.3	12.2
SiO_2	20.0	23.4
SnO_2	nr	0.215

3. EXPERIMENTAL PROCEDURE AND RESULTS

Three experiments were conducted at laboratory bench scale, in which the Moina 1 ore (12.5 g) was reacted in the Amdel process for one, two and three hours. The residue was chemically analysed for fluorine, silica and tin. The results are shown in the following table:

Expt. No.	Reaction Time (hr)	% F in residue	Residue wt. (g)	% of Total Component in Residue		
				F	SiO_2	SnO_2
1	3	3.60	18.45	44	79	96
2	2	3.55	18.77	44	81	93
3	1	3.55	18.75	44	80	96

Clearly, the time of reaction was not a limiting factor in these experiments, since in each case, 56% of the fluorine was extracted, along with 20% of the

silica. Extraction of tin was negligible.

Further experimental work demonstrated that the fluoride extracted could be readily converted to aluminium fluoride. The only impurity detectable in this product by X-ray diffraction was aluminium sulphate. Since the presence of this compound was partly attributed to the small scale of the preparation, a larger scale experiment (300 g ore) was carried out. In order to improve the extraction of fluoride, the ratio of reagent; ore was increased by 33%.

Fluorine determinations indicated the following distribution:

Expt. No.	Reaction Time (hr)	% F in residue	Residue wt. (g)	% of Total F in		
				Residue	Produce	Wash
4	2	2.3	471.5	30	36	17

The fluoride content of the above product was again converted to aluminium fluoride, after a silica removal step. The aluminium fluoride was spectrographically analysed for selected impurities, with the results shown in Table 2. The sulphate content of the aluminium fluoride was also determined, and is included in Table 2.

Typical maximum limits specified by aluminium smelters are also included in Table 2.

The residue from the fluoride extraction step was further leached, to determine whether the aluminium content of the ore could be dissolved as aluminium sulphate without dissolving too much iron. On the contrary, atomic absorption spectroscopic analyses indicated that the ratio of iron:aluminium in the leach solution was about 17:1, with a low extraction efficiency for aluminium.

4. DISCUSSION

In the 300 g scale experiment, 83% of the total fluorine was accounted for, with the balance explained by mechanical losses and incomplete washing, which would be easier to avoid on a larger scale. The extraction of fluoride, based on the residue analysis, was 70%, a substantial improvement on the 56% extraction achieved in the first three experiments, but still well below the 89% extraction achieved with another ore. However, the limited experimental work to date certainly has not yet optimised reaction conditions for the Moina 1 ore, which has a much higher fluorine content than the previous ore. Furthermore, it should be recognised that in a continuous process, the wash liquor could be recycled and its fluoride content recovered as product.

The aluminium fluoride products produced from the larger scale experiment appear to satisfy the specifications for aluminium smelting.

The copper content of product S1 was relatively high, but this is explained by the use of an item of copper apparatus at one point, which could be avoided.

There appears to be little prospect of preparing from the ore aluminium sulphate that would be suitable for use in precipitating the fluoride as aluminium fluoride.

Samples of the aluminium fluoride products, S1 and S2, are being despatched to Comalco for examination, under separate cover. It is emphasized that, on the basis of published investigations, different processing procedures could yield fluoride in the forms of calcium fluoride, or probably even as hydrofluoric acid directly.

TABLE 1: COMPOSITION OF FLUORITE SAMPLE, MOINA 1*

Component	Percent by Weight
CaF_2	25.2
CaO	5.1
$CaCO_3$	1.1
$MgCO_3$	0.2
Fe_2CO_3	0.7
Fe_2O_3	29.9
SiO_2	20.0
TiO_2	0.1
Al_2O_3	10.2
Zn	700 ppm
Pb	15 ppm
Cu	115 ppm
Bi	605 ppm
Mo	50 ppm
W	1270 ppm
Sb	55 ppm
Au	<0.1 ppm
Sub-total	92.78%
L.O.I	~ 7%

*Analysis by Comalco Limited

TABLE 2: COMPOSITION OF ALUMINIUM FLUORIDE PRODUCTS, BY SEMI-QUANTITATIVE EMISSION SPECTROSCOPY

Component	Percent by Weight		Typical maximum limit specified
	S1	S2	
Na	0.5	0.3	0.4
Si	0.05	0.03	0.25
Mg	0.03	0.03	-
Fe	0.02	0.03	0.04
Cu	0.003	0.05	-
Pb	0.001	0.001	-
SO_4	nd*	0.3	0.75

* not determined

The following elements were below the limit of detection (shown in brackets as ppm) in both samples:

As (50), Bi (1), Cd (3), Ga (1), Ge (1), In (10), Sb (30), Sn (1), Zn (20), Ca (100), U (5000).

EXTRACT FROM "PRODUCTION OF HYDROGEN FLUORIDE" BY J.M. Clayton (1977). AMDEL REPORT NO 1178

2.3.3 Ammonium Sulphate Roasting

A schematic flow sheet is given in Fig. 5.

Run-of-mine fluorite ore is recovered from stockpile, crushed, and finely ground in a dry mill. Together with excess ammonium sulphate, the ground ore is fed to an externally heated rotary kiln. The following reactions occur at a temperature of 400°C during a residence time of 2 h.

$$(NH_4)_2SO_4 \rightarrow NH_4HSO_4 + NH_3$$

$$NH_4HSO_4 + CaF_2 \rightarrow NH_4HF_2 + CaSO_4$$

$$4NH_4HF_2 + SiO_2 \rightarrow SiF_4 + 4NH_4F + 2H_2O$$

$$SiF_4 + 2NH_4F \rightarrow (NH_4)_2SiF_6$$

With the exception of gypsum, gangue and unreacted ore, all the reaction products are gaseous at 400°C and are removed from the kiln by admission of a purge gas. The hot gases pass to an absorber which operates at neutral or slightly acid pH to absorb the reaction gases. Purge gas and unabsorbed reaction gases are vented to atmosphere via a scrubber. The absorber liquor passes to a stock tank in which the liquor pH is raised to pH 9 by addition of ammonia. Silica is precipitated via the following reaction.

$$(NH_4)_2SiF_6 + 4NH_4OH \rightarrow 6NH_4F + SiO_2 + 2H_2O$$

The precipitated silica is removed on a filter, and the clean liquor passes to a stock tank. Liquor from the stock tank is pumped to the crude HF still, together with sulphuric acid. The following reaction liberates HF which is distilled as an azeotrope.

$$2NH_4F + H_2SO_4 \rightarrow (NH_4)_2SO_4 + H_2F_2$$

The distilled gases are condensed and fed to the AHF still, in which anhydrous HF is recovered by distillation over concentrated H_2SO_4. Non-condensible gases are vented to atmosphere via a lime scrubber. The bottom liquor from the crude HF still consists of $(NH_4)_2SO_4$ solution which is pumped to a crystalliser for recovery and recycle to the kiln.

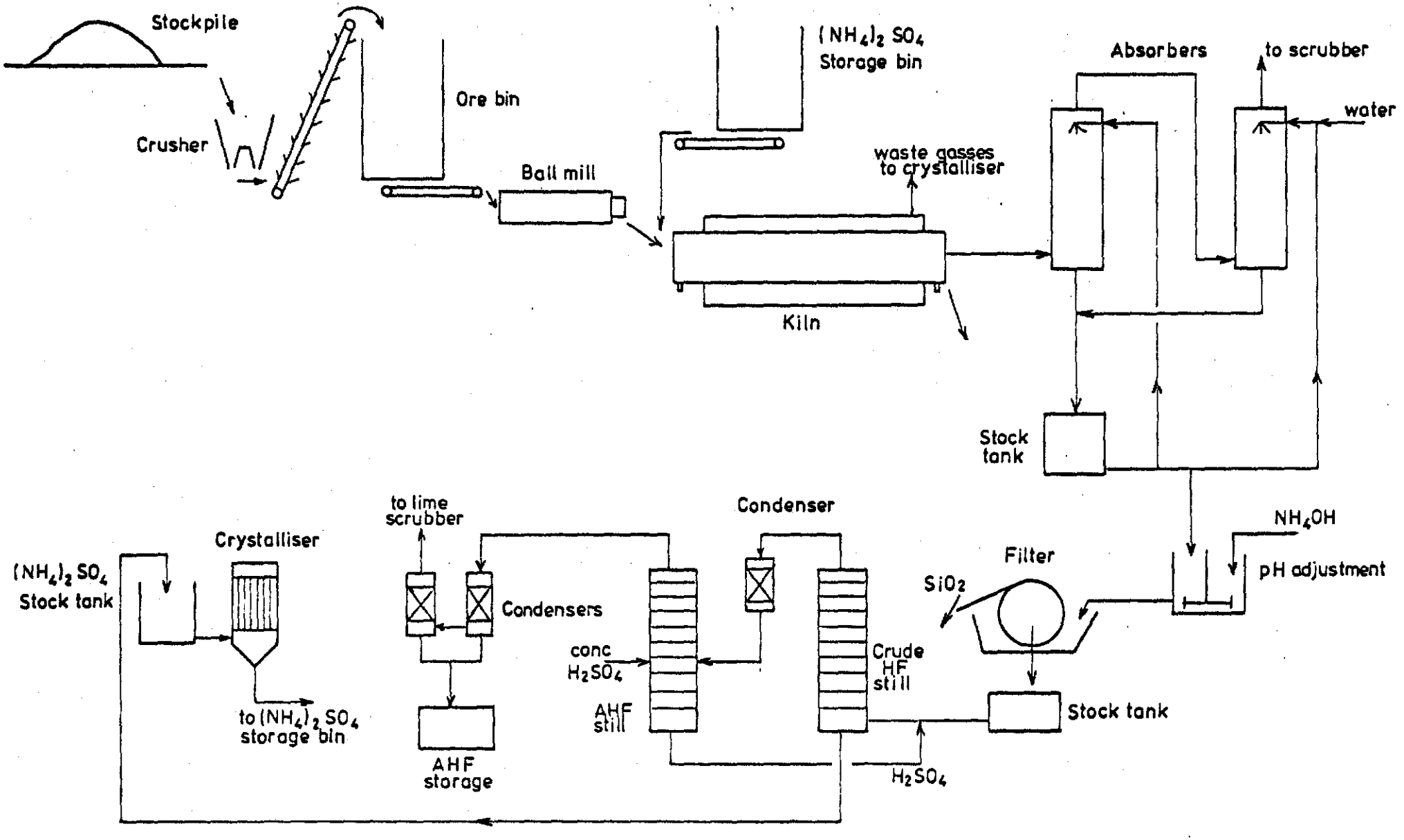

FIG. 5: SCHEMATIC FLOW DIAGRAM – AMMONIUM SULPHATE ROASTING

221475

APPENDIX 29.

AMDEL. PHYSICAL EXTRACTION INVESTIGATIONS OF MOINA WRIGGLITE.

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EL 7/74 MOINA DEPOSIT

221476

amdel
The Australian Mineral Development Laboratories

Flemington Street, Frewville, South Australia 5063
Phone Adelaide 79 1662, telex AA82520

Please address all correspondence to Frewville.
In reply quote: ME 3/1/4/0

15 December 1975

Mr A.H. Bartlett,
Divisional Manager - Exploration,
Comalco Limited,
95 Collins Street,
MELBOURNE. Vic. 3000

Dear Mr Bartlett,

Moina Fluorite

During his visit to your office on 28 October our Dr K.J. Henley discussed with you our Report MP887/76 in which the mineralogy of two specimens of fluorite ore from the Moina deposit was described.

It is evident that the Moina Fluorite is similar to the Lost River and Mount Garnet fluorites in that the fluorite is of very fine grain size, requiring grinding to 40 μm for good liberation.

You will recall that we carried out a flotation investigation on Mount Garnet fluorite in 1973 (Project 3/1/4/12). In this work we were unable to match the results quoted for the Lost River fluorite and concluded that this was due to the fact that Mt Garnet fluorite contained gangue inclusions which were too fine to liberate and which were present in sufficient quantity to prevent the attainment of an acid grade product. Such gangue inclusions were not evident in the Lost River Fluorite.

Taking into consideration the gangue inclusions within the Mr Garnet fluorite it is considered that the results which we obtained were extremely good (Test AM25):

	CaF_2 %	
	Cum. Assay	Cum. Recovery
5th Cleaner Conc.	95.9	53.5
2nd Cl Conc. + Mags	84.8	80.1
(Feed)	(33.3)	

The mineralogical report indicates that there are no gangue inclusions within the Moina fluorite and for this reason we consider that Moina fluorite will respond to flotation more readily than the Mount Garnet fluorite.

You indicated to Dr Henley that you were not interested in undertaking an extensive metallurgical investigation on Moina fluorite because of the possibility that it would perform no better than the Mount Garnet material. However, for the reason stated above, we expect that the

To: Comalco Limited 15 December 1975

Moina fluorite will perform better than Mount Garnet and it should be feasible to demonstrate this with only a few flotation tests.

We suggest that a very limited metallurgical investigation be undertaken as follows:

1. Apply the Lost River flotation process -
   (a) Feed grind 85% - 325 mesh
   (b) Conditioning with reagents
   (c) Rougher, cleaner flotation
   (d) Hot conditioning (60-70°C)
   (e) Further eight cleaning stages
   (f) Magnetic separation of flotation product.

2. Apply the flotation process developed at Amdel for Mt Garnet fluorite -
   (a) Feed grind 85% - 325 mesh with sodium silicate, KCN, soda ash added to the mill. Quebracho can be added also if the ore contains carbonate.
   (b) Flotation with collector AP845, rougher and three cleaner stages. Rougher flotation at 40°C.
   (c) Regrind of third cleaner concentrate.
   (d) Two further cleaner stages.
   (e) Magnetic separation of flotation product.

The reagent types and quantities for the Lost River process are not known, but because voluminous froths are produced we assume that the collector is oleic acid or a tall oil with low content of rosin acids.

We envisage that three flotation tests would be required to test the Lost River process and establish appropriate reagent levels. Two flotation tests should be sufficient to test the Mount Garnet process.

The estimated cost of carrying out the suggested investigation is $1850. If you prefer to test only the Lost River process at this stage, the estimated cost will be $1325.

The estimated fee is subject to any mutually agreed changes in the work programme. The project will be carried out under the standard Conditions of Contract which are included and form part of this Proposal.

Our usual terms of payment are 30 days nett with invoices being submitted at monthly intervals for work performed.

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To: Comalco Limited 15 December 1975

We wish to point out that the project may be eligible for assistance from the Australian Industrial Research and Development Grants Board and Amdel is an approved research organisation according to the Board's regulations. However, as Amdel is not an agent for the Board, applications should be made directly to:

The Executive Officer,
Australian Industrial Research and Development Grants Board,
PO Box 4009,
CANBERRA. ACT 2601

As you are aware, all investigations at Amdel are strictly client confidential. However, provided the client consents it is our practice to inform Amdel Council of the client's name and the project title and value. Beyond that disclosure the project and its results and recommendations remain strictly confidential.

A 10 kg sample of typical Moina fluorite ore should be supplied for the investigation.

Our Mr W.G. Rogers was responsible for the Mount Garnet investigation and will be available to undertake the proposed Moina work. Mr Rogers will be on leave from 25 December 1975 to 15 January 1976. Provided your official authorisation and the required sample have been received by 18 January, work can begin on that date.

The proposed work will require approximately five weeks to complete.

We look forward to receiving your comments on the above proposals.

Yours faithfully,
The Australian Mineral Development Laboratories

for F.R. Hartley
Director.

IMcP:cah
Encl.

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THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES

ADELAIDE, SOUTH AUSTRALIA

CONDITIONS OF CONTRACT

1. **Co-operation between Amdel and Client.** Subject to the terms and conditions of this Agreement, all information obtained by Amdel through work on the project will be made available to the Client during normal hours, and Amdel will communicate to the Client all information pertinent to the project as it progresses. An authorised representative of the Client may confer with the appropriate members of the Amdel staff at mutually agreed times for the purpose of providing such assistance as will make it possible for Amdel to achieve the objectives of the project.
   Amdel welcomes the co-operation of the Client in providing such information, assistance and co-operation as is necessary for Amdel to achieve the objectives of the project.

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3. **Reports.** Amdel agrees to supply the client with copies of reports of its findings and progress during the term of the project at monthly or such other intervals as Amdel and the Client agree.

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c.c. W. H. Andrews
J. C. Nixon
M. R. Rayner
Cairns Office
Devonport Office

EL 7/74 221480
MOINA BENEFICIATION

12th July 1976

Mr. F. R. Hartley,
Director,
Australian Mineral Development
Laboratories,
Flemington Street,
FREWVILLE, S.A. 5063

Dear Sir,

Beneficiation/Treatment of Fluorite Ore From Moina, Tasmania.

Following the completion of the investigation into the chemical extraction of fluoride from Moina fluorite ore by J. E. A. Gooden, R. J. and D. J. Ellis - Report: CM 2873/76, we believe the time is now opportune to consider the procedure for more thorough and detailed beneficiation/treatment studies aimed at producing saleable products.

Preliminary drilling has been completed at Moina, this had "indicated" in excess of ten million tonnes of opencuttable fluorite ore, which also contains: tin, tungsten and magnetite. Before spending more money on exploration drilling, it is essential to determine whether or not this fluorite ore can be beneficiated/treated to recover economic quantities of:-

(i) acid grade fluorspar
(ii) metallurgical grade fluorspar
(iii) or, if chemically treated -
fluorine compounds such as aluminium fluoride (AlF_3)
and hydro-fluoric acid (HF)
(iv) tin concentrate
(v) tungsten concentrate
(vi) magnetite

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Note: The recovery of 75% of the fluorine and at least 50% of the tin contained in the ore is essential for the viability of this project.

Chemical Treatment of Moina Fluorite Ore

Reference your: Letter CM 570/2502, (3/1/4/0) dated 9th December, 1975, "Chemical Treatment of Fluorite Ore" and your subsequent report: CM 2873/76, (3/1/4/0) dated 6th July, 1976, "Fluoride Extraction from Fluorite".

From your report CM 2873/76 and from discussions with Bob Allen we understand the "AMDEL process" involves:-

(i) fine grinding of the fluorite ore to pass 100 mesh BSS sieve.
(ii) mixing ore with 100% reagent (1:1, 12.5 g. of each)
(iii) three (3) experiments roasting the ore and reagent mixture for 1, 2 and 3 hours at about 400°C.
(iv) in all three experiments about 56% of the fluoride was extracted, which was converted to aluminium fluoride (AlF_3) after a silica removal step
(v) next 300 g. of ore was mixed with 400 g. reagent (ratio 3:4), roasted for 2 hours, which achieved 70% extraction of fluoride.
(vi) the fluoride can (we understand) be converted to aluminium fluoride (AlF_3), hydro-fluoric acid (HF) or pure fluorite (CaF_2).

The viability of the 'chemical treatment' of Moina fluorite ore will depend on costs and in particular:

- cost of the reagent
- amount of reagent needed to extract not less than 75% of fluoride.
- amount of reagent consumed, that is not recoverable for re-use in the process.
- percentage of contained tin recoverable from residue.

Before proceding further with this particular method of treatment we will need to know full details of the process. We will be prepared to sign an agreement to maintain confidentiality.

Beneficiation of Moina Fluorite Ore by Flotation

Reference your: Letter ME 3/1/4/0 dated 15th December, 1975, "Moina Fluorite", also your Report MP 887/76 dated 20th October, 1975, "Examination of Metasomatic Fluorite Bearing Rocks" in particular sample - M2/200/4. In addition to consideration of the 'chemical

cont'd 3

treatment' of Moina fluorite ore, we desire that laboratory work should commence shortly to study the flotation of fluorite and recovery of tin, tungsten and magnetite concentrates.

We agree with you that the work on the "Mount Garnet fluorite ore" (Project 3/1/4/12) was good and that considering the presence of gangue inclusions, caused by the presence of the mineral Cuspidine ($3CaO.2SiO_2.CaF_2$) within the "Mount Garnet fluorite ore", the results in Test A.M.25 were extremely good. Since the "Moina fluorite ore" does not contain Cuspidine it should respond more readily than "Mount Garnet fluorite ore". Also, we now know that the gangue in the "Lost River" fluorite ore at Alaska is mainly calcite, therefore, care must be taken in applying what we know about the Lost River beneficiation methods to Moina fluorite ore which has mainly silicates and magnetite as gangue. In theory silicates should be more easily separated from fluorite by flotation than calcite.

Other Work

To assist you with consideration of a suitable work programme we enclose:-

Report: CMS 75/5 16 dated 3rd June, 1976
by H. W. Fander
Central Mineralogical Services Pty. Ltd.
- in which he examined Mineralogically/Petrologically rock chips from our Samples Moina 3, 4, 5 and 6.

Programme

We would be grateful if you would consider our requirements carefully and prepare a preliminary programme for laboratory beneficiation/ treatment studies so that we can hold discussions with the aim of reaching agreement on:

(i) the programme for laboratory beneficiation/treatment studies
(ii) timetable for (i)
(iii) cost of carrying out (i)
(iv) plans for a pilot plant study

Correspondence, Reports etc.

The originals of all letters, reports, quotes, invoices etc. should be sent to myself for action so that the project may be co-ordinated.

Yours sincerely,

A. H. Bartlett,
Manager - Exploration.

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The Australian Mineral Development Laboratories

Flemington Street, Frewville; South Australia 5063
Phone Adelaide 79 1662, telex AA82520

Winner of Award for Outstanding Export Achievement, 1975

Please address all correspondence to Frewville.
In reply quote: CM 5/0/2862

6 August 1976

The Manager Exploration
Comalco Limited
GPO Box 2773Y
MELBOURNE Vic 3001

Dear Mr Bartlett,

Treatment of Moina Fluorite
Project Proposal No. 5/0/2862

Thank you for your letter of 12 July 1976. In response, we are pleased to submit for your consideration the attached project proposal.

The investigation described in Amdel Service Report No. CM 2873/76 showed that the iron content of the Moina fluorite ore was extensively attacked under the conditions used to extract the fluoride content. This would represent a major wastage of reagent, unless a recovery stage was incorporated to decompose the iron salt.

From this view point, because recovery of a magnetite concentrate is desired, and also because it is economically advantageous to have a feed material of higher CaF_2 content for chemical processing, we consider that physical beneficiation is an essential precursor to chemical treatment.

Minerals of economic interest are fluorite, scheelite, cassiterite and magnetite. Although it may be possible to beneficiate the ore by physical methods alone, separation of fluorite from scheelite may be difficult.

Accordingly, we propose aiming to produce a rougher flotation concentrate containing fluorite and scheelite for chemical processing.

Preliminary mineralogical examination carried out by Amdel and Central Mineralogical Services Pty Limited has shown that the mineral texture is extremely fine and grinding to about 30 µm will be required to obtain adequate liberation.

Because of the complexity of the ore and the requirement to make five marketable products if possible it is not possible to specify in detail the flotation beneficiation programme. It is therefore proposed that the initial oredressing investigation, Part A, be confined to the production of a bulk fluorite/scheelite concentrate for chemical extraction, plus preliminary scouting tests to indicate the feasibility of producing separate fluorite, scheelite and cassiterite concentrates by flotation. Magnetic separation would be examined as a means of producing a magnetite concentrate and for removing iron from flotation concentrates where required.

To: Comalco Limited, 6 August 1976

If warranted a more detailed beneficiation study could be undertaken as a second stage based on the results of the initial investigation.

Investigations of the chemical treatment process, Part B, would be confined to the best concentrate, at least one kg of which would be produced in Part A. Aspects to be examined experimentally in Part B would be:

(a) variation in the fluoride extraction efficiency with the proportion of reagent added

(b) effect of programmed heating on fluoride extraction at a given level of reagent addition

(c) overall reagent consumption.

Part C would comprise a preliminary estimate of capital and operating costs for the chemical treatment process.

Parts B and C of the technical programme would not be proceeded with unless warranted by the preceding results.

The fees for the proposed investigation are as follows:

Part A:	$9,700
Part B:	$5,900
Part C:	$3,800

The estimated fee is subject to any mutually agreed changes in the work programme. The project will be carried out under the standard Conditions of Contract which are included and form part of this Proposal.

Our usual terms of payment are 30 days nett with invoices being submitted at monthly intervals for work performed.

As you are aware, all investigations at Amdel are strictly client confidential. However, provided the client consents it is our practice to inform Amdel Council of the client's name and the project title and value. Beyond that disclosure the project and its results and recommendations remain strictly confidential.

We will be pleased to discuss this proposal and the chemical treatment process, in detail at the meeting suggested in your letter of 12 July 1976.

Yours faithfully,
The Australian Mineral Development Laboratories

R L Walmsley

for F.R. Hartley,
Director

RJA:rp

221485

PROJECT PROPOSAL NO. 5/0/2862

CLIENT: COMALCO LIMITED

PROPOSED TITLE: TREATMENT OF MOINA FLUORITE

AUGUST 1976

THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES

Adelaide South Australia

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THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES

TREATMENT OF MOINA FLUORITE

A1 INTRODUCTION

Comalco Limited's drilling programme has indicated large reserves (~10 million tonnes) of fluorite ore in their leases at Moina in Tasmania. Exploratory tests, described in Amdel Service Report No. CM 2873/76, showed that the fluoride content of this ore should be efficiently extracted by a chemical treatment process under development at Amdel, provided sufficient reagent is added. By further processing the extracted fluoride was converted to aluminium fluoride of acceptable grade for the electrolytic production of aluminium.

In a letter dated 12 July 1976, Mr A.H. Bartlett, Exploration Manager of Comalco Limited, requested a proposal for an experimental programme aimed at developing an overall scheme for recovering values from the ore, involving physical beneficiation and chemical treatment.

A2 OBJECTIVES

The objectives of this Proposal are listed below:

Part A To examine the effectiveness of flotation and magnetic separation procedures in producing the following:

(1) fluorspar concentrate - acid grade (97% CaF_2), metallurgical grade (70% effective CaF_2) or a lower grade concentrate suitable for chemical treatment

(2) tin concentrate

(3) tungsten concentrate

(4) magnetite concentrate.

Part B To study further the application of the Amdel chemical treatment process to a fluorspar concentrate from Part A, with particular reference to:

(1) the effect of reagent addition on fluoride extraction

(2) the effect of programmed heating on fluoride extraction

(3) the overall reagent consumption

Part C To make a first estimate of capital and operating costs for the chemical treatment process.

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THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES

TREATMENT OF MOINA FLUORITE

A3 SCOPE OF TECHNICAL PROGRAMME

The work proposed covers:

Part A: Laboratory-scale flotation tests, Davis tube magnetite separation experiments

Part B: Laboratory-scale roasting tests, with chemical analysis of all products

Part C: Brief economic study, comprising a preliminary estimate of capital and operating costs for the chemical treatment process.

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THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES

TREATMENT OF MOINA FLUORITE

B1 PROPOSED TECHNICAL PROGRAMME

Part A (1) Mineralogical Examination

The objectives of the mineralogical work will be:

a. To confirm that the ore sample has similar mineralogical composition and characteristics to the ore specimens previously examined and to note any variations which may occur.

b. To determine liberation characteristics of the economic minerals.

Mineralogical examination of test products may also be undertaken as required to aid evaluation of beneficiation processes.

(2) Magnetic Separation

Magnetic separation will be used for the production of a magnetite concentrate. It will also be used as required to remove iron from flotation concentrates.

Magnetic separation will be investigated initially on ground flotation feed. Subsequently magnetic separation may be applied to flotation products in addition to or as an alternative to primary magnetic separation.

(3) Flotation

Initial flotation testing will examine the feasibility of producing a bulk fluorite/scheelite concentrate. An initial sulphide float would be carried out followed by rougher flotation of the non-sulphides to produce the fluorite/scheelite concentrate. It is anticipated that magnetite and cassiterite will report to the flotation tailing.

A limited study will be made of the flotation cleaning of the rougher fluorite/scheelite concentrate to establish the grade-recovery relationship. From the results of this work appropriate conditions will be selected for producing 1 kg of fluorite/scheelite concentrate for leaching studies.

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THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES

TREATMENT OF MOINA FLUORITE

B1 PROPOSED TECHNICAL PROGRAMME (Continued)

A preliminary study will also be made of the feasibility of:

(a) Flotation of cassiterite from the primary fluorite/scheelite flotation residue

(b) Flotation of fluorite from the bulk fluorite/scheelite concentrate.

Part B (1) Three laboratory tube furnace roasting experiments with chemical analysis of all products.

(2) Two laboratory tube furnace roasting experiments with chemical analysis of all products.

(3) Two larger scale roasting experiments (300 g concentrate) including silica removal, aluminium fluoride precipitation and calcination stages; with chemical analysis of product for several selected elements.

Part C (1) Selection, sizing and costing of equipment for a specified annual throughput of fluorspar concentrate.

(2) Calculation of operating costs, based on the cost of reagents, capital charges, labour, etc.

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THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES

TREATMENT OF MOINA FLUORITE

D1 SAMPLE REQUIREMENT

25 kg of representative ore should be sufficient for the investigation.

Sample advice notes are enclosed. Please forward with or in advance of the samples.

Samples should be packed in strong containers and labelled inside and outside with the Proposal number shown on the front of this document.

Samples should be consigned to "Amdel, Mile End" if being forwarded by rail or to "Amdel, Osman Place, Thebarton, South Australia" if being sent by road.

Samples will be held for three months after the issue of the final report when unless advice is received to the contrary, they will be destroyed.

D2 COMMENCEMENT DATE

Within one week of receipt of sample.

D3 ANTICIPATED DATE OF COMPLETION

4 months from commencement.

D4 AMDEL OFFICERS FOR CLIENT CONTACT ON TECHNICAL MATTERS

I.W. McPheat - Beneficiation studies
R.J. Allen - Chemical investigations.

amdel

THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES

TREATMENT OF MOINA FLUORITE

E1 QUOTATION

Part A - $9,700 Part B - $5,900 Part C - $3,800	$19,400

E2 BASIS OF SUBMISSION

1. The Proposal is an offer until 1 October 1976 and if the authorisation to proceed with the work as defined in the Proposal is received at Amdel on or before the said date it shall become a contract between the parties subject to any variations in the technical programme and the terms of payment listed in the body of the attached letter. If not accepted by the due date, the Proposal is subject to confirmation.

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THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES

ADELAIDE, SOUTH AUSTRALIA

CONDITIONS OF CONTRACT

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16. **Arbitration.** Any dispute which may arise as to the interpretation of these conditions shall be referred to an arbitrator to be appointed by agreement of both parties or, in default of agreement, by the Auditor-General for the time being of the State of South Australia and such arbitrator shall act pursuant to the provisions of the Arbitration Act, 1891-1935.

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c.c. W. H. Andrews
J. C. Evans
J. C. Nixon
Cairns Office
Devonport Office

221494

20th August 1976

The Director,
Australian Mineral Development
Laboratories,
Flemington Street,
FREWVILLE, S.A. 5065

Dear Sir,

Beneficiation/Treatment of Moina Fluorite
Project No. 5/0/2862

Further to our letter dated the 12th July, 1976 and your response in the form of project proposal No. 5/0/2862 dated 6th August, 1976, we wish you to proceed with "Part A" of the proposed investigation at an estimated fee of $9,700. Consideration will be given to "Parts B and C" when the physical beneficiation - Part A, is nearing its completion.

We would wish you to start this project as soon as possible and keep us regularly informed. Four (4) copies of all reports will be required and these should be sent to:-

W. H. Andrews: Manager - Technical Services

A. H. Bartlett: Manager - Exploration

Dr. J. C. Nixon: Corporate Manager - Research

Comalco Limited,
G.P.O. Box 2773Y,
Melbourne, Vic. 3001

cont'd 2

221495

P. W. Askins Senior Geologist
Comalco - Exploration Office,
P. O. Box 691,
Devonport, Tas. 7310

Although you only need a 25 Kg. representative sample of the Moina fluorite ore, we recommend that a 50 Kg. sample be prepared from 148 of the samples of bore core presently held in your Analytical Section. We wish the sample for this project to be prepared according to the following recipe:-

Bore No.	From metres	To metres	Interval metres	No. of core samples
SMD 4	68.37	95.90	27.53	27
SMD 5	39.80	62.00	22.20	21
SMD 6	21.35	85.00	63.65	60
SMD 7	0	48.55	48.55	40
			161.93	148

To obtain a 50 Kg. "weighted" sample it will be necessary to split/ weigh out:

* 308.7 grams X sample core length for each of the 148 samples to be used.

(* 50,000 ÷ 161.93 = 308.7)

Copies of the tabulated mineral analysis data for the relevant parts of bores SMD 4, 5, 6 and 7 are attached for your information. Sample core lengths have been written in.

Thank you for your co-operation in preparing the proposal and we look forward to a successful outcome of this project.

Yours faithfully,

A. H. Bartlett,
Manager - Exploration.

Enclosures
JMS

c.c. D. K. Rowley Officer-in-Charge,
Analytical Chemistry Section

I. W. McPheat,
Mineral Engineering Section,
Amdel, Osman Place, Thebarton, S.A. 5031

COMALCO EXPLORATION DEPARTMENT

TASMANIAN E.L. 7/74 "MOINA", SHEPHERD & MURPHY MINE AREA

Mineral analysis data from bore core samples

Bore No.	Depth Metres From	To	INTERVAL METRES.	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	Au ppm	Remarks
SMD 4	68.37	69.00	0.63 metres	6	540	580	1150	10	130	1240	40	15.6	-1	0.20	SMD. 4
4	69.00	70.00	1.0	10	820	620	1200	22	150	130	100	21.0	-1	0.20	27 samples
4	70.00	70.70	0.7	18	270	340	2200	-4	10	90	50	9.6	-1	0.15	over 27.53 metres
4	70.70	71.30	0.6	14	70	190	1900	6	5	90	50	8.5	-1	0.15	
4	71.30	72.00	0.7	8	300	330	1600	12	50	180	40	12.5	-1	0.15	
4	72.00	73.00	1.0	4	640	620	1700	38	200	140	50	21.0	-1	0.20	
4	73.00	74.00	1.0	8	1000	780	1550	30	160	130	80	26.0	-1	0.15	
4	74.00	75.00	1.0	16	1750	600	1800	100	240	130	80	26.5	-1	0.20	To be taken
4	75.00	76.00	1.0	-4	1750	350	1700	95	150	120	80	24.0	-1	0.15	wt. 50 Kg.
4	76.00	77.00	1.0	12	520	540	1650	24	130	100	80	20.0	-1	0.20	
4	77.00	78.00	1.0	8	620	450	1250	55	120	120	50	16.5	-1	0.20	
4	78.00	79.00	1.0	10	1450	580	1400	75	150	90	80	20.0	-1	0.20	composite sample
4	79.00	80.00	1.0	10	760	300	2100	65	32	80	80	17.5	-1	0.10	
4	80.00	80.63	0.63	22	840	470	1400	90	250	80	80	20.0	-1	0.10	composed of
4	80.63	81.66	1.03	12	600	6	220	36	10	32	20	4.1	-1	-0.05	
4	81.66	83.70	2.04	20	460	320	1000	26	25	50	50	7.4	-1	-0.05	308.7 gms per
4	83.70	85.00	1.3	10	1600	270	1300	60	350	70	40	16.5	-1	-0.10	
4	85.00	86.00	1.0	-4	1500	320	1500	75	220	65	30	15.5	-1	0.10	metre of core
4	86.00	87.00	1.0	14	2200	410	1250	75	240	70	40	14.0	-1	0.05	
4	87.00	88.00	1.0	14	860	660	1100	32	350	190	40	12.5	-1	0.10	length.
4	88.00	89.00	1.0	10	700	350	980	55	210	310	40	9.9	-1	0.10	
4	89.00	90.60	1.6	10	390	230	1150	42	160	95	30	7.5	-1	0.10	
4	90.60	91.25	0.65	18	180	60	1450	8	48	100	80	4.4	-1	-0.05	
4	91.25	92.45	1.20	24	300	75	1150	150	15	110	30	5.8	-1	0.05	
4	92.45	93.80	1.35	18	370	80	980	-4	60	110	30	5.1	-1	-0.05	
4	93.80	94.40	0.6	14	140	14	1050	6	65	60	20	4.5	-1	0.05	
4	94.40	95.90	1.5	12	1300	200	1200	26	80	70	30	8.1	-1	0.10	
4	95.90	97.15		6	400	6	280	10	45	42	20	4.9	-1	-0.05	
4	97.15	98.40		10	270	-4	75	20	40	40	5	1.9	-1	-0.05	

221496

P.L. 7/74 "Moina"

Mineral analysis data from bore core samples

Bore No.	Depth From	Depth To	INTERVAL METRES.	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	Au ppm	Remarks
SMD 5	38.10	39.80		34	-10	16	80	-4	28	170	1	0.45			
5	39.80	40.67	0.87 metres	6	270	380	660	4	200	1740	40	10.5	-1	0.30	
5	40.67	42.00	1.33	12	490	580	1350	10	75	95	40	17.0			
5	42.00	43.00	1.0	14	740	700	860	16	210	100	50	16.5			
5	43.00	44.00	1.0	10	1750	320	1400	14	320	70	30	18.0	-1	0.15	
5	44.00	45.00	1.0	12	680	620	1050	34	170	65	30	17.0			
5	45.00	46.60	1.6	24	900	1100	1500	44	70	70	30	14.0			
5	46.60	47.50	0.9	10	170	44	250	-4	22	70	10	4.5			
5	47.50	48.85	1.35	6	480	28	500	-4	12	90	20	4.7	-1	0.05	
5	48.85	50.00	1.15	14	490	80	1100	-4	10	85	80	15.0			
5	50.00	51.30	1.3	65	540	820	1250	16	200	110	30	13.5			
5	51.30	52.80	1.5	20	780	760	940	8	150	200	40	12.5			
5	52.80	53.30	0.5	12	430	65	760		430	50	40	12.5	-1	0.10	
5	53.30	54.50	1.2	18	380	140	720	16	75	60	40	9.8			
5	54.50	55.90	0.4	20	620	160	880	22	22	65	40	11.0			
5	55.90	56.35	0.45	12	760	10	320	22	20	30	20	4.5			
5	56.35	57.45	1.1	44	110	700	540	36	15	40	20	2.5			
5	57.45	57.90	0.45	12	3350	34	740	18	85	150	40	9.9			
5	57.90	58.70	0.8	8	780	180	820	38	140	90	40	10.5			
5	58.70	60.00	1.3	12	700	270	1250	34	60	60	30	15.8			
5	60.00	60.85	0.85	10	560	220	1250	14	190	75	40	14.5			
5	60.85	62.00	1.15	6	1050	8	3800	120	42	48	20	11.0			
5	62.00	72.90		6	470	6	390	30	5	48	20	5.3			
5	72.90			8	170	24	75	4			10	2.4			
DETECTION LIMIT				(4)	(10)	(4)	(4)	(4)			(1)		(1)	(0.05)	
METHOD OF ANALYSIS				X-RAY FLUORESCENCE SCHEME B1					CODE C1				CODE C2	CODE C3	

SMD. 5.

21 core samples over 22.2 metres

To be part of 50Kg. composite sample composed of 308.7 gms per metre of core length.

E.L. 7/74 "Moina"

Mineral analysis data from bore core samples

Bore No.	Depth Metres From	To	Interval Metres	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	Au ppm	Remarks
SMD 6	21.35	22.75	1.4 metres	8	130	360	1600	-4	150	270	40	18.0			↑
6	22.75	23.50	0.75	14	580	640	1850	-4	170	310	30	19.0			S.M.D. 6.
6	23.50	24.75	1.25	6	450	310	2000	-4	35	120	40	17.5			60 core samples
6	24.75	26.15	1.4	12	640	400	1950	-4	95	100	40	19.4			over 63.65 metres.
6	26.15	28.00	1.85	14	540	290	1050	-4	15	90	30	8.4			
6	28.00	29.15	1.15	10	720	440	2050	-4	28	130	30	16.0			To be part of
6	29.15	29.50	0.35	10	720	360	1250	6	38	130	20	28.5			50 Kg. composite
6	29.50	30.80	1.3	12	410	560	1950	-4	50	120	30	19.5			sample compound
6	30.80	32.15	1.35	14	230	290	760	6	12	50	50	8.6			of 308.7 gms
6	32.15	33.90	1.75	14	1050	880	1950	8	190	120	30	21.5			per metre of
6	33.90	34.50	0.6	16	380	390	1750	-4	10	42	30	9.2			core length.
6	34.50	35.50	1.0	18	660	660	2100	-4	15	100	40	20.0			
6	35.50	36.40	0.9	12	700	880	3350	10	35	110	40	16.0			
6	36.40	38.00	1.6	12	480	640	2050	6	32	100	40	15.5			
6	38.00	39.00	1.0	20	640	210	2550	12	30	100	60	15.0			
6	39.00	40.00	1.0	12	860	900	1700	90	90	90	80	24.5			
6	40.00	41.00	1.0	12	1550	400	2700	80	35	60	40	21.5			
6	41.00	42.00	1.0	4	2650	680	1150	160	160	100	40	30.0			
6	42.00	43.00	1.0	6	4600	600	1300	110	50	120	40	30.0			
6	43.00	44.00	1.0	10	860	660	1200	75	48	140	30	24.5			
6	44.00	45.00	1.0	10	1100	470	1500	90	18	140	50	24.5			
6	45.00	46.00	1.0	12	600	700	1100	16	18	120	80	23.0			
6	46.00	47.00	1.0	6	980	700	1400	-4	15	150	30	20.5			
6	47.00	48.00	1.0	6	740	600	1100	16	18	160	30	20.5			
6	48.00	49.00	1.0	-4	940	600	1150	170	8	160	40	25.0			
6	49.00	50.00	1.0	8	1050	680	1050	14	18	160	40	24.5			
6	50.00	51.00	1.0	4	740	600	1050	24	32	220	30	15.5			
6	51.00	52.00	1.0	6	880	470	900	4	8	170	30	17.0			
6	52.00	53.00	1.0	-4	860	620	1000	-4	10	200	30	17.5			
6	53.00	54.00	1.0	12	840	440	1250	10	85	200	50	17.5			
6	54.00	55.00	1.0	-4	1000	390	1200	26	28	140	40	19.0			
6	55.00	56.00	1.0	10	1750	600	1650	34	85	110	30	20.5			
6	56.00	57.00	1.0	6	700	330	1700	42	28	110	30	16.0			continued

221498

Mineral analysis data [illegible]m bore core samples

Bore No.	Depth Metres From	To	INTERVAL METRES	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be %	CaF_2 ppm	Ag ppm	Au ppm	Remarks
SMD 6	57.00	58.00	1.0 metres	10	980	540	1500		8	100	40	21.5			SMD.6.
6	58.00	59.00	1.0	8	2050	600	1100	46	70	130	60	24.5			
6	59.00	60.00	1.0	12	740	620	1300	30	160	180	30	28.5			
6	60.00	61.00	1.0	18	2050	480	1300	110	38	140	40	28.5			
6	61.00	62.00	1.0	16	1050	450	1550	90	65	140	60	27.5			
6	62.00	63.00	1.0	10	780	450	1350	80	32	130	50	21.0			
6	63.00	64.00	1.0	16	400	410	1150	350	15	120	60	15.0			
6	64.00	65.00	1.0	8	540	600	1600	46	60	130	50	20.0			
6	65.00	66.00	1.0	6	620	340	1500	55	35	130	30	16.0			
6	66.00	67.70	1.7	22	640	370	1550	38	75	120	40	14.0			
6	67.70	69.00	1.3	6	700	12	380	28	8	120	20	7.4			
6	69.00	70.00	1.0	6	1400	30	480	150	8	80	80	9.5			
6	70.00	71.00	1.0	10	360	48	620	24	5	140	20	6.2			
6	71.00	72.00	1.0	10	900	-4	420	75	8	130	60	5.4			
6	72.00	73.00	1.0	12	1300	30	430	24	8	100	30	8.6			
6	73.00	74.00	1.0	14	400	30	520	8	15	70	20	7.8			
6	74.00	75.00	1.0	6	460	140	1300	44	20	75	40	19.0			
6	75.00	76.00	1.0	18	840	1650	1500	40	140	1700	30	19.0			
6	76.00	77.00	1.0	4	960	110	1650	40	10	140	30	16.5			
6	77.00	78.00	1.0	-4	1050	42	1600	30	18	80	30	16.5			
6	78.00	79.00	1.0	8	1100	270	1250	65	120	70	30	19.0			
6	79.00	80.00	1.0	6	1100	110	1350	44	65	80	20	14.0			
6	80.00	81.00	1.0	18	560	320	1350	55	65	4550	30	13.5			
6	81.00	82.00	1.0	75	1050	2100	1250	110	100	470	80	10.5			
6	82.00	83.05	1.05	70	820	2000	960	75	90	170	20	11.0			
6	83.05	84.00	0.95	14	660	110	1000	36	10	130	20	6.5			
6	84.00	85.00	1.0	16	920	190	2050	55	45	110	40	9.0			
6	85.00	85.80		18	350	32	240	55	38	80	20	4.9			
6	85.80	87.25		12	660	4	350	75	8	510	30	4.5			
6	87.25	88.00		38	450	36	240	180	8	75	30	4.9			
6	88.00	89.00		48	1100	8	210	32	8	90	50	5.3			
6	89.00	90.00		38	110	18	200	280	5	75	80	3.7			
6	90.00	91.00		30	1450	28	250	240	5	70	80	6.0			

221499

...7/74 "Molna"

Mineral analysis data fr... bore core samples

Core No.	Depth From	Depth To	Interval Metres	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	Au ppm	Remarks
SMD 7	00.00	01.00	1.0 mtrs	16	880	680	2750	22	38	510	20	10.5			
7	01.00	08.00	7.0	14	800	420	2350	-4	12	280	20	25.5			
7	08.00	09.62	1.62	14	880	640	1400	-4	15	220	30	28.0			
7	09.62	11.00	1.38	4	1350	640	1350	16	28	250	30	28.0			
7	11.00	12.00	1.0	6	780	420	1650	-4	12	270	50	26.5			
7	12.00	13.00	1.0	6	330	390	1900	-4	10	360	50	21.5			
7	13.00	14.00	1.0	8	580	440	1550	4	12	370	50	21.5			
7	14.00	16.00	1.0	10	840	490	1300		20	370	50	23.5			
7	16.00	17.00	1.0	4	290	340	1300	16	10	320	80	17.7			
7	17.00	18.00	1.0	6	660	260	1400	6	2	270	50	19.5			
7	18.00	19.00	1.0	18	1100	320	1550	12	5	130	80	21.5			
7	19.00	20.00	1.0	14	840	540	1600	22	85	370	80	25.0			
7	20.00	21.00	1.0	8	800	640	1500	38	10	270	30	28.0			
7	21.00	21.75	0.75	6	920	470	2200	28	5	570	50	27.0			
7	21.75	22.65	0.9	16	390	130	2400	16	5	120	50	9.9			
7	22.65	24.00	1.35	12	1000	580	1800	20	5	170	80	22.5			
7	24.00	25.00	1.0	12	1550	420	1800	80	120	9800	30	24.5			
7	25.00	26.00	1.0	16	400	940	1300	30	230	740	30	20.5			
7	26.00	27.00	1.0	8	680	920	1500	26	440	450	30	21.5			
7	27.00	28.00	1.0	8	920	580	2500	60	220	410	40	22.5			
7	28.00	29.00	1.0	12	860	460	2250	95	85	310	40	22.0			
7	29.00	30.00	1.0	4	840	520	1700	46	110	1870	30	17.0			
7	30.00	31.00	1.0	8	1850	350	1400	46	38	180	40	23.0			
7	31.00	32.00	1.0	12	500	350	1050	50	110	160	40	23.0			
7	32.00	33.00	1.0	18	2150	390	2250	48	200	190	30	19.5			
7	33.00	34.00	1.0	-4	1300	240	2300	55	8	150	20	21.5			
7	34.00	35.00	1.0	12	720	680	1450	30	32	120	80	23.5			
7	35.00	36.00	1.0	14	940	230	2850	46	40	210	30	14.0			
7	36.00	37.00	1.0	12	1100	300	1800	70	60	110	30	20.5			
7	37.00	38.00	1.0	14	1950	470	880	70	130	110	50	22.0			
7	38.00	39.00	1.0	10	4500	500	1050	230	95	110	40	23.0			
7	39.00	40.00	1.0	4	3050	620	920	290	50	120	30	22.5			
7	40.00	41.00	1.0	8	1400	520	1000	80	55	170	80	21.0			

SMD. 7.

40 core samples over 48.55 mtrs

To be part of 50 Kg. composite sample composed of 308.7 gms per mtr of core length.

continued

221500

Bore No.	Depth Metres From	To	INTERVAL METRES	Pb ppm	W ppm	Bi ppm	Sn ppm	Mo ppm	Cu ppm	Zn ppm	Be ppm	CaF_2 %	Ag ppm	Au ppm	Remarks
SMD 7	41.00	42.00	1.0 metres	6	2300	560	840	55	42	120	30	24.5			SMD.7
7	42.00	43.00	1.0	4	540	460	720	24	60	100	40	25.0			
7	43.00	44.00	1.0	8	1850	340	540	160	35	120	40	19.0			
7	44.00	45.35	1.35	10	660	800	780	26	42	90	40	19.5			
7	45.35	46.15	0.8	12	180	410	1050	6	70	55	20	13.5			
7	46.15	47.30	1.15	10	370	520	980	12	32	90	30	13.5			
7	47.30	48.55	1.25	10	280	640	900	12	75	110	40	15.5			
7	48.55	50.00		20	3400	170	400	55	8	110	30	8.6			
7	50.00	51.00		20	840	42	380	180	8	120	20	5.9			
7	51.00	52.00		30	230	1300	490	14	10	70	50	10.0			
7	52.00	52.75		18	95	2050	420	8	28	85	80	15.0			
7	52.75	54.00		12	290	390	720	-4	18	95	80	16.0			
7	54.00	55.00		18	350	640	720	-4	25	110	80	22.0			
7	55.00	56.00		32	480	180	480	36	8	80	40	4.8			
7	56.00	57.00		28	760	28	420	170	8	75	80	5.3			
7	57.00	58.00		24	110	8	370	42	5	60	30	2.7			
7	58.00	59.00		12	270	12	390	24	8	75	80	3.9			
7	59.00	60.00		18	1200	8	600	140	8	85	80	9.5			
7	60.00	61.00		26	45	6	310	8	8	60	20	5.3			
7	61.00	62.00		26	80	-4	680		8	50	20	3.2			
7	62.00	63.00		26	150	8	740	4	8	70	30	4.4			
7	63.00	64.00		30	160	-4	1150	-4	8	65	30	2.9			
7	64.00	65.00		24	980	4	1000	12	8	65	30	4.0			
7	65.00	65.75		10	200	-4	330	160	5	100	30	7.8			
7	65.75	66.75		14	330	12	660	12	8	95	30	3.4			
7	66.75	67.75		12	480	32	380	42	25	60	20	6.8			
7	67.75	69.00		16	360	10	350	24	5	80	20	3.9			
7	69.00	70.00		-4	420	3	190	42	8	55	20	5.5			
7	70.00			8	270	-4	180	8	10	55	20	4.5			
DETECTION LIMIT				(4)	(10)	(4)	(4)	(4)			(1)		(1)	(0.05)	
METHOD OF ANALYSIS				X-RAY FLUORESCENCE SCHEME B1					CODE C1				CODE C2	CODE C3	

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COMALCO LIMITED

Amdel Report

No. 1147

TREATMENT OF MOINA FLUORITE

by

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SUMMARY

Background

The Moina fluorite deposit of Comalco Limited contains low-grade fluorite mineralisation with magnetite. Minor concentrations of scheelite and cassiterite are present.

Preliminary mineralogical examination has shown that grinding to at least 30 µm is required to liberate the fluorite.

Exploratory leaching tests indicated that chemical extraction of the fluorine was feasible but would require prior elimination of magnetite. Following this work the company authorised a preliminary investigation into the physical beneficiation of the ore.

Objectives

The objectives of the investigation were:

1. To produce an acid grade fluorite concentrate at maximum recovery or a lower-grade concentrate suitable for chemical treatment;
2. To make separate magnetite, cassiterite and scheelite concentrates.

Summary of Work Done

The investigation was carried out using a composite ore sample prepared from 148 drill-core samples representing four diamond drill holes in the Moina deposit.

A detailed mineralogical examination was carried out to determine the nature and association of the minerals present and their liberation characteristics.

Flotation tests were carried out on the raw sample and on material which had been subjected to magnetic separation for removal of magnetite. Magnetic separation of the flotation concentrate was also investigated. Other aspects of flotation examined were regrinding of flotation concentrate before further cleaning and the use of Quebracho to depress calcite.

Conclusions

1. By fine grinding and flotation a concentrate assaying 92.5% CaF_2 with a recovery of 45.7% was produced from a feed assaying 16.5% CaF_2.
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2. Removal of magnetite by magnetic separation applied to either the flotation feed or concentrate resulted in a high loss of fluorite due to the poor liberation characteristics of magnetite.

3. Grinding to 17 μm is necessary for good liberation of fluorite. Scheelite is well liberated at flotation sizes but little cassiterite is liberated above 9 μm. Much of the magnetite is liberated at 33 μm but finely disseminated magnetite is evident in grains finer than 15 μm.

4. The chief contaminants in the flotation concentrate are magnetite, calcite, scheelite and amphibole. Further flotation cleaning stages should reject calcite and amphibole, but the magnetite occurs as fine inclusions in the fluorite. Differential flotation of the scheelite from the fluorite has not been studied.

5. It is unlikely that a high grade magnetite concentrate suitable for heavy medium use could be recovered from the ore because of the poor liberation characteristics of this mineral.

6. The composite ore sample contains 0.13% Sn, 45% of which is present in solid solution in garnet. The remaining tin is present as cassiterite which is too finely disseminated for physical beneficiation.

7. The composite ore sample contains 0.11% WO_3 as scheelite. This scheelite concentrated to 0.4% WO_3 with the fluorite during flotation but it is doubtful whether the scheelite could be recovered economically from this concentrate unless chemical treatment was carried out.

Recommendation

As discussed in Section 6.7 of the Report, it may be feasible to treat the Moina ore by flotation to produce a low-grade concentrate at high recovery and then leach this concentrate to extract fluoride. It is recommended that sufficient work be carried out to enable a preliminary assessment of the technical and economic feasibility of this process be made. A Proposal for an economic evaluation of the proposed treatment dated 6 August 1976 has been forwarded.

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1. INTRODUCTION

Comalco Limited is investigating a fluorite prospect on their leases at Moina in Northern Tasmania. A preliminary drilling programme has indicated substantial tonnages of low-grade fluorite mineralisation with magnetite, garnet and silica as the major gangue minerals. Minor quantities of scheelite and cassiterite are present.

Preliminary mineralogical examination has shown that grinding to at least 30 µm is required to effect substantial liberation of the fluorite.

Exploratory leaching tests, described in Amdel Report CM2873/76, indicated that chemical extraction of the fluorine was feasible but magnetite would need to be largely eliminated before leaching as this mineral consumed the leaching agent.

A Project Proposal (5/0/2862) was submitted to Comalco Limited for an investigation including physical beneficiation, chemical extraction of fluorine and a brief economic evaluation of the chemical process. The company elected to undertake an initial investigation restricted to physical beneficiation.

On the basis of preliminary beneficiation data, the Client assessed that processing of the Moina ore was not economically feasible and the investigation was terminated before the planned programme had been completed.

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2. MATERIAL EXAMINED

A composite sample of ore from the Moina deposit was prepared by combining appropriate samples representing ore intersections from four diamond drill holes within the deposit. A total of 148 samples comprising assay sample balances were included in the composite, the weight of each sample being adjusted to correspond with the footage represented.

An analysis of the composite sample is given below:

	%		ppm
CaF_2	16.5	Bi	450
Fe (acid sol.)	16.7	Mo	45
SiO_2 (total)	30.6	U	4
S (total)	0.33	Au	0.12
P_2O_5	0.06	Hg	0.35
Sn	0.13		
WO_3	0.11		

A qualitative spectrographic analysis is included in Table 1.

A mineralogical examination of the composite ore sample was carried out and the work is described in Appendix A. This work revealed the following information concerning the liberation characteristics of the minerals of economic interest:

Fluorite. In the size range minus 17 plus 9 μm liberation is about 80%. Unliberated fluorite is approximately equally locked with magnetite and silicates.

Tin. Approximately 45% of the tin is contained in solid solution in garnet. The remaining tin exists as cassiterite and less than 5% of this is liberated at sizes greater than 9 μm.

Tungsten. Tungsten occurs as scheelite. About half of the scheelite is well liberated in the size range minus 150 plus 75 μm (and perhaps at an even coarser size), the remaining scheelite requiring a grind finer than 75 μm for significant liberation to occur.

Magnetite. Magnetite is closely associated with silicates and fluorite and abundant composites of these minerals in the size range 7 to 33 μm were noted.

3. FLOTATION REAGENTS

The following reagents were used:

Reagent	Supplier
MR Sodium Silicate (38% solids, $Na_2O:SiO_2$ = 1:3.22)	Nightingale Chemicals
Oleic acid/Akypomine AT	Fabriek van Chemische Producten (Holland)
AP845	Cyanamid Australia Ltd
Teric 400	ICIANZ Ltd
Quebracho	Cyanamid Australia Ltd

4. EQUIPMENT

The following equipment was used:

Davis Tube magnetic separator

Wet drum magnetic separator — Eriez, Model L-8

Laboratory flotation machine - Agitair Model LA-500

5. EXPERIMENTAL PROCEDURE AND RESULTS

5.1 Magnetic Separation

A sample of ore was ground to minus 100 mesh and sub-samples treated in a Davis Tube magnetic separator. The feed sizing is given in Table 2 (10 min. grind). Test conditions and results are given in Table 3.

5.2 Magnetic Separation and Flotation

All flotation tests were carried out at 40°C and circuit water used was demineralised.

Test 1. A 500-g charge was ground to minus 100 mesh and separated in an Eriez wet drum magnetic separator at a field strength of 1000 gauss. The results are shown in Table 4.
The non-magnetic fraction was treated by rougher flotation using a fatty acid collector. The flotation conditions and overall magnetic separation-flotation results are given in Table 5.

Test 4. This test was carried out using a finer feed grind of 100% passing 170 mesh (15 min. grind, Table 2). After magnetic separation the non-magnetic products were floated to five rougher stages for high recovery, and the combined concentrate was then cleaned three times. The test conditions and results are detailed in Table 6.

Test 5. This test was carried out at the finer feed sizing used in Test 4. Sodium silicate was added to the mill and magnetic separation was applied to the third cleaner concentrate instead of the flotation feed. Test data and results are presented in Table 7.

5.3 Flotation Without Magnetic Separation

Because of the high fluorite losses associated with the magnetic separations in the above tests, magnetic separation was not included in the following tests:

Test 3. The coarser feed grind of 100% minus 100 mesh was employed in this test. Three rougher concentrates were collected, combined and cleaned once. Test details and results are contained in Table 8.

5.4 Flotation with Regrind of Cleaner Feed

Test 6. In this test the combined rougher concentrates were cleaned once and the cleaner concentrate reground before cleaning a further three times. The results of this test together with the procedure are included in Table 9.

Test 8. This test followed the general procedure of Test 6 but with the following differences:

(a) Regrind was applied to the total rougher concentrate;

(b) A finer regrind product sizing was sought;

(c) The number of cleaning stages was increased to five.

The test conditions and results are detailed in Table 10.

Test 9. This test followed the general procedure of Test 8 except for the following differences:

(a) An additional (sixth) cleaning stage was carried out;

(b) Quebracho was employed in the third and sixth cleaning stages to depress calcite.

Table 11 outlines the details of the test conditions and results.

Test 7. In this test, AP845 was used as the collector, replacing the fatty acid collector used in all other tests. Four rougher concentrates were collected, combined and cleaned three times. Table 12 presents the detailed procedure and results.

6. DISCUSSION

6.1 Magnetic Separation

Fluorite losses were high when magnetic separation was employed. The loss was reduced to 23.8% in Test 4 compared with 33.6% in Test 1 due to the finer grind employed in the former test.

In Test 5 which employed the same feed sizing as Test 4, fluorite losses increased to 31.2% when magnetic separation was applied to the final concentrate instead of the flotation feed.

6.2 Regrind

Regrinding of the rougher concentrate before cleaning resulted in a considerable improvement in selectivity and this can be seen readily from the comparisons given in Fig. 1.

6.3 Depression of Calcite

In the final cleaning stages of Test 8 it was evident from Fig. 1 that substantial fluorite losses were incurred in seeking higher concentrate grades (maximum 88.4% CaF_2).

Mineralogical examination (see Appendix B) shows that the main diluent in the final concentrate was well liberated calcite.

The use of Quebracho in the cleaning stages of Test 9 was effective in depressing most of the calcite and improving the overall cleaner flotation response (refer to Fig. 1). Assay of the final cleaner concentrate in this case using the normal specific ion electrode method gave 92.5% CaF_2. This compares with 93.1% CaF_2 by check titration method.

The obvious benefit gained in the cleaner flotation response as a result of employing Quebracho in the third and sixth cleaning stages in Test 9 justifies further work into the investigation of its use in the primary and secondary grinding stages. Benefits likely to result in this case are:

(a) Overall improvement in the flotation response.

(b) Lower collector requirement.

6.4 Iron Contamination

The results of Tests 8 and 9 show a minimum iron value in the final concentrate of 1.4% Fe. This arises due to the fact that despite the fine sizing of the concentrate (nominally minus 15 μm) there is a significant amount of magnetite present, mostly composite with the fluorite which cannot be removed without affecting the fluorite recovery.

6.5 Scheelite Extraction

The use of Quebracho in the cleaning stages was not very effective in rejecting the scheelite in a low weight middling fraction as 46% of the scheelite still reported in the final concentrate for Test 9 when using Quebracho compared to an estimated 57% when it was omitted (Test 8).

6.6 Collector AP845

It is obvious from the results of Test 7 that excessive collector was used in this test. Further testing with AP845 will need to be carried out using a more appropriate addition along with Quebracho depressant for calcite.

6.7 Prospects for Chemical Treatment

Clearly, magnetic separation was not a satisfactory means of removing iron (i.e., magnetite) from the concentrate. The residual iron content of the non-magnetic fraction exceeded 5% Fe, and the magnetic fraction contained 20 to 30% of the total fluorite.

On the other hand, a series of cleaner flotation steps, with intermediate regrinding to improve liberation, progressively lowered the iron content with only a gradual decline in fluorite recovery. From Fig. 1, the best fluorite concentrate for subsequent chemical treatment might be prepared by the use of three cleaning stages, since the improvement in CaF_2 grade tails off beyond this. At this point, the concentrate contained 80% of the fluorite, at a grade of 82% CaF_2, and a residual iron content of 2.9% Fe (97.3% of iron eliminated in tailings). Some improvement in the CaF_2 grade may result from the recommended use of Quebracho in the primary and secondary grinding stages, by depressing more calcite.

This concentrate would be an attractive feed material for the proposed chemical treatment process, which utilizes an acidic reagent, because the principal acid-consuming gangue minerals, magnetite and calcite, are largely eliminated by flotation. Moreover, the five-fold increase in CaF_2 grade is, of course, of great significance in reducing the size of the roaster, a major capital cost item.

This fluorite concentrate feed to the chemical treatment process would contain only about 60% of the tungsten content of the ore at a grade of about 0.4% WO_3. The tungsten would remain in the residue after the chemical extraction of fluoride, but at a grade of around 0.2% WO_3 it is doubtful that this residue could be considered an economic source of tungsten. Furthermore, the prospects for economic recovery of tin are not encouraging,

in view of the low tin content and the nature of the tin (solid solution in garnet and intergrown cassiterite).

An economic evaluation of the proposed chemical treatment process, applied to such a flotation concentrate is recommended, along the lines of the previous Project Proposal (No. 5/0/2862, dated 6 August, 1976)

7. CONCLUSIONS

1. By fine grinding and flotation a concentrate assaying 92.5% CaF_2 with a recovery of ~~45.7%~~ 64.9% was produced from a feed assaying 16.5% CaF_2.

2. Removal of magnetite by magnetic separation applied to either the flotation feed or concentrate resulted in a high loss of fluorite due to the poor liberation characteristics of magnetite.

3. Grinding to 17 μm is necessary for good liberation of fluorite. Scheelite is well liberated at flotation sizes but little cassiterite liberated above 9 μm. Much of the magnetite is liberated at 33 μm but finely disseminated magnetite is evident in grains finer than 15 μm.

4. The chief contaminants in the flotation concentrate are magnetite, calcite, scheelite and amphibole. Further flotation cleaning stages should reject calcite and amphibole, but the magnetite occurs as fine inclusions in the fluorite. Differential flotation of the scheelite from the fluorite has not been studied.

5. It is unlikely that a high grade magnetite concentrate suitable for heavy medium use could be recovered from the ore because of the poor liberation characteristics of this mineral.

6. The composite ore sample contains 0.13% Sn, 45% of which is present in solid solution in garnet. The remaining tin is present as cassiterite which is too finely disseminated for physical beneficiation.

7. The composite ore sample contains 0.11% WO_3 as scheelite. This scheelite concentrated to 0.4% WO_3 with the fluorite during flotation but it is doubtful whether the scheelite could be recovered economically from this concentrate unless chemical treatment was carried out.

8. RECOMMENDATION

As discussed in Section 6.7 of the Report it may be feasible to treat the Moina ore by flotation to produce a low-grade concentrate at high recovery and then leach this concentrate to extract fluoride. It is recommended that sufficient work be carried out to enable a preliminary assessment of the technical and economic feasibility of this process to be made. A proposal for an economic evaluation of the proposed treatment dated 6 August 1976 has been forwarded.

APPENDIX A

ORE-DRESSING MINERALOGY OF MOINA FLUORITE COMPOSITE ORE SAMPLE

SUMMARY

A composite head sample of Moina ore has been studied in detail mineralogically to provide basic information relevant to the metallurgical treatment of the ore.

The head sample has the following chemical composition:

	%		ppm
F	8.05	(equivalent to 16.5% fluorite)	Bi 450
Acid-soluble Fe	16.7	(equivalent to 23.1% magnetite)	Mo 45
SiO_2	30.6		U 4
S	0.33		Au 0.12
P_2O_5	0.06		
Sn	0.13		
W	0.085		

The approximate bulk mineralogy of the head sample is as follows:

Mineral	Wt %
Garnet	25
Magnetite	25
Quartz	20
Fluorite	16
Amphibole	10
Others	4

'Others' includes biotite, chlorite, calcite, feldspar, vesuvianite, pyrite, chalcopyrite, ?pyrrhotite, arsenopyrite, hematite, goethite, cassiterite and scheelite.

Fluorine is present overwhelmingly as fluorite (traces may be present in micas but no cuspidine was observed), which is less than 50% liberated at particle sizes greater than 33 μm and about 80% liberated in the particle size range minus 17 plus 9 μm. The unliberated fluorite is approximately equally locked with magnetite and with silicates.

Tin is present in both cassiterite and in solid solution in garnet. The tin in garnet is present at low levels, ranging from 0.0 up to 1.3% Sn with an average of 0.26% Sn. Approximately 45% of the tin in the sample is in garnet and, of the 55% in cassiterite, only a very small amount (<5%) is liberated at particle sizes coarser than 9 μm. There is thus little if

any, possibility of recovering a significant proportion (e.g. >20%) of the tin in the sample into a high-grade tin concentrate by physical methods of beneficiation.

Tungsten is present as scheelite, which occurs in two different forms; about half occurs in relatively coarse crystals which are well liberated in the size range minus 150 plus 75 µm (and are possibly liberated at even coarser particle sizes), and half occurs as much more finely disseminated crystals which are progressively liberated with decreasing particle size below 75 µm. The mineralogical data suggest that it may be possible to recover 50 to 70% of the tungsten into relatively high-grade concentrates by physical beneficiation.

Magnetite is closely associated with silicates and fluorite and considerable problems may be expected in obtaining high-grade magnetite concentrates by low-intensity magnetic separation even in the particle size range of minus 33 plus 17 µm because of the presence of abundant magnetite-bearing composites containing silicates and fluorite.

1. INTRODUCTION

As part of a bench-scale metallurgical testing programme on a composite head sample of fluorite ore from Moina, Tasmania, a detailed ore-dressing mineralogical study has been undertaken on crushed and ground head ore to determine features of metallurgical signficance, in particular the mineralogical location and liberation/locking characteristics of fluorine, tin and tungsten, and the liberation/locking characteristics of magnetite. This Report includes the results of chemical analyses and the detailed ore-dressing mineralogical investigation.

2. PROCEDURES

A representative sub-sample of the head was analysed quantitatively for selected elements and semi-quantitatively (by emission spectrography) for a wide range of elements.

A further sample of the ore was crushed and ground (rod-milled) to approximately 90% passing 75 µm (200 mesh BSS) and wet screened and cyclosized to give various size fractions. Selected size fractions were recombined to give the following size fractions for detailed study:

(-150) +75 µm
-75 +33 µm
-33 +17 µm
-17 + 9 µm
-9 µm

Each size fraction was then separated in a Davis Tube using a field strength of 1000 gauss; the products were dried and weighed. The non-magnetic products were then separated at specific gravities of 3.1 and 3.3 using methylene iodide (in the former case diluted with acetone). Because of the small particle size of the products, centrifugal separation techniques were used throughout. The products were washed, dried and weighed. Riffled portions of the products were used in the preparation of polished briquettes (PS Nos 25107-22) and further riffled portions were analysed for acid-soluble fluorine, tungsten and tin.

In this separation scheme, virtually all grains containing magnetite report into the magnetic product; fluorite, scheelite and cassiterite locked with light gangue minerals report in the <3.1 sp.gr. product; liberated fluorite reports into the 3.1 to 3.3 sp.gr. product (together with some scheelite and cassiterite locked with light gangue and/or fluorite); the >3.3 sp.gr. product contains all the liberated scheelite and cassiterite together with these minerals and fluorite that are locked with dense gangue minerals. By assay and mineralogical assessment of the separation products, it is possible to quantify the liberation of the valuable minerals.

Two products were analysed by X-ray diffraction and all were examined both in transmitted and reflected light. Two >3.3 sp.gr. products were examined using an electron-probe microanalyser in order to determine whether or not wolframite or stannospinel are present in the ore and to assess the amount of tin in garnet. High intensity magnetic separation products were prepared of the minus 75 plus 33 μm >3.3 sp.gr. non-magnetic product and the more magnetic products were examined for the presence of wolframite.

All results were collated and computer-processed.

3. HEAD ANALYSES

The accurate chemical analyses of the head sample are as follows:

F	%	8.05
Acid-soluble Fe	%	16.7
SiO_2	%	30.6
S	%	0.33
P_2O_5	%	0.06
Sn	%	0.13
W	%	0.085
Bi	ppm	450.0
Mo	ppm	45.0
U	ppm	4.0
Au	ppm	0.12

Semi-quantitative emission spectrographic analyses of the head sample are given in Table 1 in the body of the Report.

4. ELEMENT DISTRIBUTION DATA

4.1 Assays and Distribution of Fluorite

The results of magnetic and heavy-liquid separations and fluorine assays are given in Table A-1. Acid-soluble fluorine determinations have been converted to values in terms of calcium fluoride (fluorite). The following conclusions can be drawn from this table:

1. The calculated head assay is 15.6% fluorite.
2. About 90% of the ore passes 75 µm, and crushing to this size has resulted in the production of 20% of slimes (-9 µm) which contain 25% of the fluorite in the ore. The enrichment of fluorite in the slimes is due to preferential grinding of the fluorite as compared to the other constituents and this effect is also shown, by the increasing assays of the size fractions with decreasing size, from 11% fluorite in the plus 75 µm fraction to 20.2% fluorite in the minus 17 plus 9 µm fraction.
3. Sixty percent of the fluorite in the 'sands' fractions (i.e. the +9 µm fractions) reports into the 3.1 to 3.3 sp.gr. product of the non-magnetic and this material is essentially liberated fluorite. Recovery of fluorite into this product increases from 23% in the plus 75 µm fraction to 83% in the minus 7 plus 9 µm fraction.
4. Increased liberation of fluorite with decreasing particle size causes a decrease in the fluorite assays of the ferromagnetic and <3.1 and >3.3 sp.gr. non-magnetic products. The distributions of fluorite into these products also show a decrease with decreasing particle size. The following tabulation summarises these results in the case of the distributions:

Size Fraction (µm)	Ferromagnetic Product	Non-magnetic Products		
		<3.1	3.1-3.3	>3.3
+75	42	19	23	16
-75+33	37	13	37	13
-33+17	15	9	72	5
-17+9	5	9	83	2
Total sands	(21)	(11)	(60)	(8)

These results show that effective liberation of fluorite from gangue minerals (particularly magnetite) occurs only at particle sizes of less than 33 μm.

4.2 Assays and Distribution of Tungsten

The results of assaying the magnetic and specific gravity products are shown in Table A-2 (assays are expressed as the element, tungsten) and the following conclusions can be drawn from this table:

1. The calculated head assay is about 875 ppm.
2. In the plus 9 μm size fractions, 64% of the tungsten reports into the >3.3 sp.gr. non-magnetic product and hence could be liberated. Almost 27% of the tungsten in the plus 9 μm material is locked in the ferromagnetic product and only 10% reports in the two lighter products (<3.1 and 3.1-3.3 sp.gr.) and hence is locked with light gangue minerals or fluorite.
3. The assays of the ferromagnetic and <3.1 sp.gr. non-magnetic products decrease with decreasing particle size but the 3.1 to 3.3 sp.gr. non-magnetic product shows a less well-defined trend.
4. The tungsten assays of the >3.3 sp.gr. non-magnetic products (in which all the liberated tungsten minerals occur) increase with decreasing particle size from 1900 ppm in the plus 75 μm fraction to 2700 ppm in the minus 17 plus 9 μm fraction. Throughout the plus 9 μm size range the distribution of tungsten into the >3.3 sp.gr. non-magnetic product varies between 60 and 68% only.
5. There is no preferential grinding of the tungsten minerals and the minus 9 μm slimes contain 18% of the tungsten in the ore.

4.3 Assays and Distribution of Tin

Table A-3 shows the results of analysis of the magnetic and specific gravity products for tin; from this table the following conclusions can be drawn:

1. The calculated head assay is about 1200 ppm Sn.
2. Overall in the plus 9 μm fractions only 43% of the tin reports into the >3.3 sp.gr. non-magnetic products, which contain all the liberated tin in the ground ore. Most of the locked tin in these size fractions occurs in the ferromagnetic product (33% of the tin) and 24% only occurs in the <3.3 sp.gr. products where

the tin-bearing mineral(s) is locked with light gangue and/or fluorite.

3. The assays of the ferromagnetic products and the <3.3 sp.gr. non-magnetic products decrease with decreasing particle size and this indicates that the tin-bearing mineral(s) is progressively liberated from gangue.

4. The assays of the >3.3 sp.gr. products tend to increase with decreasing particle size although the distribution of tin into these products shows relatively little variation:

Size Fraction (μm)	Tin in >3.3 sp.gr. Non-Mag Product Assay (ppm)	Distribution %
+75	1000	32
-75+33	2200	49
-33+17	1700	40
-17+9	2600	38

Even at particle sizes of less than 17 μm, 42% of the tin occurs in composite grains in such small quantities that the grains report in ferromagnetic or <3.3 sp.gr. products. In addition, the tin assay of the minus 17 plus 9 μm >3.3 sp.gr. non-magnetic product is only twice the head assay of the ore.

5. MINERALOGY

5.1 Mineralogy of the Ferromagnetic Products

The ferromagnetic products contain 25% magnetite (+75 μm) to >70% magnetite (-17+9 μm) and the fluorite occurs in these products almost completely locked.

In the plus 75 μm ferromagnetic product the fluorite crystals are commonly 20 to 60 μm in size and many occur in complex ternary (+) grains (i.e. composite grains composed of 3 or more minerals) with complex intergrowths (on a scale of <10 μm). Such textures are shown in Plate A-2a.

With decreasing particle size the locking between fluorite and magnetite and non-opaques becomes relatively simpler on the scale of the particle size and, in the minus 17 plus 9 μm ferromagnetic product, composites with magnetite and fluorite are generally binary grains (i.e. composite grains composed of two minerals) with simple textures.

In the plus 75-μm ferromagnetic product magnetite is less than 10% liberated but in the minus 75 plus 33-μm product about 75% of the magnetite is present in particles which contain more than 80% of magnetite. In the minus 33-μm ferromagnetic products the liberation of magnetite is excellent.

Traces of pyrrhotite were noted in these products.

5.2 Mineralogy of the <3.1 sp.gr. Non-magnetic Products

The mineralogy of the <3.1 specific gravity non-magnetic products was assessed by optical observation and also by means of an X-ray diffraction study of the <3.1 specific gravity non-magnetic product of the minus 75 plus 33-μm fraction. The result is as follows:

Quartz	Dominant
Biotite	Accessory
Feldspar	Accessory
Fluorite	Accessory
Chlorite?	Trace
Calcite	Trace
Magnetite?	Trace
Amphibole?	Trace

Key:

D = Dominant. Used for the component apparently most abundant, regardless of its probable percentage level.

A = Accessory. Components judged to be present between the levels of roughly 5 and 20%.

Tr = Trace. Components judged to be below about 5%.

In temporary grain mounts many particles appear to be dark, fine-grained material and, where fluorite can be seen, it forms extremely complex intergrowths in particles containing several mineral components. In polished sections, where the fluorite can be more readily identified, grains consisting of several non-opaque phases and specks of fluorite are common (Plate A-2d).

5.3 Mineralogy of the 3.1-3.3 sp.gr. Non-magnetic Products

The 3.1 to 3.3 sp.gr. non-magnetic products contain abundant liberated fluorite and less than 0.5% magnetite. The principal diluent mineral is amphibole, which has a specific gravity close to that of fluorite.

In the plus 75 μm 3.1 to 3.3 sp.gr. non-magnetic product fluorite is approximately 80% liberated and most fluorite crystals are completely clear. A few, however, contain small (<5 μm) inclusions of ?garnet and/or amphibole. The liberation of fluorite in the minus 75 plus 9 μm 3.1 to

3.3 sp.gr. non-magnetic products is >90% and only a few grains contain small inclusions of gangue minerals (generally silicates). The photomicrographs in Plate A-1 show the textures of grains in the products.

5.4 Mineralogy of the >3.3 sp.gr. Non-magnetic Products

The overall mineralogy of the >3.3 sp.gr. non-magnetic products was assessed by means of an X-ray diffraction trace taken of the >3.3 sp.gr. non-magnetic product of the minus 75 plus 33 µm fraction; the results are as follows:

Garnet	Dominant
Amphibole	Accessory
Vesuvianite	Accessory
Mica	Accessory
?Magnetite	Trace

Key:

D = Dominant. Used for the component apparently most abundant, regardless of its probable percentage level.

A = Accessory. Components judged to be present between the levels of roughly 5 and 20%.

Tr = Trace. Components judged to be below about 55%.

Other minerals which are generally present in small quantities in these >3.3 sp.gr. products are scheelite, fluorite, cassiterite, quartz, pyrite, chalcopyrite, bismuthinite, arsenopyrite, hematite and goethite.

Fluorite occurs in the >3.3 sp.gr. products in composites with garnet and/or amphibole; in the coarser size fractions some of these composites have extremely complex textures and fluorite crystals are as little as 5 to 10 µm in size (Plate A-2b). In the finer fractions the fluorite forms simple granular binary grains with garnet and/or amphibole, as shown in Plate A-2c.

Magnetite is present in only very small quantities (probably <0.5%) and is locked with non-opaque silicates.

Scheelite was identified largely by its fluorescence in short wavelength ultraviolet light. By this means several liberated grains were identified in the >3.3 sp.gr. non-magnetic products and no locked scheelite was noted, even in the plus 75 µm >3.3 sp.gr. products.

Locked scheelite was identified in the two plus 33 µm 3.1 to 3.3 sp.gr. non-magnetic products and in the plus 75-µm ferromagnetic product.

Wolframite was sought in the >3.3 sp.gr. products and some high-intensity magnetic separations of the minus 75 plus 33 µm >3.3 sp.gr. non-magnetic

product were made in order to concentrate this mineral. No wolframite was observed in either the >3.3 sp.gr. products or in the high-intensity magnetic products referred to above and it is concluded, therefore, that tungsten occurs in this ore in scheelite only.

Only traces of cassiterite were noted in the >3.3 sp.gr. non-magnetic products, which have tin assays of 0.1 to 0.26%; consequently an attempt was made to locate tin in spinel and in garnet. The minus 75 plus 33 μm >3.3 sp.gr. non-magnetic product was examined using an electron microprobe. No stannospinel was found but some tin was detected in garnet and a more detailed study was undertaken. Randomly selected points in each of 70 garnet grains were analysed for tin: at a detection limit of 0.1% Sn, 20 grains contained no detectable tin but the mean tin value in the 70 grains analysed was 0.26% Sn (value corrected for atomic number, absorption and fluorescence). The maximum value noted was 1.3% Sn.

In order to assess the distribution of tin in garnet and cassiterite, the amount of garnet in the ferromagnetic products and the various non-magnetic specific gravity products was estimated visually, with the results given in Table A-4.

In approximate terms, therefore, the distribution of tin in the ore is calculated to be as in Table A-5 (some data from Table A-3).

These values are semi-quantitative only but indicate that 35 to 50% of the tin in the ore is in solid solution in garget. Where cassiterite was identified in the >3.3 specific gravity products a moderate proportion in the plus 33-μm material was locked with garnet whereas liberated grains were common in the mins 33 plus 9 μm material.

5.5 Overall Mineralogy of Sample

From the mineralogies and weight distributions of the various separation products, the approximate overall mineralogical composition of the total sample can be calculated. This is as follows:

	Wt %	
Garnet	25	
Magnetite	25	
Quartz	20	
Fluorite	16	(from assay)
Amphibole	10	
Others	4	
	100	

'Others' includes biotite, chlorite, calcite, feldspar, vesuvianite, pyrite, chalcopyrite, ?pyrrhotite, arsenopyrite, hematite, goethite, cassiterite and scheelite.

6. OVERALL LIBERATION AND LOCKING CHARACTERISTICS OF FLUORITE TIN AND TUNGSTEN

The ore contains approximately 16% fluorite which is the mineral of principal interest at this stage of the investigation. From the sizing, separation, assay and optical data obtained in this work, the overall distribution of fluorite can be determined (Tables A-6 and A-7).

It can be seen from Table A-6 that fluorite liberation is less than 50% at particles sizes greater than 33-μm. The unliberated fluorite is approximately equally locked with magnetite and silicates.

In the minus 33 plus 17-μm material the liberation of fluorite is about 70% and grinding to pass approximately 30 μm will be needed to produce a high degree of liberation of fluorite.

The overall distribution of tin in the size fractions and the total sample is given in Tables A-8 and A-9. Approximately 45% of the tin in the ore is present in solid solution in garnet (0.06% Sn out of 0.13% Sn) and, of the 55% tin in cassiterite, only a very small amount is liberated at any size above 9 μm. There is thus little if any possibility of recovering a significant (e.g. >20%) proportion of the tin in the sample into a high grade tin concentrate by physical beneficiation.

The overall distribution of tungsten in the size fractions and the total sample are given in Tables A-11 and A-12. The tungsten is present as scheelite and it can be seen from Table A-11 that even in the plus 75-μm fraction approximately 50% of the scheelite is liberated, and that with

decreasing particle size there is only a small increase in scheelite liberation. This suggests that the scheelite occurs in two different forms in the ore (as indicated previously by Mr H.W. Fander*), approximately half as relatively coarse crystals and half as much more finely disseminated crystals some of which are progressively liberated with decreasing particle size below 75 µm. The mineralogical results therefore indicate that it may be possible to recover 50 to 70% of the tungsten into relatively high-grade concentrates by physical beneficiation.

Magnetite is of potential value in the ore but the mineralogical work indicates that considerable problems will be met in obtaining high-grade magnetite concentrates. The ferromagnetic products obtained from minus 33 plus 17-µm material contain about 50% magnetite of which only 60% is completely liberated. This information, as well as that in Table A-6, shows the close association between magnetite and fluorite even in particles as small as 20 µm.

Investigation and Report by: B.G. Steveson

Officer in Charge, Mineralogy/Petrology Section: K.J. Henley

*Central Mineralogical Services Report CMS 76/5/16.

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TABLE A-1: ASSAYS AND DISTRIBUTION OF FLUORITE

Size Fraction µm	Specific Gravity or Magnetic Fraction	In Size Fractions			In Head Sample		Cumulative Data		
		Weight %	Assay %	Fluorite Distribution,%	Weight %	Fluorite Distribution,%	Weight %	Assay %	Distribution %
+75	<1.00*	32.9	14.00	41.9	3.62	3.25	3.62	14.00	3.25
	<3.10	24.0	8.70	19.0	2.64	1.47	2.64	8.70	1.47
	<3.30	9.5	26.50	22.9	1.05	1.78	1.05	26.50	1.78
	>3.30	33.6	5.30	16.2	3.69	1.25	3.69	5.30	1.25
	Total	100.0	(11.00)	100.0	11.00	7.75	11.00	11.00	7.75
-75+33	<1.00	40.9	10.10	36.8	12.19	7.89	15.82	10.99	11.14
	<3.10	16.6	8.80	13.0	4.94	2.79	7.58	8.77	4.26
	<3.30	9.4	45.00	37.5	2.79	8.04	3.83	39.86	9.82
	>3.30	33.1	4.30	12.7	9.87	2.72	13.57	4.57	3.97
	Total	100.0	(11.23)	100.0	29.80	21.44	40.80	11.17	29.19
-33+17	<1.00	28.7	9.00	14.9	6.52	3.76	22.33	10.41	14.90
	<3.10	22.8	6.70	8.8	5.19	2.23	12.77	7.93	6.48
	<3.30	23.4	53.20	71.6	5.31	18.11	9.15	47.65	27.93
	>3.30	25.0	3.30	4.8	5.69	1.20	19.25	4.20	5.18
	Total	100.0	(17.39)	100.0	22.70	25.29	63.50	13.39	54.48
-17+ 9	<1.00	19.9	5.50	5.4	3.18	1.12	25.51	9.80	16.02
	<3.10	26.4	7.00	9.2	4.23	1.90	16.99	7.70	8.38
	<3.30	37.5	44.70	83.0	6.00	17.18	15.14	46.48	45.10
	>3.30	16.2	3.00	2.4	2.60	0.50	21.85	4.05	5.68
	Total	100.0	(20.18)	100.0	16.00	20.69	79.50	14.76	75.18
Totals for above size fractions	<1.00	32.1	(9.80)	21.3	-	-	-	-	-
	<3.10	21.4	(7.70)	11.1	-	-	-	-	-
	<3.30	19.1	(46.48)	60.0	-	-	-	-	-
	>3.30	27.5	(4.05)	7.6	-	-	-	-	-
Overall Totals	Sands	79.5	(14.76)	75.2					
	Slimes	20.5	18.90	24.8					
	Total	100.0	(15.61)	100.0	Calculated head assay = 15.61%				

*In Tables A2-A4 <1.000 refers to the ferromagnetic product; the remaining three values are the specific gravity ranges of the non-magnetic products (<3.30 refers to the 3.10 to 3.30 sp.gr. product).

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TABLE A-2: ASSAYS AND DISTRIBUTION OF TUNGSTEN

Size Fraction mm	Specific Gravity or Magnetic Fraction	In Size Fractions			In Head Sample		Cumulative Data		
		Weight %	Assay ppm	Tungsten Distribution,%	Weight %	Tungsten Distribution,%	Weight %	Assay ppm	Distributio %
+75	<1.00	32.9	700.00	23.9	3.62	2.90	3.62	700.00	2.90
	<3.10	24.0	250.00	6.2	2.64	.75	2.64	250.00	.75
	<3.30	9.5	400.00	3.9	1.05	.48	1.05	400.00	.48
	>3.30	33.6	1900.00	66.0	3.69	8.03	3.69	1900.00	8.03
	Total	100.0	(966.61)	100.0	11.00	12.17	11.00	966.61	12.17
-75+33	<1.00	40.9	900.00	35.1	12.19	12.56	15.82	854.18	15.46
	<3.10	16.6	200.00	3.2	4.94	1.13	7.58	217.39	1.89
	<3.30	9.4	200.00	1.8	2.79	.64	3.83	254.53	1.12
	>3.30	33.1	1900.00	60.0	9.87	21.47	13.57	1900.00	29.50
	Total	100.0	(1049.67)	100.0	29.80	35.80	40.80	1027.28	47.96
-33+17	<1.00	28.7	700.00	23.1	6.52	5.22	22.33	809.20	20.68
	<3.10	22.8	150.00	3.9	5.19	.89	12.77	190.01	2.78
	<3.30	23.4	250.00	6.7	5.31	1.52	9.15	251.90	2.64
	>3.30	25.0	2300.00	66.2	5.69	14.96	19.25	2018.13	44.46
	Total	100.0	(869.78)	100.0	22.70	22.59	63.50	970.98	70.56
-17+9	<1.00	19.9	350.00	10.7	3.18	1.27	25.51	752.01	21.96
	<3.10	26.4	100.00	4.1	4.23	.48	16.99	167.62	3.26
	<3.30	37.5	300.00	17.4	6.00	2.06	15.14	270.95	4.70
	>3.30	16.2	2700.00	67.8	2.60	8.03	21.85	2099.22	52.49
	Total	100.00	(646.92)	100.0	16.00	11.85	79.50	905.76	82.40
Totals for above size fractions	<1.00	32.1	(752.01)	26.6	-	-	-	-	-
	<3.10	21.4	(167.62)	4.0	-	-	-	-	-
	<3.30	19.1	(270.95)	5.7	-	-	-	-	-
	>3.30	27.5	(2099.27)	63.7	-	-	-	-	-
Overall Totals	Sands	79.5	(905.76)	82.4					
	Slimes	20.5	750.00	17.6					
	Total	100.0	(873.83)	100.0	Calculated head assay = 873.83 ppm				

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TABLE A-31 ASSAYS AND DISTRIBUTION OF TIN

Size Fraction	Specific Gravity or Magnetic Fraction	In Size Fractions			In Head Sample		Cumulative Data		
µm		Weight %	Assay ppm	Tin Distribution,%	Weight %	Tin Distribution,%	Weight %	Assay ppm	Distribution %
-75	<1.00	32.9	1450.00	44.9	3.62	4.36	3.62	1450.00	4.36
	<3.10	24.0	550.00	12.4	2.64	1.20	2.64	550.00	1.20
	<3.30	9.5	1250.00	11.2	1.05	1.08	1.05	1250.00	1.08
	>3.30	33.6	1000.00	31.6	3.69	3.07	3.69	1000.00	3.07
	Total	100.0	(1064.19)	100.0	11.00	9.72	11.00	1064.19	9.72
-75+33	<1.00	40.9	1400.00	38.5	12.19	14.17	15.82	1411.46	18.53
	<3.10	16.6	550.00	6.1	4.94	2.26	7.58	550.00	3.46
	<3.30	9.4	1000.00	6.3	2.79	2.32	3.83	1068.16	3.40
	>3.30	33.1	2200.00	49.0	9.87	18.03	13.57	1873.28	21.09
	Total	100.0	(1486.61)	100.0	29.80	36.77	40.80	1372.72	46.49
-33+17	<1.00	28.7	1150.00	30.9	6.52	6.22	22.33	1335.18	24.75
	<3.10	22.8	500.00	10.7	5.19	2.15	12.77	529.69	5.61
	<3.30	23.4	850.00	18.6	5.31	3.75	9.15	941.46	7.15
	>3.30	25.0	1700.00	39.8	5.69	8.02	19.25	1822.11	29.12
	Total	100.0	(1069.05)	100.0	22.70	20.14	63.50	1264.17	66.63
-17+9	<1.00	19.9	850.00	15.3	3.18	2.24	25.51	1274.76	26.99
	<3.10	26.4	450.00	10.8	4.23	1.58	16.99	509.87	7.19
	<3.30	37.5	1050.00	35.7	6.00	5.23	15.14	984.44	12.37
	>3.30	16.2	2600.00	38.3	2.60	5.61	21.85	1914.62	34.73
	Total	100.0	(1103.55)	100.0	16.00	14.66	79.50	1231.04	81.28
Totals for above size fractions	<1.00	32.1	(1274.76)	33.2	-	-	-	-	-
	<3.10	21.4	(509.87)	8.8	-	-	-	-	-
	<3.30	19.1	(984.44)	15.2	-	-	-	-	-
	>3.30	27.5	(1914.62)	42.7	-	-	-	-	-
Overall Totals	Sands	79.5	(1231.84)	81.3					
	Slimes	20.5	1100.00	18.7					
	Total	100.0	(1204.81)	100.0	Calculated head assay = 1204.81 ppm				

TABLE A-4: GARNET CONTENT AND TIN DISTRIBUTION IN THE SEPARATION PRODUCTS

Product Size Fraction	Approx. Wt % Garnet	% Sn in Garnet*	Proportion of Sn in Product in Garnet, %
>3.3 sp.gr. non-magnetic			
+75 µm	55	0.14	~100
-75+33 µm	60	0.16	~ 70
-33+17 µm	75	0.19	~100
-17+ 9 µm	80	0.21	~ 80
3.1-3.3 sp.gr. non-magnetic			
+75 µm	10	0.03	~ 25
-75+33 µm	5-10	0.01-0.03	10-30
-33+17 µm	<5	<0.01	<10
-17+ 9 µm	<5	<0.01	<10
<3.1 sp.gr. non-magnetic			
+75 µm	5	0.01	~ 20
-75+33 µm	3	<0.01	< 20
-33+17 µm	<3	<0.01	< 20
-17+ 9 µm	<3	<0.01	< 20
Ferromagnetic			
+75 µm	15	0.04	~ 25
-75+33 µm	<15	<0.04	~ 25
-33+17 µm	5	0.01	~ 10
-17+ 9 µm	5	0.01	~ 10

*Assuming garnet contains an average of 0.26% Sn.

TABLE A-5: OVERALL TIN DISTRIBUTION BETWEEN GARNET AND CASSITERITE

	Distribution in Total Ore %	% in Garnet	% in Cassiterite
1. Tin in minus 9 µm slimes	19	?9	?10
2. Tin in plus 9 µm ferromagnetic products (i.e. locked with magnetite)	27	4	23
3. Tin in <3.3 sp.gr. products (i.e. locked with light gangue and fluorite)	19	3	16
4. Tin in plus 9 µm >3.3 sp.gr. non-magnetic products (i.e. liberated cassiterite, cassiterite locked with dense gangue and liberated garnet)	35	26	9
Total	100	42	58

TABLE A-6: LOCATION OF FLUORITE IN EACH SIZE FRACTION (VALUES IN %)

Size Fraction μm	Liberated	Locked			Total
		With Magnetite	With Silicates	Total	
+75	17	42	41	83	100
-75+33	35	37	28	65	100
-33+17	70	15	15	30	100
-17+9	80	5	15	20	100
-9*	(100)	0	0	0	100
Total	(70)	(16)	(14)	(30)	100

*It was assumed that all the fluorite in the minus 9-μm fraction is liberated.

TABLE A-7: OVERALL DISTRIBUTION OF FLUORITE (% OF FLUORITE IN ORE)

Size Fraction μm	Liberated	Locked			Total
		With Magnetite	With Silicates	Total	
+75	2	3	3	6	8
-75+33	8	8	5	13	21
-33+17	18	4	3	7	25
-17+9	17	1	3	4	21
-9*	(25)	0	0	0	25
Total	(70)	(16)	(14)	(30)	100

*It was assumed that all the fluorite in the minus 9-μm fraction is liberated.

TABLE A-8: LOCATION OF TIN IN EACH SIZE FRACTION (VALUES IN %)

Size Fraction (μm)	Tin in Garnet	Tin in Cassiterite				Total
		Liberated	Locked			
			With Magnetite	With Silicates	Total	
+75	50	0	33	17	50	100
-75+33	46	7	29	18	47	100
-33+17	46	<5	28	24	52	100
-17+9	37	6	14	43	57	100
-9*	(50)	(50)	0	0	0	100
Total	(44,5)	(11)	(20.5)	(24)	(44.5)	100

*Values for the slimes fraction are estimates only.

TABLE A-9: OVERALL DISTRIBUTION OF TIN IN THE ORE (% OF Sn IN ORE)

Size Fraction (μm)	Tin in Garnet	Tin in Cassiterite			Total
		Liberated	Locked		
			with Magnetite	with Silicates	
+75	5	0	3	2	10
-75+33	19	<1	10	7	37
-33+17	9	<1	5.5	5.5	20
-17+9	5.5	1	2	6.5	15
-9*	(9)	(10)	0	0	19
Total	(44.5)	(11)	(20.5)	(24)	100

*Values for the slimes fraction have been estimated on the basis of the overall distribution of tin between garnet and cassiterite and by assuming that all cassiterite in the slimes is liberated.

TABLE A-10: LOCATION OF TUNGSTEN IN EACH SIZE FRACTION (VALUES IN %)

Size Fraction (μm)	Liberated	Locked			Total
		with Magnetite	with Silicates	Total	
+75	50	24	26	50	100
-75+33	43	35	22	57	100
-33+17	66	23	11	34	100
-17+9	68	11	21	32	100
-9*	(100)	0	0	0	100
Total	(62)	(22)	(17)	(39)	100

*It was assumed that all the scheelite in the minus 9-μm fraction is liberated.

TABLE A-11: OVERALL DISTRIBUTION OF TUNGSTEN (% OF W IN ORE)

Size Fraction (μm)	Liberated	Locked			Total
		with Magnetite	with Silicates	Total	
+75	6	3	3	6	12
-75+33	15	13	8	21	36
-33+17	15	5	3	8	23
-17+9	8	1	3	4	12
-9*	(18)	0	0	0	18
Total	(62)	(22)	(17)	(39)	100

*It was assumed that all the scheelite in the minus 9-μm fraction is liberated.

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PLATE A-1

a to d Photomicrographs of the non-magnetic 3.1 to 3.3 sp.gr. products (all 240×).

a. -75+33 μm
b. -75+33 μm
c. -33+17 μm
d. -17+9 μm

Fluorite in the material darker than the background; bright white material in lc is pyrite and light-grey material is garnet or amphibole. Much of the fluorite in the minus 33 plus 17-μm fraction is liberated but in the minus 75 plus 33-μm material a considerable amount of locked fluorite is present.

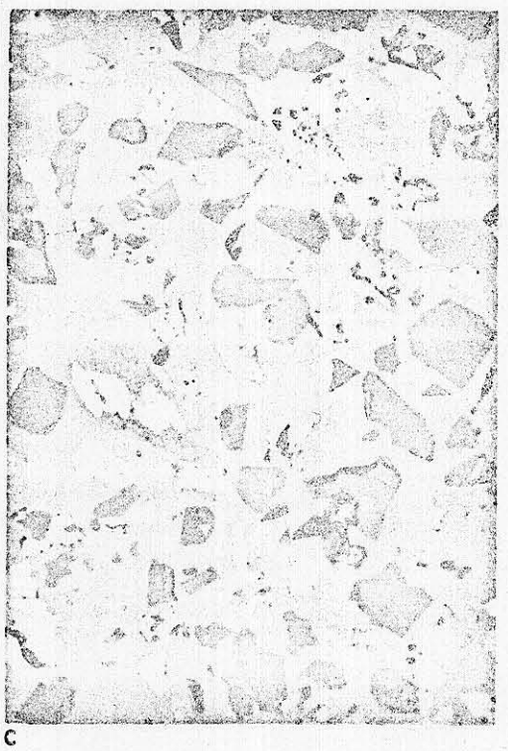


a.



b.

PLATE 1



c



d.

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PLATE A-2

a to d All are 240×.

2a. Plus 75-μm ferromagnetic product. Magnetite (bright white) forms almost liberated grains but much is locked. Complex magnetite-fluorite-garnet ternary grains are abundant.

2b. Plus 75-μm >3.3 sp.gr. non-magnetic product. Most grains consist largely of garnet but note the widely dispersed and fine-grained fluorite.

2c. Minus 33 plus 17 μm >3.3 specific gravity non-magnetic product. Largely liberated garnet and amphibole. Bright specks are pyrite. Very little fluorite (near centre and upper part).

2d. Plus 75 μm <3.1 sp.gr. non-magnetic product. Composites of garnet and quartz (similar shade to background). Fluorite in ternary (+) grains on right and lower part. Central etched grain is a carbonate mineral.

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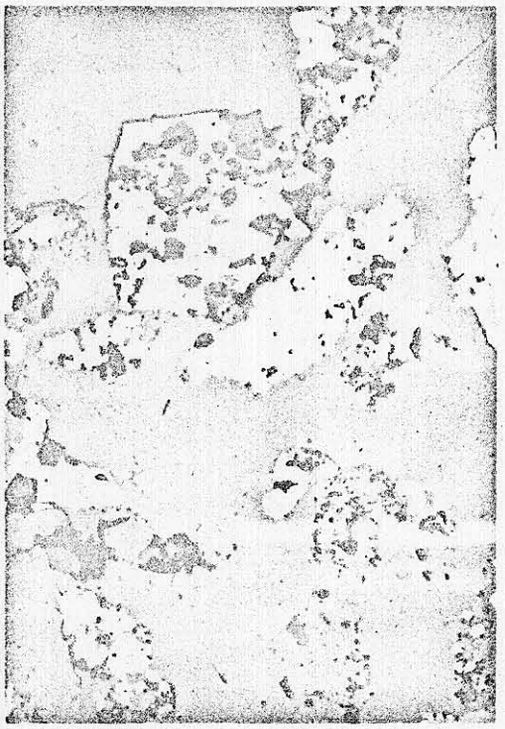

a.



b.

PLATE 2



c.



d.

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APPENDIX B

EXAMINATION OF FLUORITE CONCENTRATES FROM MOINA ORE

Two fluorite concentrates were received from the ME Section. One, the fifth cleaner concentrate Test 8, derived from flotation testwork carried out on a composite from Comalco's Moina mineralisation. The second sample, sixth cleaner concentrate Test 9, was similar to the first, but was obtained from flotation carried out with the addition of a suppressant for calcite.

Each sample was examined in temporary grain mounts and the first sample was also examined in reflected light (PS 25175) and by X-ray diffractometry.

Fifth Cleaner Concentrate (Test 8)

This sample contains approximately 88% of fluorite (by assay). The principal diluent minerals are calcite and magnetite; amphibole and scheelite crystals were also identified.

Magnetite appears mainly to be locked with fluorite and much occurs as small crystals enclosed in grains consisting essentially of fluorite. Most of the fluorite forms particles less than 15 μm in size and locked magnetite crystals are very small indeed.

The other diluent minerals form liberated, discrete grains similar in size to those of fluorite.

Sixth Cleaner Concentrate (Test 9)

This fluorite concentrate was prepared using a calcite depressant. As far as can be determined calcite is present in this concentrate at about the same concentration level as magnetite, scheelite and amphibole and each may comprise (very approximately) 1 to 3% of the concentrate. Liberated calcite amphibole and scheelite were identified. Magnetite is also present but generally in binary grains in fluorite.

In summary, therefore, some suppression of calcite has been achieved but the concentrate grade is still probably 92 to 95% CaF_2 and most of the material has been ground to particle sizes of 5 to 15 μm. Apart from calcite, magnetite, scheelite and amphibole have been specifically identified; very small amount of other gangue minerals may be present but are unlikely to contribute, together, more than 1 to 2% of these concentrates.

Investigation and Report by: B.G. Steveson

Officer in Charge, Mineralogy/Petrology Section: K.J. Henley

446

221540

TABLES 1 TO 12

FIG. 1

Symbol	Test No.	Number of Cleaning Stages	Regrind	Quebracho Addition kg/t
x	3	2	No	-
o	6	4	Yes	-
+	8	5	Yes	-
Δ	9	6	Yes	0.2

——— CaF_2 grade v. recovery.

------ Fe grade v. CaF_2 recovery.

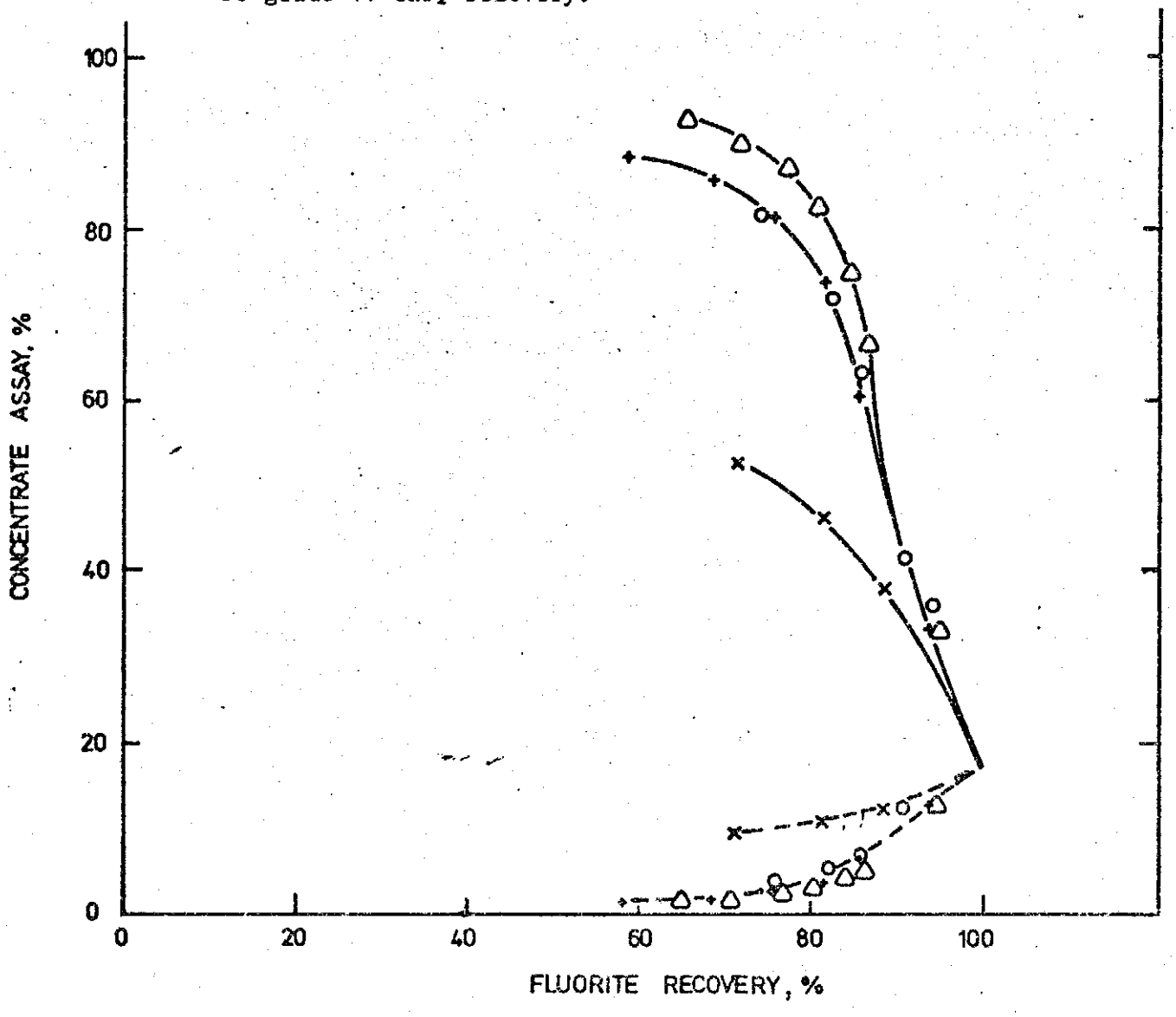

FIG.1: FLUORITE FLOTATION

TABLE 1: SEMI-QUANTITATIVE EMISSION SPECTROGRAPHIC ANALYSIS OF HEAD SAMPLE

Sample No.	Major	Minor	Heavy Trace (0.1 – 1%)	Trace (0.01 – 0.1%)	Faint Trace (0.001 – 0.01%)	Very Faint Trace (0.0001 – 0.001%)
Head Sample	Ca Fe Si	Al K Mg	Mn Na	Rb W Ti Zr Bi Cu Sn Zn	Be Ni Co Cr V Nb Li Ag Y B Pb Cs	Ge Ag

Note: Elements not sought: Hg U.

TABLE 2: SIZING OF GROUND ORE

Screen Aperture or Cyclosizer Cone		Weight Passing, Cum. %	
Nominal Size µm	Equivalent Mesh BSS	10 min. Grind*	15 min. Grind*
150	100	99.8	–
105	150	97.4	–
90	170	–	99.6
75	200	89.0	–
63	240	–	97.3
53	300	76.2	–
45	350	–	88.5
33	C1	59.2	–
25	C2	47.1	–
17	C3	36.5	–
11	C4	25.7	–
9	C5	20.5	–
-9			

*Ground in rod mill at 60% solids.

TABLE 3: DAVIS TUBE SEPARATIONS

Field Strength K gauss	Product	Product Weight %	Assay %		Distribution, %	
			CaF_2	Fe	CaF_2	Fe
1.0	Magnetic	23.9	8.9	43.6	12.8	61.1
1.0	Non-magnetic	76.1	19.2	8.7	87.2	38.9
	Head	100.0	16.8	17.0	100.0	100.0
4.7	Magnetic	25.5	9.0	43.8	13.9	64.8
4.7	Non-magnetic	74.5	19.1	8.2	86.1	35.2
	Head	100.0	16.5	17.2	100.0	100.0

TABLE 4: MAGNETIC SEPARATION IN ERIEZ WET DRUM SEPARATOR

Product	Weight	Assay, %		Distribution, %	
	%	CaF_2	Fe	CaF_2	Fe
Magnetic	45.3	11.7	28.6	33.6	74.6
Non-magnetic	54.7	19.2	8.1	66.4	25.4
Head	100.0	15.8	17.4	100.0	100.0

TABLE 5: FLOTATION TEST NO. 1
Grinding time: 10 min
Sample: Moina fluorite

Reagents				Flotation			
Name	Addition kg/t	Point of Addition	Conditioning Time min	Stage	Time min	Temp °C	pH
				Magnetic separation			
MR Sodium silicate	2.1	Cell	10	Conditioning	–	40	9.7
Collector*	0.11	Cell	1	Rougher 1	2	40	
Collector	0.11	Cell	1	Rougher 2	2	40	
Collector	0.11	Cell	1	Rougher 3	2	40	
Teric 400 (frother) used as required							

Product	Weight	Assay, %		Distribution, %	
	%	CaF_2	Fe	CaF_2	Fe
Rougher 1	10.1	61.7	4.05	39.5	2.4
Rougher 2	5.3	36.2	6.85	12.2	2.1
Rougher 3	3.9	20.1	8.65	5.0	1.9
Magnetic product	45.3	11.7	28.6	33.6	74.6
Flotation tailing	35.4	4.3	9.3	9.7	19.0
Head (calc.)	100.0	15.8	17.4	100.0	100.0

*Oleic acid 75%, Akypomine AT 25%.

TABLE 6: FLOTATION TEST NO. 4
Grinding time: 15 min
Sample: Moina fluorite

Reagents				Flotation			
Name	Addition kg/t	Point of Addition	Conditioning Time min	Stage	Time min	Temp °C	pH
				Grind			
MR Sodium silicate	1.89	Cell	1	Magnetic separation	-	-	-
Collector*	0.094	Cell	1	Rougher 1	1½	40	9.1
Collector	0.094	Cell	1	Rougher 2	1	39	9.0
Collector	0.094	Cell	1	Rougher 3	1	41	8.8
Collector	0.094	Cell	1	Rougher 4	1½	40	8.9
Collector	0.094	Cell	1	Rougher 5	1	40	8.9
MR Sodium silicate	0.31	Cell	1	1st cleaner	2½	40	9.1
				2nd cleaner	2½	40	9.2
				3rd cleaner	3	41	9.4

Product	Weight, %		Assay, %				Distribution, %			
			CaF_2		Fe		CaF_2		Fe	
	Product	Cumulative	Prod.	Cum.	Prod.	Cum.	Product	Cumulative	Product	Cumulative
3rd Clnr conc	19.0	19.0	54.0	54.0	5.0	5.0	61.4	61.4	5.5	5.5
3rd Clnr tail	1.5	20.5	30.4	52.2	8.35	5.25	2.8	64.2	0.7	6.2
2nd Clnr tail	2.2	22.7	14.4	48.6	9.9	5.7	1.9	66.1	1.2	7.4
1st Clnr tail	7.4	30.1	6.2	38.2	10.4	6.85	2.7	68.8	4.5	11.9
Magnetic	35.8	65.9	11.1	23.5	33.8	18.4	23.8	92.6	69.7	81.6
Ro tail	34.1	100.0	3.6	16.7	9.4	17.4	7.4	18.5	18.4	100.0
Head	100.0	-	16.7	-	17.4	-	100.0	100.0	100.0	-

*Oleic acid 75%, Akypomine AT 25%.

TABLE 7: FLOTATION TEST NO. 5
Grinding time: 15 min
Sample: Moina fluorite

Reagents				Flotation			
Name	Addition kg/t	Point of Addition	Conditioning Time min	Stage	Time min	Temp °C	pH
MR Sodium silicate	2.0	Mill	15	Grind			
Collector*	0.1	Cell	1	Rougher 1	3	40	9.7
Collector	0.1	Cell	1	Rougher 2	2½	40	9.5
Collector	0.1	Cell	1	Rougher 3	1	40	9.35
Collector	0.2	Cell	1	Rougher 4	1½	40	9.30
Collector	0.2	Cell	1	Rougher 5	1½	40	9.25
	-	-	-	1st Cleaner	3½	41	9.3
				2nd Cleaner	3	41	9.2
				3rd Cleaner	3	40	9.3

Product	Weight, %		Assay, %				Distribution, %			
			CaF_2		Fe		CaF_2		Fe	
	Product	Cumulative	Prod.	Cum.	Prod.	Cum.	Prod.	Cum.	Prod.	Cum.
3rd Clnr conc non-mag	26.1	26.1	37.0	37.0	7.25	7.25	60.4	60.4	18.0	18.0
3rd Conc mag	17.2	43.3	29.0	33.8	19.5	12.1	31.2	91.6	32.0	50.0
[illegible]lnr tail	4.6	47.9	8.5	31.4	17.9	12.7	2.5	94.1	7.9	57.9
[illegible]nr tail	5.6	53.5	4.6	28.6	19.3	13.4	1.6	95.7	10.3	68.2
Clnr tail	13.4	66.9	2.1	23.3	22.0	15.1	1.8	97.5	26.0	96.2
Ro tail	33.1	100.0	1.2	15.9	21.8	17.3	2.5	100.0	3.8	100.0
Head	100.0	-	15.9	-	17.3	-	100.0	-	100.0	-

*Oleic acid 75%, Akypomine AT 25%.

TABLE 8: FLOTATION TEST NO. 3
Grinding time: 10 min
Sample: Moina fluorite

Reagents				Flotation			
Name	Addition kg/t	Point of Addition	Conditioning Time min	Stage	Time min	Temp °C	pH
MR Sodium silicate	2.0	Mill		Grind			10.0
Collector*	0.1	Cell	1	Rougher 1	2	40	9.8
Collector	0.1	Cell	1	Rougher 2	2	40	-
Collector	0.1	Cell	1	Rougher 3	2	40	
				1st Cleaner	2	38	
				2nd Cleaner	1.5	38	

Product	Weight, %		Assay, %				Distribution, %			
	Product	Cumulative	CaF_2		Fe		CaF_2		Fe	
			Prod.	Cum.	Prod.	Cum.	Prod.	Cum.	Prod.	Cum.
2nd Clnr conc	22.3	22.3	52.4	52.4	9.6	9.6	71.3	71.3	12.5	12.5
2nd Clnr tail	6.6	28.9	24.7	46.1	14.6	10.7	10.0	81.3	5.7	18.2
1st Clnr tail	9.3	38.2	12.9	38.0	16.6	12.2	7.3	88.6	9.0	27.2
Rougher tail	61.8	100.0	3.0	16.3	20.1	17.1	11.4	100.0	92.8	100.0
Head (calc)	100.0	-	16.3	-	17.1	-	100.0	-	100.0	-

*Oleic acid 75%, Akypomine AT 25%.

TABLE 9: FLOTATION TEST NO. 6
Grinding time: Primary grind - 10 min
Regrind - 5 min
Sample: Moina fluorite

Reagents				Flotation			
Name	Addition kg/t	Point of Addition	Conditioning Time min	Stage	Time min	Temp °C	pH
MR Sodium silicate	2.0	Mill	10	Primary grind			
Collector*	0.1	Cell	1	Rougher 1	2½	40	9.8
Collector	0.1	Cell	1	Rougher 2	2	40	9.5
Collector	0.1	Cell	1	Rougher 3	3	39	9.4
				1st Cleaner	4½	40	9.5
MR Sodium silicate	0.2	Mill	5	Regrind (1st Clnr conc)			
				2nd Cleaner	6	40	9.5
				3rd Cleaner	4½	40	9.5
				4th Cleaner	4	41	9.5

Product	Weight, %		Assay				Distribution, %			
	Product	Cumulative	CaF_2		Fe		CaF_2		Fe	
			Prod.	Cum.	Prod.	Cum.	Prod.	Cum.	Prod.	Cum.
4th Clnr conc	15.5	15.5	81.8	81.8	3.8	3.8	73.8	73.8	3.4	3.4
4th Clnr tail	4.1	19.6	34.3	71.8	12.2	5.6	8.2	82.0	2.9	6.3
3rd Clnr tail	3.6	23.2	16.4	63.3	17.0	7.3	3.4	85.4	3.5	9.8
2nd Clnr tail	14.3	37.5	6.0	41.5	20.4	12.3	5.0	90.4	16.8	26.6
1st Clnr tail	7.3	44.8	8.2	36.0	17.7	13.2	3.5	93.9	7.4	34.0
Rougher tail	55.2	100.0	1.9	17.3	20.8	17.4	6.1	100.0	66.0	100.0
Head	100.0	-	17.3	-	17.4	-	100.0	-	100.0	-

*Oleic acid 75%, Akypomine AT 25%.

TABLE 10: FLOTATION TEST NO. 8
Grinding times: Primary grind - 10 min
Regrind - 10 min
Sample: Moina fluorite

Reagents				Flotation			
Name	Addition kg/t	Point of Addition	Conditioning Time min	Stage	Time min	Temp °C	pH
MR Sodium silicate	2.0	Mill	10	Primary grind	-	-	9.8
Collector*	0.1	Cell	2	Rougher 1	2½	40	9.5
Collector	0.1	Cell	2	Rougher 2	2½	40	9.4
Collector	0.1	Cell	2	Rougher 3	3	39	9.3
MR Sodium silicate	-	Mill	10	Regrind bulk rougher concentrate			
				1st Cleaner	4	40	9.5
				2nd Cleaner	3½	39	9.4
				3rd Cleaner	3	40	9.4
				4th Cleaner	2½	40	9.3
				5th Cleaner	2	41	9.3

Product	Weight, %		Assay, %					Distribution, %			
	Product	Cumulative	CaF_2		Fe		WO_3	CaF_2		Fe	
			Prod.	Cum.	Prod.	Cum.	Prod.	Prod.	Cum.	Prod.	Cum.
5th Clnr conc	10.7	10.7	88.4	88.4	1.4	1.4	0.56	58.4	58.4	0.9	0.9
5th Clnr tail	2.2	12.9	72.7	85.7	4.2	1.9	-	9.9	68.3	0.5	1.4
4th Clnr tail	2.1	15.0	54.0	81.3	7.2	2.6	-	7.0	75.3	0.9	2.3
3rd Clnr tail	2.8	17.8	34.3	73.9	11.0	4.0	-	6.0	81.3	1.8	4.1
2nd Clnr tail	5.1	22.9	12.3	60.3	16.1	6.6	-	3.8	85.1	4.7	8.8
1st Clnr tail	22.6	45.5	6.1	33.4	20.1	13.3	-	8.4	93.5	26.3	35.1
Ro tail	54.5	100.0	1.95	16.3	20.6	17.3	0.08	6.5	100.0	64.9	100.0
Head	100.0	-	16.8	-	17.3	-	0.106	100.0	-	100.0	-

*Oleic acid 75%, Akypomine AT 25%.

TABLE 11: FLOTATION TEST NO. 9
Grinding time: Primary grind - 10 min
Regrind - 10 min
Sample: Moina fluorite

Reagents				Flotation			
Name	Addition kg/t	Point of Addition	Conditioning Time min	Stage	Time min	Temp °C	pH
MR Sodium silicate	2.0	Mill	10	Primary grind		23	9.7
Collector*	0.1	Cell	2	Rougher 1	2½	40	9.5
Collector	0.1	Cell	2	Rougher 2	2½	40	9.3
Collector	0.1	Cell	2	Rougher 3	3	39	9.3
MR Sodium silicate	0.2	Mill	10	Regrind bulk rougher concentrate			
				1st Cleaner	4	40	9.4
				2nd Cleaner	3½	41	9.4
Quebracho	0.1	Cell	3	3rd Cleaner	4	41	9.1
				4th Cleaner	3½	41	9.1
				5th Cleaner	3¼	41	9.3
Quebracho	0.1	Cell	3	6th Cleaner	3	41	9.0

Product	Weight, %		Assay, %						Distribution, %					
	Product	Cumulative	CaF_2		Fe		WO_3		CaF_2		Fe		WO_3	
			Prod.	Cum.	Prod.	Cum.	Prod.	Cum.	Prod.	Cum.	Prod.	Cum.	Prod.	Cum.
6th Clnr conc	11.4	11.4	92.5	92.5	1.4	1.4	0.40	0.40	64.9	64.9	0.9	0.9	45.7	45.7
6th Clnr tail	1.5	12.9	69.4	89.8	5.0	1.8	0.49	0.41	6.4	71.3	0.4	1.3	7.4	53.1
5th Clnr tail	1.4	14.3	59.2	86.8	6.8	2.3	0.31	0.40	5.1	76.4	0.6	1.9	4.3	57.4
4th Clnr tail	1.5	15.8	40.7	82.4	9.3	2.9	0.26	0.39	3.8	80.2	0.8	2.7	3.9	61.3
3rd Clnr tail	2.5	18.3	24.2	74.5	11.7	4.1	0.20	0.36	3.7	83.9	1.8	4.5	5.0	66.3
2nd Clnr tail	2.8	21.1	11.3	66.1	15.3	5.6	0.09	0.32	1.9	85.8	2.6	7.1	2.4	68.7
1st Clnr tail	25.5	46.6	5.3	32.8	19.1	13.0	0.07	0.19	8.3	94.1	29.3	36.4	17.9	86.6
Rougher tail	53.4	100.0	1.8	16.3	19.8	16.6	0.025	0.10	5.9	100.0	63.6	100.0	13.4	100.0
Head	100.0	-	16.3	-	16.6	-	0.10	-	100.0	-	100.0	-	100.0	-

*Oleic acid 75%, Akypomine AT 25%.

TABLE 12: FLOTATION TEST NO. 7
Grinding time: 10 min
Sample: Moina fluorite

Reagents				Flotation			
Name	Addition kg/t	Point of Addition	Conditioning Time min	Stage	Time min	Temp °C	pH
MR Sodium silicate	2.0	Mill	10	Grind			
Collector*	0.4	Cell	1	Rougher 1	3½	40	9.7
Collector	0.2	Cell	2	Rougher 2	3	40	9.5
Collector	0.2	Cell	2	Rougher 3	3	40	9.4
Collector	0.2	Cell	2	Rougher 4	2½	40	9.3
				1st Cleaner	4½	40	9.2
				2nd Cleaner	3	40	9.2
				3rd Cleaner	2½	41	9.2

Product	Weight, %		Assay, % CaF_2		Distribution, % CaF_2	
	Product	Cumulative	Product	Cumulative	Product	Cumulative
3rd Cleaner concentrate	40.8	40.8	36.6	36.6	90.1	90.1
3rd Cleaner tail	1.5	42.3	12.9	35.8	1.2	91.3
2nd Cleaner tail	3.4	45.7	9.0	33.7	1.9	93.2
1st Cleaner tail	6.4	52.1	5.3	30.2	2.1	95 3
Rougher tail	47.9	100.0	1.6	16.6	4.7	100.0
Head	100.0	-	16.6	-	100.0	-

*Aeropromoter 845.

APPENDIX 29A

AMDEL. PYROHYDROLYSIS PROPOSAL

45.

COMALCO

221549

COMALCO LIMITED

Incorporated in Victoria

95 Collins Street
Melbourne Australia

6 June, 1978

The Director,
The Australian Mineral Development Laboratories,
Flemington Street,
FREWVILLE,
S.A. 5063.

Dear Sir,

Project Proposal No. 5/0/2862
Beneficiation and Treatment of Moina Fluorite
Amdel reference ME 3/1/4/16

During 1976 Amdel carried out some valuable work on the beneficiation and treatment of low grade fluorite "ore" from Moina in Tasmania.
The relevant reports are:-

ME 3/1/4/16	6 October, 1976 by: I.W. McPheat "Beneficiation of Moina Fluorite"
MP 3/1/4/16	28 October, 1976 by: B.G. Steveson "Treatment of Moina Fluorite" Progress Report No. 1. "Ore-dressing mineralogy of a fluorite-tin-tungsten ore composite from Moina, Tasmania."
ME 3/1/4/16	2 December, 1976 by: I.W. McPheat "Treatment of Moina Fluorite" Progress Report No. 1. Service Report 2873/76 6 July, 1976 by: J.E.A. Gooden "Fluoride Extraction from Fluorite"

This project to beneficiate the Moina Fluorite was terminated because it appeared that it was uneconomic to extract the contained fluorine.

.../2

GPO Box2773Y Melbourne 3001
Telegraphic 'Forcomal' Telex 30108
Telephone 6511211

Recently, I have read Amdel Report No. 1178 "Production of hydrogen fluoride" by J.M. Clayton dated August, 1977. In the light of this report, please would you advise me if Moina fluorite has any potential to be treated by method 3: "Production of HF by pyrohydrolysis of fluorite ore". If there is any chance that pyrohydrolysis may be able to extract the fluorine from the Moina fluorite I would like to discuss the subject with you with object of getting some more work done.

Yours faithfully,

A.H. Bartlett
Manager - Exploration

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1/1/185

August 1977

SOUTH AUSTRALIAN DEPARTMENT OF MINES

Amdel Report

No. 1178

PRODUCTION OF HYDROGEN FLUORIDE

by

J.M. Clayton

Investigated by: Operations Division

Manager, Operations Division: R.E. Wilmshurst

Brian S. Hickman Acting Managing Director

THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES

Flemington Street, Frewville South Australia 5063

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SUMMARY

Background

South Australia has been proposed as a possible site for an uranium enrichment industry which would require a source of hydrogen fluoride for the production of uranium hexafluoride. The South Australian Mines Department requested a study of methods available for producing hydrogen fluoride.

Summary of Work Done

The costs of producing hydrogen fluorides by 5 alternative methods have been estimated. The methods are:

1. Treatment of imported acid-grade fluorspar in existing HF plants.
2. Beneficiation of Australian fluorite to acid-grade and treatment in existing HF plants.
3. Production of HF by pyrohydrolysis of fluorite ore.
4. Production of HF by aluminium sulphate leaching of fluorite ore.
5. Production of HF by roasting ammonium sulphate with fluorite ore.

The cost of production by the above methods are estimated as follows (on the basis of delivery of HF to Port Pirie):

Method	1	2	3	4	5
Cost $/t HF	820	910	333	787	848

Conclusions and Recommendations

The most attractive method for producing anhydrous hydrogen fluoride is the pyrohydrolysis process which could produce HF at less than half the existing price. However, the process is not proven commercially and requires experimental verification.

It is recommended that a programme of experimental work be undertaken to acquire better knowledge of the pyrohydrolysis process. Aspects to be studied should include the limiting temperature to avoid fusion during roasting, the optimum quantities and type of silica to be added to react with the calcium, the optimum ratio of steam required, and methods for recovery of HF from the reaction gases.

1. INTRODUCTION

South Australia has been proposed as a possible location for a uranium enrichment plant. The process considered most attractive for enrichment is the centrifuge process, which uses uranium hexafluoride, UF_6, as feed material. Existing technology requires that the 'tails' product from the enrichment plant, containing natural uranium and a depleted proportion of fissile uranium-235 is stored in cylinders as UF_6. The enrichment process therefore consumes considerable fluorine which is not available for recycle.

Uranium hexafluoride is produced by the successive reaction of hydrogen fluoride and elemental fluorine (produced from hydrogen fluoride) upon a purified uranium dioxide. The quantity of hydrogen fluoride required for the proposed enrichment complex would involve a major expansion by existing producers or the construction of a new plant.

On behalf of the South Australian Department of Mines, Amdel is investigating the feasibility of producing hydrogen fluoride from Australian resources. As Stage 1 of the investigation, Allen (1976)* reviewed the sources of fluorine in Australia and the various processes available for the recovery of fluorine. His study concluded that of the various alternatives available, the following processes were most likely to be competitive with the existing method, in which imported acid-grade fluorspar is reacted with sulphuric acid.

1. Production of an acid-grade fluorspar from Australian fluorite ores by physical beneficiation, followed by reaction with sulphuric acid.

2. Production of hydrogen fluoride from a fluorite ore via one of three processes which have been demonstrated at laboratory or pilot scale but not commercially.

3. Recovery of fluorine from the exhaust gases of a phosphate fertiliser plant via one of three processes which have been demonstrated at laboratory or pilot scale but not commercially.

The second stage of the investigation consists of an economic comparison of the following processes for producing hydrogen fluoride.

---

*ALLEN (1976). Amdel Report 1135.

1. The reaction of imported acid-grade fluorspar with sulphuric acid by expansion of existing plants.

2. The production of acid-grade fluorspar from Australian fluorite ores for subsequent reaction with sulphuric acid at the existing (expanded) plants.

3. The chemical treatment of Australian fluorite ore via one of the following processes.

   A. Pyrohydrolysis
   B. Aluminium sulphate leaching
   C. Roasting with ammonium sulphate.

The study was also intended to include the costs of processes for the recovery of fluorine from phosphate fertiliser plant exhaust gases, but it has not been possible to include these costs within the time and finance available.

221556

2.3 Chemical Treatment of Fluorite Ores

The three processes described in this section are based upon laboratory and/or small pilot-scale demonstrations. No commercial operations are known. Parts of the processes have not been demonstrated in conjunction with the principal reaction steps and although they are assumed to be feasible, may require verification in particular types of process equipment. The reaction conditions may need to be varied for any particular ore, according to its mineral content.

2.3.1 Pyrohydrolysis

A schematic flow sheet is given in Fig. 3.

Run-of-mine fluorite ore is recovered from stockpile and crushed to pass 10 mm. The crushed ore is reacted with steam at 1,000 to 1,300°C in an externally heated reactor. A rotary kiln should be suitable. The reaction may require a residence time of from 1 to 4 h to achieve 85 to 90% extraction of fluorine.

$$CaF_2 + SiO_2 + H_2O \rightarrow CaSiO_3 + H_2F_2$$

With some ores the reactivity may be improved by the addition of silica. If the stoichiometric quantity of steam is sufficient to achieve the required fluorine extraction, the off-gases may be simply cooled and anhydrous HF condensed. However, if excess steam is necessary, this will also condense and the condensate will need to be redistilled over concentrated sulphuric acid to produce the anhydrous HF.

In either case, the off-gas from the condensers is passed through a lime scrubber to remove residual HF. Spent lime is recycled to the kiln. Steam required for the reaction is produced by a waste-heat boiler situated in the combustion gas outlet duct. Further heat recovery is achieved by preheating combustion air.

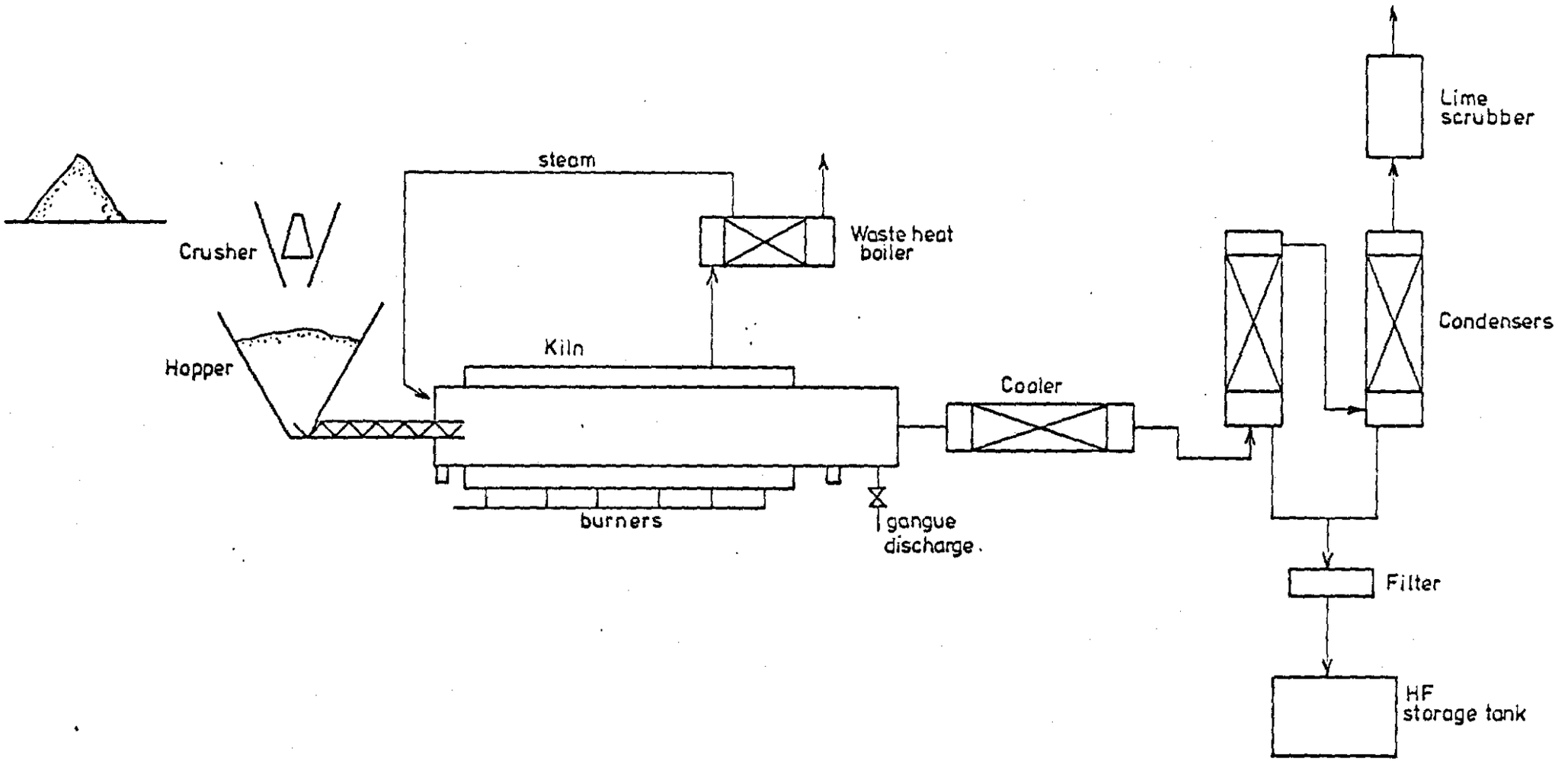

FIG. 3: SCHEMATIC FLOW DIAGRAM - PYROHYDROLYSIS

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amdel

The Australian Mineral Development Laboratories

Flemington Street, Frewville, South Australia 5063
Phone Adelaide 79 1662, telex AA 82520

Winner of Award for Outstanding Export Achievement, 1975

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Associated with: Professional Consultants Australia Pty Ltd

Please address all correspondence to Frewville.

221558

In reply quote: OD 5/0/3175

4 July 1978

The Manager-Exploration
Comalco Limited
GPO Box 2773Y
MELBOURNE Vic 3001

Dear Mr Bartlett,

Treatment of Moina Fluorite

Please forgive our delay in replying to your letter of 6 June 1978.

Yes, we believe pyrohydrolysis would be an effective means of recovering fluorine from Moina fluorite. In fact, we are enthusiastic about the potential of a process which yields HF directly with such a simple flowsheet, and which uses steam as the only reagent. The principal problems are likely to concern the engineering of a reactor to withstand a corrosive atmosphere at temperatures of 1100°C or higher. Initially we would envisage some laboratory tube furnace experiments to decide optimum conditions of temperature, time, particle size etc. for the Moina ore. A small, specially-constructed fluid bed reactor would then be used for trials which should aid in materials selection and with scale-up considerations.

We would be pleased to discuss these suggestions with you in more detail at a time you nominate.

Yours faithfully,
The Australian Mineral Development
Laboratories

for Norton Jackson,
Managing Director

RJA:rp

221559

APPENDIX 30.

DEPARTMENT OF MINES INVESTIGATION R 731.

EXPLORATORY INVESTIGATION, MOINA WRIGGLITE

466

221560 w/
EL 7/74 MOINA BENEFICIATION

Comalco Exploration,
P.O. Box 691,
Devenport. Tas.7310

31st August, 1976.

Mr. H.K. Wellington,
Chief Metallurgist,
Department of Mines,
Wellington Street,
LAUNCESTON. Tas.7250.

Dear Mr. Wellington,

Re: Beneficiation Study Moina Skarn.

During the discussion between you and Mr. A.H. Bartlett and myself last Thursday, 26th August, 1976, you indicated that about 100 kg of the fluorite skarn ("wrigglite") would be necessary for your investigations.

I will personally deliver to you a parcel of about 140 kg of "wrigglite". The parcel is made up of cobbles and boulders selected off the dump about 50 metres NNW of the old main shaft of the Shepherd and Murphy Mine and off a bulldozer costeam spoil heap immediately adjacent to the old main shaft.

The proportions of rock types in the parcel were selected to be representative of the skarn body as a whole.

All the cobbles and boulders will need to be broken up, crushed and mixed to achieve uniformity. If you wish I can arrange head grade analyses of the bulk sample after it is mixed and prepared. Our Melbourne Laboratory is well experienced in total F analyses, and Amdel have a special method for determining soluble fluorite (CaF_2).

We hope your investigations are successful and fruitful. Of course, if I can assist with aspects of the geology, or extra samples, etc, please do not hesitate to contact me.

Yours faithfully,

P.W. Askins
Senior Geologist.

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OS. 1144

221561

DEPARTMENT OF MINES—TASMANIA

TELEPHONES:
Metallurgical Research Laboratory .. .. .. } 44 2431-2 (2 lines)
Mines Inspection .. .. }
Explosives & Inflammable Liquids .. ..

LAUNCESTON OFFICES
287 WELLINGTON STREET
SOUTH LAUNCESTON 7250

3rd September, 1976.

Comalco Exploration,
P.O. Box 691,
DEVONPORT, 7310
Tasmania.

Attn: Mr. P.W. Askins

Dear Sir,

R.731: Moina Wrigglite Oredressing

Following receipt of your sample 761736 of wrigglite from dumps etc around the Shepherd and Murphy Mine we have begun a scout testing program a copy of which is attached for your information.

This scout program is a preliminary examination to allow us to get a better appreciation of the problem before proposing a more detailed investigation.

We have now progressed with the scout program to para. 3 and have samples 3b and 3c available for disposal as you direct.

Yours faithfully,

(H.K. Wellington)
Chief Chemist & Metallurgist

Enc.

468

221562

R.731

Company: Comalco Ltd.

Aim: Scout testing for further information on which to base oredressing program.

Sample: 761736 which is handpicked wrigglite from old mine waste and prospecting costeans.

Source: Sample was supplied by P.W. Askins of Comalco who selected it from the Shepherd and Murphy mine area at Moina.

Method:

1. Crush to pass 12.7 mm (½") screen, mix well then riffle out about 1 kg. Screen this 1 kg on 1 mm and use oversize for heavy liquid separation at density 2.95 t/m³.

2. On material not used for H/L
   a crush to all pass 3.17 mm (⅛")
   b riffle out an assay head sample and 1 kg lots for scout tests.

3. Pulverize head sample, riffle into 3 parts
   a assay one part for Ca,Mg,Fe,SiO_2,Ti,Al,F,CO_2,S, Sn,Zn,Pb,Cu,Bi,Mo,Sb,W,L.O.I.,Insol.,(Au & Ag;AAS)
   b another part will be sent Comalco for analysis
   c reserve third part for possible analysis by AMDEL should Comalco request it.

4. Scout tests. Grind to 95% passing 600 µm for series A, 95% passing 150 µm for series B and 95% passing 38 um for series C.
   a do magnetic separations on Series A,B & C and assay M/A & N for F, Sn,W, S.Fe.
   b do tabling on series B & C (& A or perhaps jig) and assay C,M & T for F, Sn,W, and Davis tube.
   c do sulphide froth flotation on series B & C and perhaps table flotation on series A coarse fractio Assay sulphide products for S,Fe,Zn,Pb,Cu,Bi, Mo,Sb,F and Davis tube and on tailings do non sulphide flotation using oleic acid,sulphonated castor oil and tannic acid as principal reagents. Assay products for F,Sn,W, and Davis tube.

(H.K. Wellington)
Chief Chemist & Metallurgist

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EL 7/74 MOINA BENEFICIATION

221563

WW

Comalco Exploration,
P.O. Box 691,
Devonport. Tas. 7310

12th October, 1976.

Mr. H. Wellington,
Chief Chemist,
Department of Mines,
287 Wellington Street,
LAUNCESTON. TAS.7250.

Dear Mr. Wellington,

R731 Moina Wrigglite Ore Dressing.

At last we have received some analyses from Amdel of the head sample R731. They are:-

CaF_2	20.0%
Sn	2250 ppm
W	1400 ppm
Bi	350 ppm
Mo	115 ppm
Be	30 ppm (revised to 200 ppm)
Cu	120 ppm
Pb	110 ppm
Zn	810 ppm
Au	0.035 ppm

No analyses have yet been received from our Melbourne Lab.

Yours sincerely,

P.W. Askins.

Senior Geologist.

470

221564 7/74 MOINA BENEFICIATION

WJ

Exploration Department,
P.O. Box 691,
DEVONPORT. TAS.7310.

22nd November, 1976.

Mr. H.K. Wellington,
Chief Metallurgist,
Department Mines,
Wellington Street,
LAUNCESTON. TAS. 7350.

Dear Mr. Wellington,

Attached are analyses from our laboratory for the head sample R731 (fluorite skarn).

They agree well with Amdel's previous analyses, except our lab reports lower W values. Note that Amdel reports analyses as CaF_2% and our lab as F% - doubling the F% gives a close approximation to CaF_2%.

Yours sincerely,

P.W. Askins.

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Sample: Moina, Pulp (Head Sample for Tas. Mines Dept. Bone. study). Report No.: AM-15 221565

Submitted by: P.A. Requisition No.: 704 Comm. - Cont. - Conc. ✓

~~ANALYSIS~~

SAMPLE →		R 731.			Std. Moina 1.		
ANALYSIS ↓							
SiO_2		14.2 %			20.2 %		
Mg		1.4			1.7		
Ca		15.2			16.8		
Fe		20.7			20.6		
Al		3.9			8.9		
F		9.9			12.3		
		ppm.			ppm		
Zn		820			730		
Pb		75			20		
Cu		115			125		
Sb		45			50		
Bi		400			675		
Mo		190			90		
W		800			870		
Au.		0.04			0.08		

Copies to: 1. P.A. (Tas) 2. A.H.B. Chief Chemist: [signature]

3. WHA. 4. JCN/JAE Date: 15/11/76

472 R.P. 1144 BENEFICIATION 12.

DEPARTMENT OF MINES—TASMANIA

221566

TELEPHONES:
Metallurgical Research .. ..
Laboratory .. .. ..
Mines Inspection .. .. ..
Explosives & Inflammable Liquids
44 2431-2 (2 lines)

LAUNCESTON OFFICES
287 WELLINGTON STREET
SOUTH LAUNCESTON 7250

3rd December, 1976.

Comalco Ltd.,
P.O. Box 691,
DEVONPORT, 7310
Tasmania.

Attn: P.W. Askins

R.731: Moina Wrigglite Project

The following is a summary of work so far in this exploratory project.

Heavy Liquid Separation A separation has been made on a sample passing a 12.8 mm screen at a density of 2.95 t/m^3 the minus 1 mm material not being treated. There was no significant separation.

Grinding Three grinding conditions have been used in the following tests. The nominal sizing of the ground products are:-

Grind	Sizing at which 95% passes
A	600 µm
B	150 µm
C	38 µm

Magnetic Separation

Grind	Magnetic Fraction Assay %S.Fe*	Iron Distribution in the Magnetic Fraction(%)
A	37.8	76
B	44.9	73
C	56.8	69

In Grind 'C' the non magnetic fraction contained about 90% of the fluorite and assayed 9.9% F. (The head assay 7.8%F.)

* S.Fe = HCl soluble Fe.

Gravity Concentration

No significant results although in all grinds the table tailings assay higher in fluorine.

Flotation

Grind B. In a sulphide flotation the sulphide concentrate removed assayed:-

Bi	0.5%	S	25%
Zn	2.5	Fe	30
Pb	0.3	Mo	0.5
Cu	0.4	Sb	50 g/t.

Fluorite flotation yielded concentrates grading 14% F using oleic acid and sulphonated castor oil. Tannic acid was not as successful yielding only a 9% F grade concentrate.

Grind C produced poor results than Grind B due to the large quantities of slime present.

General Comment As I have explained to you on the telephone we are having troubbe with fluorine analyses and until this is rectified we can not publish the full results. As fluorine concentration is the main object of the exercise the results can not be properly assessed until the fluorine distribution is known.

Yours faithfully,

(H.K. Wellington)
Chief Chemist & Metallurgist

474 R.P. 1144

BENEFICIATION 221568

DEPARTMENT OF MINES—TASMANIA

TELEPHONES:
Metallurgical Research .. .. } 44 2431-2 (2 lines)
Laboratory .. .. ..
Mines Inspection .. .. ..
Explosives & Inflammable Liquids

LAUNCESTON OFFICES
287 WELLINGTON STREET
SOUTH LAUNCESTON 7250

22nd December, 1976.

Comalco Exploration,
P.O. Box 691,
DEVONPORT, 7310
Tasmania

copy to: Comalco Ltd.,
G.P.O. Box 2773Y,
MELBOURNE, 3001
Victoria.
Attn: Mr. A.H. Bartlett

Dear Sirs, R.731: Moina Wrigglite Oredressing

Please find attached a copy of the above report which covers scout testing at various degrees of grinding using a variety of oredressing methods.

This whole project has been delayed by difficulties experienced in fluorine analysis. The method has been rather long and involved hence repetition has consumed much time. Even now the results are not satisfactory hence the rounding of the fluorine distribution figures. It was felt better to issue the report now rather than await satisfactory fluorine results.

However we expect to rectify this deficiency as we have now received a fluoride electrode which will be used in conjunction with your method for fluorine.

Although the gravity tests do not show any meaningful results the fine grind necessary for magnetite removal would be too fine for the gravity recovery of cassiterite and scheelite.

It may be best to ignore the minor constituents and aim initially for a fluorite concentrate. When the feasibility of this has been demonstrated the recovery of minor metals could be undertaken. I should like to hear from your oredressing people on this and the economic aspects of the metals concerned.

Yours faithfully,

(H.K. Wellington)
Chief Chemist & Metallurgist

ABBREVIATIONS

Symbols used in metallurgical reports

Product symbols

A	Anode	L	Liquid	S/D	Solid
C	Concentrate	M	Middling	S/K	Sink
C/E	Calcine	M/A	Magnetic	S/N	Solution
D	Discharge	N	Non-magnetic	T	Tailing
E	Extract	O/F	Overflow	U/F	Underflow
F	Fraction	O/S	Oversize	U/S	Undersize
F/D	Feed	P	Precipitate	#	Mesh (B.S.S.)
F/T	Float	R	Residue	µm	Microns
G	Gas	R/E	Raffinate	+	Retained on screen
H	Head	R/X	Reflux	-	Passed through screen
K	Cathode	S	Spigot		

Operation symbols

B	Batch	H/L	Heavy Liquid Separation
B/M	Ball Mill	H/M	Heavy Media Separation
C	Cyclone or Classifier	H/S	Hydrosizer
C/E	Cradle	I/S	Infrasizer
C/H	Crush	I/X	Ion Exchange
C/L	Column	J	Jig
C/N	Continuous	L	Leach
C/S	Cyclosizer	L/L	Liquid/Liquid Extraction
C/R	Condenser	M	Mixer
D	Tilting Deck or Strake	M/S	Magnetic Separation
D/N	Distillation	P	Panning
D/R	Dryer	P/N	Precipitation
E	Electrostatic Separation	R	Roasting
E/F	Elutriation Fraction	R/M	Rod Mill
E/S	Electrolysis	S	Screening or Sieving
F	Flotation	S/B	Stamp Battery
F/C	Filter or Centrifuge	T	Table
G	Grind	T/H	Thickener
		V	Vanner

Chemical symbols

The normal chemical symbols for the elements and formulae for compounds are used, with the addition of the following:

C.Sn	Cassiterite Tin
S.Sn	Acid soluble tin (Stannite Tin)
T.Sn (or Sn)	Total tin (C.Sn + S.Sn)
S.Fe	Acid (HCl) Soluble iron
T.Fe (or Fe)	Total Iron

Use of symbols

(1) Where there are a number of products from the one operation, *e.g.* magnetic products, spigot products, these are distinguished by numbers, *e.g.* M/A1, M/A2, M/A3, S3, S4 etc.

(2) Where there are a number of operations, these are numbered, *e.g.* Two jigging stages J1 and J2, flotation stages F2, F3.

(3) Products from operations are described by compounding operation and product symbols thus: F4C, C/S4 means the cyclosizer fraction 4 from the concentrate made in flotation stage 4.

Note: Research projects are given a number prefixed with R, *e.g.* R.606. The tests done in the project are numbered N1, N2 etc.

476 R.P. 1144

221570

DEPARTMENT OF MINES—TASMANIA

TELEPHONES:
Metallurgical Research .. .. .. } 44 2431-2
Laboratory .. .. .. } (2 lines)
Mines Inspection .. .. .. } (2 lines)
Explosives & Inflammable Liquids }

LAUNCESTON OFFICES
287 WELLINGTON STREET
SOUTH LAUNCESTON 7250

21st December, 1976.

R.731

Introduction

A sample of wrigglite ore selected from surface dumps at the Shepherd and Murphy mine, Moina was submitted by Comalco Ltd., for concentration tests to recover the fluorite which occurs as very fine grains disseminated throughout the mass.

As it was envisaged that concentration would not be simple a series of scout tests applying normal concentration methods at three degrees of grinding was done to get an appreciation of the ore.

Sample

1. Chemical analysis of the head sample gave the following results.

Element etc.	Percent
S.Fe	21.2
acid insoluble	26.0
loss on ignition	4.36
Mg	1.1
SiO_2	28.5
Ti	0.08
Al	4.7
CO_2	0.33
S	0.55
F (as CaF_2)	7.8 (see method, para 7)
WO_3	0.14
Sn	0.24
Zn	880 g/t
Pb	150 "
Bi	440 "
Sb	24 "
Cu	140 "
Mo	120 "
Au	0.6 "
Ag	1.7 "

2. Davis tube separation of the head sample gave a result of 26 percent of magnetic material at a nominal size of grind of 90 per cent -45µm.

Method

1. Sample preparation: The 'as received' material was crushed to pass 12.7mm mixed, and then riffled to produce a sample weighing approximately 6 kg for heavy liquid separation tests. The remainder of the material was then crushed to pass 3.17mm, mixed and then riffled to produce a head sample and samples for scout tests.

..../2

Method

2. Size of grind: Foreeach particular scout test, a size of grind was chosen as shown below

designation	size of grind
A	95% passing 600µm
B	95% passing 150µm
C	95% passing 38µm

3. Heavy liquid separation: Asample of -12.7mm +1mm ore was subjected to heavy liquid separation at a density of 2.96 t/m^3

4. Magnetic separation: 1 kg lots of ore, ground to a size of grind of A, B or C were magnetically separated on a crockett wet magnetic separator with primary magnetics being subjected to one stage of cleaner magnetic separation.

5. Gravity concentration: 3 kg lots of ore, ground to a size of grind of A, B or C, were sized, classified, and then gravity concentrated on either a Diester sand or slime table.

6. Flotation: 1 kg lots of ore, ground to a size of grind of B or C, were subjected to bulk sulphide flotation followed by fluorite flotation using one of the following reagents in each test.

(a) Oleic acid
(b) Sulphonated castor oil
(c) Tannic acid

7. Fluorine analysis. The fluorine determinations used in this report are for the fluorine present as fluorite. However the method has not yielded consistent results hence fluorine figures in this report should be taken as indicative only.

Results Obtained

1. H/L Separation of -12.7mm ore at 2.95 t/m^3

Fraction	Per Cent					Per Cent Distribution			
	Mass	Davis Tube M/S M/A	S.F	Sn	WO_3	Davis Tube M/S M/A	S.F	Sn	WO_3
H/L S/K	82.5	27.0	7.8	0.27	0.13	87.3	82	83.0	73.5
H/L F/T	6.0	6.0	8.6	0.24	0.26	1.4	7	5.4	10.7
-12.8m+1mm head	88.5	(25.6)	(7.9)	(0.27)	(0.14)	88.7	89	88.4	84.2
-1mm untreated	11.5	25.0	7.6	0.27	0.20	11.3	11	11.6	15.8
Head -12.8mm	100	(25.5)	(7.8)	(0.27)	(0.15)	100	100	100	100
Head Analysis		26.0	7.8	0.24	0.14				

S.F = Fluorite fluorine

From the above, it can be seen that heavy liquid separation at 2.95 t/m^3 of -12.7mm +1mm ore gives neither significant concentration of fluorite in sink nor float. A decrease in concentration of magnetic in the 6 percent mass of float does however occur.

../3

Results Obtained

2. Crockett Magnetic Separation

Grind	Fraction	Percent			Percent Distribution	
		Mass	S.F	S.Fe	S.F	S.Fe
A	M/S M/A1	41.9	5.4	37.8	30	76.1
	M/S M/A2	6.4	9.2	14.1	8	4.3
	M/S N N	51.7	9.2	7.9	62	19.6
	HEAD	100	(7.6)	(20.8)	100	100
B	M/S M/A1	34.1	4.2	44.9	18	72.7
	M/S M/A2	9.0	8.8	16.0	10	6.8
	M/S N	56.9	10.2	7.6	72	20.5
	HEAD	100	(8.0)	(21.1)	100	100
C	M/S M/A1	25.8	1.4	56.8	5	68.6
	M/S M/A2	4.8	8.6	11.3	5	2.5
	M/S N	69.4	9.9	8.9	90	28.9
	HEAD	100	(7.6)	(21.4)	100	100

Magnetic separation of the material at a size of grind of C, (95% -38μm) has been virtually ideal. That is, the percent mass of magnetics equals the Davis tube head value for magnetics of 26 percent. The fluorite content of the magnetics in each test has decreased as the size of grind has become smaller in particle size. Grind C magnetics contains only 5 percent of the overall fluorite compared with 30 percent for grind A magnetics.

.../4

3. Gravity Concentration

Grind	Fraction	Per Cent Mass	Per Cent Davis Tube M/A	Per Cent S.F	Per Cent Sn	Per Cent WO_3	Percent Dist. Davis Tube M/A	Percent Dist. S.F	Percent Dist. Sn	Percent Dist. WO_3
	+500µm untreated	8.6	23.0	6.8	.23	.10	7.9	8	7.1	4.8
A	-500µm+295µm T/C	1.7	41.0	5.7	.28	.32	2.8	1	1.7	3.0
	-500µm+295µm T/T	14.3	23.0	5.8	.25	.12	13.1	11	12.9	9.4
	HEAD-500µm+295µm	16.0	(24.9)	(5.8)	(.25)	(.14)	15.9	12	14.6	12.4
	+295µm+150µm T/C	1.5	47.0	4.4	.32	.69	2.8	1	1.7	5.7
	-295µm+150µm T/T	7.2	27.0	6.4	.29	.16	7.7	6	7.6	6.3
	HEAD-295µm+150µm	8.7	(30.4)	(6.1)	(.30)	(.25)	10.5	7	9.3	12.0
	C/O T/C	3.5	76.0	1.7	.46	1.2	10.6	1	5.8	23.1
	C/O T/M1	22.5	37.0	8.5	.28	0.13	33.1	25	22.8	16.1
	C/O T/M2	23.9	9.0	8.0	.25	.10	8.6	25	21.6	13.1
	C/O T/T	16.8	20.0	10.7	.31	.20	13.4	23	18.8	18.5
	HEAD C/O	66.7	(24.7)	(8.5)	(.29)	(.19)	65.7	74	69.0	70.8
	HEAD, GRIND A	100	(25.1)	(7.7)	(.28)	(.18)	100	100	100	100
	+150µm Untreated	1.7	27.0	6.1	.25	.17	1.8	1	1.6	1.6
	C/O T/C	4.7	71.0	1.6	.47	1.1	13.1	1	8.2	29.3
B	C/O T/M1	16.0	46.0	6.2	.28	.18	28.8	13	16.6	16.3
	C/O T/M2	48.6	24.5	7.2	.24	.09	46.6	46	43.4	24.8
	C/O T/T	29.0	8.5	10.3	.28	.17	9.7	39	30.2	28.0
	C/O HEAD	98.3	(25.5)	(7.7)	(.27)	(.18)	98.2	99	98.4	98.4
	HEAD, GRINDB	100	(25.5)	(7.7)	(.27)	(.18)	100	100	100	100
	C/O T/C	8.5	62.5	3.9	.47	.62	21.1	4	14.5	32.7
	C/O T/M1	28.7	47.0	5.3	.23	.08	53.5	20	24.0	14.2
C	C/O T/M2	17.2	16.0	9.3	.24	.10	10.9	20	15.0	10.7
	C/O T/T	45.6	8.0	9.5	.28	.15	14.5	56	46.5	42.4
	HEAD GRIND C	100	(25.2)	(7.8)	(.27)	(.16)	100	100	100	100

Gravity concentration has achieved little in respect to separation of any specific mineral. Concentrations of magnetite have generally occurred in gravity concentrates but the same cannot be stated for cassiterite or scheelite. Fluorite, on the other hand has been slightly concentrated in the gravity tailings.

.../5

4. Flotation (a) GRIND B

Test No. and Reagent	Fraction	Per Cent Mass	WO_3	Sn	Davis Tube M/A	S.F	Bi	Zn	Pb	Cu	S	Fe	Mo	Sb	Percent Distribution WO_3	Sn	Davis Tube M/A	S.F
	CFC	24.7	.14	(.30)	18.0	14.3									23.3	25.1	17.5	47
N1	F3T	18.1	.20	.33	27.0	5.9									24.4	20.3	19.2	14
Oleic	(F2C)	42.8	(.17)	(.31)	(21.8)	(10.7)									47.7	45.4	36.7	61
Acid	F2T	55.4	.14	.29	29.0	5.4									52.3	54.6	63.3	39
	(F1T)	98.2	(.15)	(.30)	(25.9)	(7.7)									100	100	100	100
	FC Sulphides	1.8	-	-	-	-	.48	2.33	.26	.35	24.2	29.2	.47	50ppm	-	-	-	-
	HEAD	100																
	CFC	23.2	.21	.30	16.0	13.4									33.2	23.2	14.3	40
N2	F3T	18.1	.13	.32	28.0	6.6									16.0	19.3	19.5	15
Sulphon-	(F2C)	41.3	(.17)	(.31)	(21.3)	(10.4)									49.2	42.5	33.8	55
ated	F2T	57.4	.13	.30	30.0	6.0									50.8	57.5	66.2	45
Castor	(F1T)	98.7	(.15)	(.30)	(26.3)	(7.8)									100	100	100	100
Oil	FC Sulphides	1.3	-	-	-	-	.61	3.27	.38	.42	30.5	32.4	.67	51ppm	-	-	-	-
	HEAD)	100.0																
N3	CFC	1.6	.16	.30	-	9.3									1.7	1.8	-	2
Tannic	F3T	5.3	.15	.27	11.0	10.1									5.4	5.4	-	7
Acid	(F2C)	6.9	(.15)	(.28)	-	(9.9)									7.1	7.2	-	9
	F2T	91.3	.15	.27	27.0	7.8									92.9	92.8	-	91
	(F1T)	98.2	(.15)	(.27)	-	(7.9)									100	100	-	100
	FC Sulphides	1.8	-	-	-	-	.54	2.53	.27	.39	26.7	31.6	.52	37ppm	-	-	-	-
	HEAD	100																

(b) GRIND C

Test No. and Reagent	Fraction	Per Cent												Percent Distribution		
		Mass	WO_3	Sn	S.P.	Bi	Zn	Pb	Cu	S	Fe	Mo	Sb	WO_3	Sn	S.P.
N1 Oleic Acid	CFC	31.3	.17	.29	11.0									36.7	33.1	46
	F3T	27.9	.15	.30	6.9									28.8.	30.5	26
	(F2C)	59.2	(.16)	(.29)	(9.1)									65.5	63.6	72
	F2T	35.7	.14	.28	5.8									34.5	36.4	28
	(F1T)	94.9	(.15)	(.29)	(7.8)									100	100	100
	FC Sulphides	5.1	-	-	-	.28	1.00	0.10	0.18	9.5	18.1	.38	28ppm	-	-	-
	HEAD	100.0														
N2 Sulphon-ATED Castor Oil	CFC	20.7	.18	.27	12.3									25.1	21.2	33
	F3T	39.0	.15	.28	7.4									39.3	41.5	37
	(F2C)	59.7	(.16)	(.28)	(9.1)									64.4	62.7	70
	F2T	37.8	.14	.26	6.1									35.6	37.3	30
	(F1T)	97.5	(.15)	(.27)	(7.7)									100	100	100
	FC Sulphides	2.5	-	-	-	.54	1.87	.18	.37	19.4	24.0	.19	22ppm	-	-	-
	HEAL	100.0														
N3 Tannic Acid	CFC	5.3	.15	.31	8.8									5.4	5.6	4
	F3T	22.9	.16	.28	9.1									24.7	22.9	26
	(F2C)	26.2	(.16)	(.28)	(9.1)									28.1	26.5	30
	F2T	71.0	.15	.29	8.0									71.9	73.5	70
	(F1T)	97.2	(.15)	(.29)	(8.5)									100	100	100
	FC Sulphides	2.8	-	-	-	.47	1.57	.17	.34	16.9	22.2	.35	34ppm	-	-	-
	HEAD	100.0														

Note: assuming very little WO_3, Sn, Magnetic material or S.F., were in sulphide concentrates, distribution figures are quoted.

The bulk sulphide flotation concentrate contained 78% of the sulphides in Grind B and 87% in Grind C.

Fluorite flotation results revealed that a maximum recovery of 47 percent could be achieved at a grade of 14.3 percent of soluble fluorine through use of oleic acid as collector. Tannic acid was found tc be of no value as a fluorite collector and is better known as a depressant for iron oxides in metallic oxide flotation. Sulphonated castor oil as a collector for fluorite was found to be inferior to oleic acid giving lower recovery figures.

Conclusions

1. Fine grinding will be necessary.

2. Gravity concentration may be effective after removal of the magnetite but some compromise is indicated as too fine a grind would slime the cassiterite and scheelite and hence preclude recovery by gravity.

Metallurgist..K Austin.....

(H.K. Wallington)
Chief Chemist & Metallurgist

221577

APPENDIX 31.

DEPARTMENT OF MINES INVESTIGATION R 745.

SCHEELITE CONCENTRATION PROJECT.

484

OS. 1144

cc 1974 MOINA BENEFICIATION

16-
W

DEPARTMENT OF MINES—TASMANIA

221578

LAUNCESTON OFFICES
287 WELLINGTON STREET
SOUTH LAUNCESTON 7250

TELEPHONES:
Metallurgical Research
Laboratory
Mines Inspection
Explosives & Inflammable Liquids
44 2431-2 (2 lines)

28th March, 1977

Comalco Ltd.,
Exploration Department,
P.O. Box 691,
DEVONPORT, 7310
Tasmania.

Dear Sir, R745: Moina Wrigglite

As discussed with Messrs Bartlett and Askins it is planned to conduct this next stage as a tungsten concentration project but to evaluate the by-products for fluorite recovery.

2 As an initial approach products from R731 will be further examined and a further kilogram magnetic separation done to decide at what sizing the first magnetic separation should be made. This work as outlined in the attached project sheet for Stage I is proceeding.

3 It is envisaged that in Stage 2 initial magnetic separation will be imperfect and that two similar investigations will proceed on the magnetic and non-magnetic fractions. It is anticipated that on each such fraction there will be a series of gravity concentration steps each resulting in a concentrate, a reject tailing and a middling for further grinding.

4 You will be advised when we can plan Stage 2 of this project, the tungsten recovery tests.

Yours faithfully,

(H.K. Wellington)
Chief Chemist & Metallurgist

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221579

R. 745: Stage I

Company: Comalco Ltd.

Samples: 761736, Head sample for R.731 and products made in R.731.

Source: Surface dumps at Shepherd and Murphy Mine, Moina.

Aim: To concentrate the tungsten.

Method: Assay the magnetic separation products from grinds A, B & C in R.731 for WO_3.

2 From R.731 grind the three products made in Grind C magnetic separation and submit these for Davis Tube testing to see what ultimate liberation of WO_3 is possible.

3 Grind a 1 kg head sample to 95% -300 µm, magnetically separate and assay products for S.Fe and WO_3.

4 At this point it should be possible to decide how to proceed with the initial magnetic separation ahead of gravity concentration.

(H.K. Wellington)
Chief Chemist & Metallurgist

486
OS. 1144

BENEFICIATION
221580

DEPARTMENT OF MINES—TASMANIA

TELEPHONES:
Metallurgical Research .. .. } 44 2431-2
Laboratory .. .. .. } (2 lines)
Mines Inspection .. .. ..
Explosives & Inflammable Liquids
.. ..

LAUNCESTON OFFICES
287 WELLINGTON STREET
SOUTH LAUNCESTON 7250

10th May, 1977.

Comalco Ltd.,
Exploration Department,
P.O. Box 691,
DEVONPORT, 7310
Tasmania.

Dear Sirs, R.745: Moina Wrigglite

The first stage of this project has now been completed. On Table I the tin and tungsten distributions on the magnetic separation products from Grinds A, B and C are given together with the tungsten and iron distributions from Grind D, a new grind with 95% passing 300µm.

Although the results from D do not fit into the sequence of results obtained from A, B and C the main difference is in the mass of the M/S 4 M/A 2 product.

In Table II are given the results of heavy liquid separations of the size fractions made from the head sample. It will be seen that scheelite appears to be freed from 300µm down hence we propose in the second stage to grind to pass 300µm then do a magnetic separation. The magnetic fraction would be further ground and magnetically separated in stages until all passed 38µm.

The initial non magnetic fraction would be sized hydraulically then the sized fractions tabled. Table middlings would be further ground and tabled down to all passing 38µm.

Some combination of products from the non magnetic products from the initial magnetic fraction treatment and middlings produced from the initial non-magnetic fraction is possible. Some further magnetic separation is possible in the non magnetic treatment. Also at some stage flotation to remove the sulphides will have to be fitted in to the program.

We will begin Stage II now but until we get some results from the first tests details of what will be done are difficult to forecast.

Yours faithfully,

(H.K. Wellington)
Chief Chemist & Metallurgist

Table I Crockett Magnetic Separation

Grind	Fraction	Per Cent				%Distribution		
		Mass	WO_3	Sn	S.Fe	WO_3	Sn	S.Fe
	M/S 1 M/A1	41.9	0.08	.19		23.0	32.9	
A	M/S 1 M/A2	6.4	0.14	.28		6.1	7.4	
	M/S 1 N	51.7	0.20	.28		70.9	59.7	
(95% -600)	H1	100	(0.15)	(0.24)		100	100	
	M/S 2 M/A1	34.1	0.06	.18		15.3	25.1	
B	M/S 2 M/A2	9.0	0.12	.27		8.1	9.9	
	M/S 2 N	56.9	0.18	.28		76.6	65.0	
(95% -150)	H2	100	(0.13)	(0.25)		100	100	
	M/S 3 M/A1	25.8	0.02	.13		4.1	13.8	
C	M/S 3 M/A2	5.8	0.18	.31		6.9	6.1	
	M/S 3 N	69.4	0.16	.28		89.0	80.1	
(95% -38)	H3	100	(0.12)	(0.24)		100	100	
	M/S 4 M/A1	42.7	0.09		37.0	27.5		76.9
D	M/S 4 M/A2	17.8	0.17		9.6	21.6		8.3
	M/S 4 N	39.5	0.18		7.7	50.9		14.8
(95% -300μm)	H4	100	(0.14)		(20.5)	100		100

Table II Heavy Liquid Separation of Size Fractions

Fraction	% Mass	Scheelite occurrence from (uv examination)
+2.36mm H/L F/T	0.2	nil
+2.36mm H/L S/K	trace	nil
+2.36mm	0.2	nil
+1.18mm H/L F/T	0.2	nil
+1.18mm H/L S/K	1.8	composite
+1.18mm	2.0	composite
+600µm H/L F/T	0.3	nil
+600µm H/L S/K	1.3	composite
+600µm	1.6	composite
+300µm H/L F/T	1.0	nil
+300µm H/L S/K	4.4	free
+300µm	5.4	-
+150µm H/L F/T	2.3	-
+150µm H/L S/K	13.4	free
+150µm	15.7	-
+75µm H/L F/T	3.9	-
+75µm H/L S/K	12.5	free
+75µm	16.4	-
+38µm	12.8	free
-38µm	45.9	-
HEAD	100	

489

221583

R. 745 Stage II

Company: Comalco Ltd.

Samples: 761736, Head sample for R. 731 and products made in R. 731.

Source: Surface dumps at Shepherd and Murphy Mine, Moina.

Aim: To concentrate the tungsten.

Method: Grind in closed circuit with 300µm screen then magnetically separate screen undersize.

2. M/A will be reground successively to pass other screens and M/S again. Then M/A will be ground again down eventually to 38µm. The N fractions produced may be combined with products from the initial N fraction treatment for further treatment.

3. The initial N fraction will be sized (Geco) and the fractions tabled. Table M's will be reground successively through screens and retabled eventually to 38µm.

4. It may be necessary to incorporate in the N fraction treatment further M/S.

5. Depending on what testing shows will be the incorporation of sulphide flotation into the above.

Note: As this testing program could be come quite complex draw up a flow sheet as you go to identify products.

(H. K. Wellington),
Chief Chemist & Metallurgist.

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SYMBOLS USED IN METALLURGICAL REPORTS

221584

Product Symbols

A	Anode	L	Liquid	S/D	Solid
C	Concentrate	M	Middling	S/K	Sink
C/E	Calcine	M/A	Magnetic	S/N	Solution
D	Discharge	N	Non-magnetic	T	Tailing
E	Extract	O/F	Overflow	U/F	Underflow
F	Fraction	O/S	Oversize	U/S	Undersize
F/D	Feed	P	Precipitate	#	Mesh (Brit. Std.)
F/T	Float	R	Residue	μ	Microns
G	Gas	R/E	Raffinate	+	Retained on screen
H	Head	R/X	Reflux	-	Passed through screen
K	Cathode	S	Spigot		

Operation Symbols

B	Batch	H/M	Heavy Media Separation
C	Cyclone or Classifier	H/S	Hydrosizer
C/E	Cradle	I/S	Infrasizer
C/L	Column	I/X	Ion Exchange
C/N	Continuous	J	Jig
C/S	Cyclosizer	L	Leach
C/R	Condenser	L/L	Liquid/Liquid Extraction
D	Tilting Deck or Strake	M	Mixer
D/N	Distillation	M/S	Magnetic Separation
D/R	Dryer	P	Panning
E	Electrostatic Separation	R	Roasting
E/F	Elutriation Fraction	S	Screening or Sieving
E/S	Electrolysis	T	Table
F	Flotation	T/H	Thickener
F/C	Filter or Centrifuge	V	Vanner
H/L	Heavy Liquid Separation		

Chemical Symbols

The normal chemical symbols for the elements and formulae for compounds are used, but the following are also used:

C. Sn	Cassiterite Tin
S. Sn	Acid soluble tin (Stannite Tin)
T. Sn (or Sn)	Total tin (C.Sn + S.Sn)
S. Fe	Acid (HCl) Soluble iron
T. Fe (or Fe)	Total Iron

Use of Symbols

1. Where there are a number of products from the one operation, e.g. magnetic products, spigot products, these are distinguished by numbers, e.g. M/A1, M/A2, M/A3, S3, S4 etc.

2. Where there are a number of operations, these are numbered, e.g. Two jigging stages J1 and J2, flotation stages F2, F3.

3. Products from operations are described by compounding operation and product symbols thus:

   F4C, C/S4 means the cyclosizer fraction 4 from the concentrate made in flotation stage 4.

Note

Research projects are given a number prefixed with R, e.g. R.606. The tests done in the project are numbered N1, N2 etc.

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R.P. 4144

BENEFICIATION

221585

DEPARTMENT OF MINES—TASMANIA

TELEPHONES:
Metallurgical Research Laboratory .. .. .. } 44 2431-2 (2 lines)
Mines Inspection .. .. }
Explosives & Inflammable Liquids }

LAUNCESTON OFFICES
287 WELLINGTON STREET
SOUTH LAUNCESTON 7250

5th September, 1977

Comalco Exploration Ltd.,
P.O. Box 691,
Devonport. Tas. 7310

R.745: Moina Wrigglite

Dear Sirs,

This project has now reached a stage where successive grinding and concentration techniques have reduced the bulk of the 95 kilograms used in this investigation to a particle size of −53µm.

An outline of the concentration process used is as follows: The ore was initially ball mill ground to −300µm, producing the fractions +170µm, +76µm, +53µm and −53µm. The +170µm fraction was then subjected to magnetic separation on an Eriez drum type wet magnetic separator. The non magnetic fraction was then gravity concentrated on a Deister table and the concentrate produced was subjected to sulphide flotation. The flotation tailing was then fed to a Rapid high intensity dry magnetic separator.. This total process was repeated for each sized fraction together with its appropriately sized and reground table tailing and Eriez magnetics. Thus, by successive stages, the ore was concentrated at +170µm, +76µm, +53µm and −53µm.

A condensed summary of the overall treatment result is enclosed as Table I. A tungstic oxide concentrate, at 22.7 percent WO_3 and containing 33 percent of the overall WO_3 in the sample, was obtained. −53µm table tailings (T4T) require a further examination as they retain 44 percent of the overall WO_3, in 66 percent of the original mass at 0.11 percent WO_3. The −53µm cleaner magnetics (M/S 10 M/A) represent 30 percent of the original mass and contain 76 percent of the acid soluble iron in the sample at 54.3 percent acid soluble iron. This product will be further examined to determine the reason for its low, acid soluble iron content.

As the amount of scheelite produced in this investigation was far too small in quantity for scheelite flotation tests, and also as no further quantities of the original wrigglite sample exist, it would be very desirable if a further sample of approximately 1 tonne could be supplied.

Yours faithfully,

(H. K. Wellington),
Chief Chemist & Metallurgist.

Encl.

Table I

Condensed Summary of Overall Treatment

Fraction	Percent				Percent Distribution		
	Mass	WO_3	Sn	S.Fe	WO_3	Sn	S.Fe
TC, FT, M/S N	(0.05)	(30.9)	(21.0)	–	9.0	3.8	–
TC, FT, M/S M/A	(0.20)	(20.8)	(3.4)	(18.2)	24.2	2.8	0.3
TC, FT	0.25	(22.7)	(6.8)	(15.3)	33.2	6.6	0.3
TC, FC	0.55	(0.54)	(0.70)	4.7	1.6	1.4	Tr
TC	0.80	(7.5)	(2.6)	(8.0)	34.8	8.0	0.3
Middlings *	3.55	(0.24)	(0.42)	(11.2)	6.1	5.6	1.9
Tailings *	65.63	(0.11)	0.27	7.3	44.2	67.6	22.2
Magnetics M/S 10 M/A *	30.02	0.05	0.17	54.3	14.9	18.8	75.6
Head	100	(0.16)	(0.26)	(21.6)	100	100	100

* all minus 53μm

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E 7/74 MOINA BENEFICIATION

M 1588

DEPARTMENT OF MINES—TASMANIA 221587

TELEPHONES:
Metallurgical Research .. ..
Laboratory .. .. .. } 44 2431-2 (2 lines)
Mines Inspection .. .. ..
Explosives & Inflammable Liquids

LAUNCESTON OFFICES
287 WELLINGTON STREET
SOUTH LAUNCESTON 7250

19th May 1978

Comalco Ltd,
Exploration Dept,
P.O. Box 691,
Devonport,
TASMANIA 7310

R745: Moina Wrigglite.

Dear Sir,

Please find attached the report on the above project a summary of which was given you by letter on 5th Sep'77. The long delay has been due to problems with equipment for fluorine assays.

Further work on Moina Wrigglite (R758) on a ton sample is proceeding. A head assay of this material gave:-

Sn	0.3%
WO_3	0.16%
S	0.8%
Ca	12.7%
T.Fe	21.2%
S.Fe	20.4%

Yours faithfully,

(H. K. Wellington)
Chief Chemist & Metallurgist.

221588

ABBREVIATIONS

Symbols used in metallurgical reports

Product symbols

A	Anode	L	Liquid	S/D	Solid
C	Concentrate	M	Middling	S/K	Sink
C/E	Calcine	M/A	Magnetic	S/N	Solution
D	Discharge	N	Non-magnetic	T	Tailing
E	Extract	O/F	Overflow	U/F	Underflow
F	Fraction	O/S	Oversize	U/S	Undersize
F/D	Feed	P	Precipitate	#	Mesh (B.S.S.)
F/T	Float	R	Residue	µm	Microns
G	Gas	R/E	Raffinate	+	Retained on screen
H	Head	R/X	Reflux	-	Passed through screen
K	Cathode	S	Spigot		

Operation symbols

B	Batch	H/L	Heavy Liquid Separation
B/M	Ball Mill	H/M	Heavy Media Separation
C	Cyclone or Classifier	H/S	Hydrosizer
C/E	Cradle	I/S	Infrasizer
C/H	Crush	I/X	Ion Exchange
C/L	Column	J	Jig
C/N	Continuous	L	Leach
C/S	Cyclosizer	L/L	Liquid/Liquid Extraction
C/R	Condenser	M	Mixer
D	Tilting Deck or Strake	M/S	Magnetic Separation
D/N	Distillation	P	Panning
D/R	Dryer	P/N	Precipitation
E	Electrostatic Separation	R	Roasting
E/F	Elutriation Fraction	R/M	Rod Mill
E/S	Electrolysis	S	Screening or Sieving
F	Flotation	S/B	Stamp Battery
F/C	Filter or Centrifuge	T	Table
G	Grind	T/H	Thickener
		V	Vanner

Chemical symbols

The normal chemical symbols for the elements and formulae for compounds are used, with the addition of the following:

C.Sn	Cassiterite Tin
S.Sn	Acid soluble tin (Stannite Tin)
T.Sn (or Sn)	Total tin (C.Sn + S.Sn)
S.Fe	Acid (HCl) Soluble iron
T.Fe (or Fe)	Total Iron

Use of symbols

(1) Where there are a number of products from the one operation, *e.g.* magnetic products, spigot products, these are distinguished by numbers, *e.g.* M/A1, M/A2, M/A3, S3, S4 etc.

(2) Where there are a number of operations, these are numbered, *e.g.* Two jigging stages J1 and J2, flotation stages F2, F3.

(3) Products from operations are described by compounding operation and product symbols thus: F4C, C/S4 means the cyclosizer fraction 4 from the concentrate made in flotation stage 4.

Note: Research projects are given a number prefixed with R, *e.g.* R.606. The tests done in the project are numbered N1, N2 etc.

495

M 1588

221589

DEPARTMENT OF MINES—TASMANIA

TELEPHONES:
Metallurgical Research .. ..
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Mines Inspection .. .. ..
Explosives & Inflammable Liquids
44 2431-2 (2 lines)

LAUNCESTON OFFICES
287 WELLINGTON STREET
SOUTH LAUNCESTON 7250

9th May 1978

COMALCO LTD - MOINA WRIGGLITE.

R745

Introduction:

This investigation extends the work of report R731 in an examination of a sample of wrigglite ore (761736) selected from surface dumps of the Shepherd and Murphy Mine at Moina. This investigation deals essentially with the concentration and recovery of scheelite only. The situation of minerals such as fluorite, magnetite and cassiterite, at the end of the scheelite concentration process, is however, defined.

Aim:

To concentrate the scheelite present in the wrigglite.

Method:

Successive grinding and concentration techniques were used to reduce a sample bulk of 95 Kg to a final particle size of - 53 µm.

An outline of the concentration process (see flowsheet) used is as follows. The ore was initially ball mill ground to - 300 µm, producing the fractions + 170 µm, + 76 µm, + 53 µm and - 53 µm. The + 170 µm fraction was then subjected to magnetic separation on an Eriez drum type wet magnetic separator. The resulting non-magnetic fraction was then gravity concentrated on a laboratory Deister table and the concentrate produced was subjected to sulphide flotation. The flotation tailing was then dried and fed to a Rapid high intensity dry magnetic separator. This total process was repeated for each sized fraction together with its appropriately sized and reground table tailing and Eriez magnetics. Thus by successive stages, the ore was concentrated at the following particle sizes.

Primary section	-300µm + 170 µm
Secondary section	-170µm + 75 µm
Tertiary section	- 75µm + 53 µm
Quaternary section	- 53µm

Results Obtained

1. **Primary Section** (Grinding through 300µm)

(a) **Sizing and Magnetic Separation**

Fraction µm	Percent					Percent Distribution			
	Mass	WO_3	Sn	F	S.Fe	WO_3	Sn	F	S.Fe
-300+170 M/S1N	8.84	0.21	0.26	7.7	6.8	11.1	9.6	7.2	2.8
-300+170 M/S1M/A	15.18	0.11	0.21	7.6	28.1	9.9	13.2	12.2	20.1
-300 + 170	24.02	(0.15)	(0.23)	(7.6)	(20.3)	21.0	22.8	19.4	22.9
-170 + 76	18.89	0.17	0.24	7.6	20.6	19.1	18.9	15.2	18.3
-76 + 53	11.86	0.16	0.23	8.4	20.9	11.3	11.3	10.5	11.7
-53	42.17	0.18	0.25	11.7	22.0	45.3	43.9	52.1	43.7
Residue in Scoop Box	3.06	0.19	0.24	8.8	23.1	3.3	3.1	2.8	3.4
Head	100	(0.17)	(0.24)	(9.5)	(21.2)	100	100	100	100
Head Analysis	-	0.14	0.24	9.8	21.2				

(b) **Concentration of -300 + 170 M/S 1N**

Fraction µm	Percent					Percent Distribution			
	Mass	WO_3	Sn	F	S.Fe	WO_3	Sn	F	S.Fe
M/S2N	Tr.*	38.2	25.7	-	-	0.5	0.2	Nil	Nil
M/S2M/A	0.02*	48.3	0.91	1.3	14.3	3.3	0.1	Tr.	Tr.
F1T	0.02	(46.8)	(4.7)	(1.1)	(12.1)	3.8	0.3	Tr.	Tr.
F1C	0.14	1.1	0.82	2.8	4.1	0.8	0.4	Tr.	Tr.
T1C	0.16	(5.7)	(1.2)	(2.6)	(4.8)	4.6	0.7	Tr.	Tr.
T1T	8.68	0.15	0.28	8.8	7.3	6.5	8.9	7.2	2.8
Head-300+170M/S1N	8.84	(0.25)	(0.30)	(8.7)	(7.3)	11.1	9.6	7.2	2.8
Head Analysis	-	0.21	0.26	7.7	6.8				

* Calculated masses Tr. = 0.0025% & 0.02 = 0.0138.

2. Secondary Section (Grinding Through 170um)

(a) Feed to Section

Fraction µm	Percent					Percent Distribution			
	Mass	WO_3	Sn	F	S.Fe	WO_3	Sn	F	S.Fe
Residue *	3.06	0.19	0.24	8.8	23.1	3.3	3.1	2.8	3.4
M/S 1 M/A	15.18	0.11	0.21	7.6	28.1	9.9	13.2	12.2	20.1
T1T	8.68	0.15	0.28	8.8	7.3	6.5	8.9	7.2	2.8
Primary + 76	18.89	0.17	0.24	7.6	20.6	19.1	18.9	15.2	18.3
Head	45.81	(.15)	(.24)	(7.9)	(20.7)	38.8	44.1	37.4	44.6

* Residue from scoop box of ball mill.

(b) Sizing and Magnetic Separation

Fraction µm	Percent					Percent Distribution			
	Mass	WO_3	Sn	F	S.Fe	WO_3	Sn	F	S.Fe
M/S 3N	12.10	0.21	0.27	10.5	7.5	13.8	13.1	12.0	4.2
M/S 3M/A	15.14	0.12	0.23	7.4	31.8	9.9	13.9	10.5	22.0
-170+76	27.24	(0.16)	(0.25)	(8.8)	(21.0)	23.7	27.0	22.5	26.2
-76+53	9.14	0.14	0.23	7.9	21.0	6.9	8.4	6.8	8.8
-53	9.43	0.16	0.23	9.2	22.3	8.2	8.7	8.1	9.6
Head	45.81	(0.16)	(0.24)	(8.7)	(21.3)	38.8	44.1	37.4	44.6

(c) Concentration of -170 + 76 M/S 3 N

Fraction µm	Percent					Percent Distribution			
	Mass	WO_3	Sn	F	S.Fe	WO_3	Sn	F	S.Fe
M/S 4N	Tr.*	29.0	21.3	-	-	0.2	0.1	-	-
M/S 4M/A	0.08*	12.8	0.73	2.2	17.3	4.4	0.3	Tr.	0.1
F2T	0.08	(14.1)	(1.1)	(2.2)	17.0	4.6	0.4	Tr.	0.1
F2C	0.14	0.13	0.18	1.1	4.2	0.1	0.1	Tr.	Tr.
T2C	0.22	(4.4)	0.47	(1.4)	(8.3)	4.7	0.5	Tr.	0.1
T2T	11.88	0.16	0.24	11.2	6.9	9.1	12.6	12.0	4.1
M/S 3N	12.10	(0.24)	(0.24)	(11.0)	(6.9)	13.8	13.1	12.0	4.2
Head Analysis	-	(.21)	.27	10.5	7.5				

* Calculated masses. Tr. = 0.0012% 0.08 = 0.0718%

3. Tertiary Section (Grinding Through 76 um)

(a) Feed to Section

Fraction µm	Percent					Percent Distribution			
	Mass	WO_3	Sn	F	S.Fe	WO_3	Sn	F	S.Fe.
Primary-76+53	11.86	0.16	0.23	8.4	20.9	11.3	11.3	10.5	11.7
Secondary-76+53	9.14	0.14	0.23	7.9	21.0	6.9	8.4	6.8	8.8
M/S 3 M/A	15.14	0.12	0.23	7.4	31.8	9.9	13.9	10.5	22.0
T2T	11.88	0.16	0.24	11.2	6.9	9.1	12.6	12.0	4.1
Head	48.02	(0.14)	(0.23)	(8.7)	(20.9)	37.2	46.2	39.8	46.6

(b) Sizing and Magnetic Separation.

Fraction µm	Percent					Percent Distribution			
	Mass	WO_3	Sn	F	S.Fe	WO_3	Sn	F	S.Fe
M/S 5N	19.95	0.12	0.29	9.3	7.4	17.6	21.1	19.5	6.8
M/S 5M/A	17.93	0.08	0.23	6.1	36.3	10.6	15.1	11.5	30.2
-76+53	37.88	(0.10)	(0.26)	(7.8)	(21.1)	28.2	36.2	31.0	37.0
-53	10.14	0.12	0.27	8.2	20.3	9.0	10.0	8.8	9.6
Head	48.02	(0.11)	(0.26)	(7.9)	(20.9)	37.2	46.2	39.8	46.6

(c) Concentration of -76 + 53 M/S 5N

Fraction µm	Percent					Percent Distribution			
	Mass	WO_3	Sn	F	S.Fe	WO_3	Sn	F	S.Fe
M/S 6N	Tr.*	32.2	26.6	-	-	1.4	.7	-	-
M/S 6M/A	0.04*	23.8	3.4	1.1	15.4	5.8	.4	Tr.	0.1
F3T	0.04	(25.1)	(7.0)	(1.1)	(13.1)	7.2	1.1	Tr.	0.1
F3C	0.11	0.13	0.18	0.8	4.1	0.1	0.1	Tr.	Tr.
T3C	0.15	(7.5)	(2.2)	(0.9)	(6.8)	7.3	1.2	Tr.	0.1
T3T	19.80	0.08	0.27	9.3	6.7	10.3	19.9	19.5	6.7
M/s 5N	19.95	(0.14)	(0.28)	(9.2)	6.7	17.6	21.1	19.5	6.8
Head Analysis		0.12	0.29	9.3	7.4				

" * Calculated masses Tr. = 0.0068% & 0.04 = 0.0373% "

4. Quaternary Section (Grinding Through 53 um)

(a) Feed to Section

Fraction µm	Percent					Percent Distribution			
	Mass	WO_3	Sn	F	S.Fe	WO_3	Sn	F	S.Fe
Primary -53	42,17	0.18	0.25	11.7	22.0	45.3	43.9	52.1	43.7
Secondary -53	9.43	0.16	0.23	9.2	22.3	8.2	8.7	8.1	9.6
Tertiary -53	10.14	0.12	0.27	8.2	20.3	9.0	10.0	8.8	9.6
M/S 5 M/A	17.93	0.08	0.23	6.1	36.3	10.6	15.1	11.5	30.2
T3T	19.80	0.08	0.27	9.3	6.7	10.3	19.9	19.5	6.7
Head	99.47	(0.13)	0.25	(9.6)	(21.4)	83.4	97.6	100.0	99.8

(b) Magnetic Separation

Fraction µm	Percent					Percent Distribution			
	Mass	WO_3	Sn	F.	S.Fe	WO_3	Sn	F	S.Fe
M/S 7N	67.97	0.14	0.31	12.0	7.4	65.9	76.9	88.7	23.3
M/S 7 M/A	31.50	0.08	0.18	3.3	52.4	17.5	20.7	11.3	76.5
-53 Head	99.47	(0.12)	(0.27)	(9.2)	(21.7)	83.4	97.6	100.0	99.8

(c) Concentration of -53 M/S 7N

Fraction µm	Percent					Percent Distribution			
	Mass*	WO_3	Sn	F	S.Fe	WO_3	Sn	F	S.Fe
M/S 8N	0.04	30.2	19.7	-	-	6.9	2.8	-	-
M/S 8 M/A1	0.05	19.7	4.0	-	35.7	5.8	.8	-	.1
M/S 8 M/A2	0.03	24.8	9.7	-	-	4.9	1.2	-	-
M/S 9 N	0.02	2.2	1.4	-	-	.3	.1	-	-
M/S 9 M/A1	0.05	1.3	0.60	-	32.7	.4	.1	-	.1
M/S 9 M/A2	0.06	0.78	0.48	-	-	.3	.1	-	-
F4T	0.25	(12.1)	(5.3)	-	(13.0)	18.6	5.1	-	.2
F4C	0.16	0.68	1.4	0.9	6.0	.6	.8	Tr.	Tr.
T4C	0.41	(7.7)	(3.7)	(0.4)	(10.3)	19.2	5.9	Tr.	.2
T4M	1.93	0.21	0.46	9.7	9.8	2.5	3.4	2.2	.9
T4T	65.63	0.11	0.27	11.1	7.3	44.2	67.6	86.5	22.2
M/S 7N	67.97	(0.16)	(0.30)	(11.0)	(7.4)	65.9	76.9	88.7	23.3
Head Analysis		0.14	0.31	12.0	7.4				

*Calculated figures in order are:- 0.04 (0.0373), 0.05 (0.0482), 0.03 (0.0321), 0.02 (0.0242) 0.05 (0.0466) & 0.06 (0.0620)

(d) Cleaning of Magnetics -53 M/S 7 M/A

Fraction µm	Percent					Percent Distribution			
	Mass	WO_3	Sn	F	S.Fe	WO_3	Sn	F	S.Fe
M/S 10 M/a	30.02	0.05	0.17	3.0	54.3	14.9	18.8	9.7	75.6
M/S 10 N	1.48	0.18	0.35	10.3	13.0	2.6	1.9	1.6	0.9
M/S 7 M/A	31.50	(0.06)	(0.18)	(3.3)	52.4	17.5	20.7	11.3	76.5
Head Analysis		0.08	0.18	3.3	52.4				

A Davis tube test of M/S 10 M/A gave the following result:

82.5 percent magnetics.
At 62.9 percent S.Fe.
and 19.8 percent Fe^{++}
The ratio of ferrous iron (Fe^{++}) to acid soluble iron (S.Fe) was 0.31

(e) WO_3 & Sn Distribution in T4T (-53 µm T/T)

Fraction	Mass	Percent			Percent Distribution		Cumulative Percent Dist.	
	g	Mass	WO_3	Sn.	WO_3	Sn	WO_3	Sn.
+53 µm	=1.1	0.36						
+45 µm	1.9	0.62	0.04	0.24	1.5	4.6	1.5	4.6
+38 µm	12.3	4.04						
C/S 1*	0.3	0.10						
C/S 2	5.6	1.84	0.04	0.25	3.2	10.2	4.7	14.8
C/S 3	26.5	8.69						
C/S 4	31.5	10.34	0.09	0.25	7.1	9.9	11.8	24.7
C/S 5	21.1	6.92	0.13	0.29	6.8	7.7	18.6	32.4
O/F	99.7	32.72	0.10	0.28	25.6	35.2	44.2	67.6
Head (Calc)			(0.09)	(0.27)				
Head (Assay)	200	65.63	0.11	0.27	44.2	67.6	-	-

Cyclesizing (C/S) at 9° C.

*

5. Summary of overall Treatment.

Fraction µm	Percent					Percent Distribution			
	Mass	WO_3	Sn	F	S.Fe	WO_3	Sn	F	S.Fe
M/S 2 M	0.0025	38.2	25.7	-	-	0.5	0.2	-	-
M/S 4 M	0.0012	29.0	21.3	-	-	0.2	0.1	-	-
M/S 6 M	0.0068	32.2	26.6	-	-	1.4	0.7	-	-
M/S 8 M	0.0373	30.2	19.7	-	-	6.9	2.8	-	-
Total M/S M	0.0478	(30.9)	(21.0)	-	-	9.0	3.8	-	-
M/S 2 M/A	0.0138	48.3	0.91	1.3	14.3	3.3	0.1	Tr.	Tr.
M/S 4 M/A	0.0718	12.8	0.73	2.2	17.3	4.4	0.3	Tr.	0.1
M/S 6 M/A	0.0373	23.8	3.4	1.1	15.4	5.8	0.4	Tr.	0.1
M/S 8 M/A1	0.0482	19.7	4.0	-	35.7	5.8	0.8	-	0.1
M/S 8 M/A2	0.0321	24.8	9.7	-	-	4.9	1.2	-	-
Total M/S M/A	0.2032	(20.8)	(3.4)	(1.1)	(18.8)	24.2	2.8	Tr.	.3
Total FT.	0.251	(22.7)	(6.8)	(0.9)	(15.3)	33.2	6.6	Tr.	.3
F1C	0.14	1.1	0.82	2.8	4.1	.8	.4	Tr.	Tr.
F2C	0.14	0.13	0.18	1.1	4.2	.1	.1	Tr.	Tr.
F3C	0.11	0.13	0.18	0.8	4.1	.1	.1	Tr.	Tr.
F4C	0.16	0.68	1.4	0.9	6.0	.6	.8	Tr.	Tr.
Total FC	0.55	(.54)	(.70)	(1.4)	(4.7)	1.6	1.4	Tr.	Tr.
Total TC	0.80	(7.5)	(2.6)	(1.3)	(8.0)	34.8	8.0	Tr.	.3
M/S 9 M	.0244	2.2	1.4	-	-	.3	.1	-	-
M/S 9 M/A1	.0466	1.3	0.60	-	32.7	.4	.1	-	.1
M/S 9 M/A2	.062	0.78	0.48	-	-	.3	.1	-	-
T4 M	1.93	0.21	0.46	9.7	9.8	2.5	3.4	2.2	0.9
M/S 10 M	1.48	0.18	0.35	10.3	13.0	2.6	1.9	1.6	0.9
Total M	3.55	(.24)	(.42)	(9.6)	(11.2)	6.1	5.6	3.8	1.9
T4T	65.63	0.11	0.27	11.1	7.3	44.2	67.6	86.5	22.2
M/S 10 M/A	30.02	0.05	0.17	3.0	54.3	14.9	18.8	9.7	75.6
Overall Head	100	(0.16)	(0.26)	(8.5)	(2.6)	100	100	100	100

6 Flotation Concentrates

Chemical analysis of a sulphide composite made on a weighted basis from cleaner flotation concentrates from the four stages of concentration, was found to contain the following:

Element	Percent
As	0.13
Sb	0.01
Bi	1.30
Pb	0.52
Cu	0.33
Zn	2.3
S	40.8
Mo	0.22
Fe	38.7

Discussion:

1. Scheelite - (a) An overall recovery of 33.2 percent of the tungstic oxide at a grade of 22.7 percent WO_3 was obtained in 0.25 percent of the original mass as table concentrate, flotation tailing. High intensity dry magnetic separation of this product did enable the production of a non magnetic concentrate containing 30.9 percent WO_3 at an overall recovery of only 9.0 percent. The magnetics from these separations contained 20.8 percent WO_3 with an overall distribution of 24.2 percent. The presence of wolframite producing the above result is not totally discounted, but, the result is thought to be due totally to the presence of scheelite magnetite composites.

(b) A WO_3 distribution of the -53µm final tailing (T4T) indicated that 39.5 percent of the overall WO_3 present was finer than 16µm in particle size and as such, unrecoverable by gravity concentration methods. Further, 25.6 percent of the overall WO_3 was finer than 9 µm in particle size.

(c) The final -53µm cleaner magnetics (M/S 10 M/A) contained 14.9 percent of the overall tungstic oxide at a grade of 0.05 percent WO_3.

2. Cassiterite - (a) Table concentrate flotation tailings were found to contain 6.6 percent of the overall tin at a grade of 6.8 percent. High intensity dry magnetic separation raised this grade to 21 percent with a subsequent drop in recovery to 3.8 percent overall.

(b) The very fine cassiterite particle size is shown in the tin distribution of the final tailing product (T4T) where 52.8 percent of the overall tin in the sample is smaller than 16µm in particle size, with some of this possibly even being of a composite nature.

(c) 18.8 percent of the overall tin was found to be locked in the -53µm cleaner magnetics (M/S 10 M/A) at a concentration of 0.17 percent.

3. Magnetite: A magnetite concentrate was produced that contained 82.5 percent magnetics (Davis Tube determination) in 30 percent of the overall mass and also contained 54.3 percent HCl soluble iron. The Davis tube magnetics had an HCl soluble iron content of 62.9 percent, a ferrous iron content of 19.8 percent, and hence an Fe^{++} to Fe ratio of 0.31.

4. Fluorite: (a) 86.5 percent of the original fluorine was contained in the -53µm final gravity tailings (T4T). The fluorine grade was increased slightly from the original value of 9.8 percent to 11.1 percent. 9.7 percent of the original fluorine was contained in the -53µm cleaner magnetics (M/S 10 M/A) at a grade of 3.0 percent fluorine.

Conclusions 1. 33.2 percent of the overall tungstic oxide present in the sample was recovered at a grade of 22.7 percent.

2. The final gravity tailings (T4T) and cleaner magnetics (M/S 10 M/A) together contain 59.1 percent of the unrecovered WO_3 of which 54.4 percent is considered unrecoverable.

3. More work on a larger sample is required to have sufficient mass to investigate further the upgrading of the tungsten and tin products.

Notes on Flowsheet

Magnetic Separation Magnetic Separations M/S 1, M/S 3, M/S 5, M/S 7 and M/S 10 were all done on an Eriez wet down laboratory separator. Magnetic Separations M/S 2, M/S 4, M/S 6, M/S 8 and M/S 9 were all done on a Rapid dry belt laboratory separator.

Flotation Each flotation operation shown e.g. F 1 consisted of a rougher stage and cleaning of the rougher concentrate to produce the final product e.g. F 10. The tailings were combined as F 11 for magnetic separation M/S 2.

Ball Milling The scoop box cleanings from ball milling (B/M 1) were fed to the next ball milling stage, B/M 2.

Metallurgist. K. Anderson

(H. K. Wellington)
Chief Chemist & Metallurgist

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221600

PRIMARY SECTION

FEEDER

SCREEN 300µm
U/S
O/S

SCREEN 170µm
O/S
U/S

BALL MILL 1

SCREEN 76µm
O/S
U/S

SCREEN 53µm
U/S
O/S

MAG SEP. 1
N
M/A

TABLE 1
C
T

FLOATATION 1
C
T

MAG. SEP. 2
N
M/A

SECONDARY SECTION

BALL MILL 2

SCREEN 170µm
U/S
O/S

SCREEN 76µm
O/S
U/S

SCREEN 53µm
U/S
O/S

MAG. SEP. 3
N
M/A

TABLE 2
C
T

FLOATATION 2
C
T

MAG. SEP. 4
N
M/A

TERTIARY SECTION

BALL MILL 3

SCREEN 76µm
U/S
O/S

SCREEN 53µm
O/S
U/S

MAG. SEP. 5
N
M/A

TABLE 3
C
T

FLOATATION 3
C
T

MAG. SEP. 6
N
M/A

QUATERNARY SECTION

BALL MILL 4

SCREEN 53µm
U/S
O/S

MAG. SEP. 7
M/A
N

MAG. SEP. 10
N
M/A

TABLE 4
M
C
T

FLOATATION 4
T
C

TABLE 5
M
C

MAG SEP. 9
N
N/A1
M/A2

MAG. SEP. 8
N
M/A1
M/A2

FLOWSHEET R745

COMALCO EXPLORATION, MOINA WRIGGLITE ORE

APPENDIX 32.

SPECIFICATIONS FOR MAGNETITE FOR HEAVY MEDIA USE

508

BENEFICIATION

221602

Telephone:
51-2444
(5 Lines)
NEWCASTLE

All Communications to be Addressed to
RESEARCH MANAGER

The Broken Hill Proprietary Coy. Ltd.

(INCORPORATED IN THE STATE OF VICTORIA)

RESEARCH DIVISION

CENTRAL RESEARCH LABORATORIES
SHORTLAND, N.S.W. 2307

YOUR REF. JMS - 21/11/75
OUR REF. PHS/CWB/BM.404-1.

13th September, 1976.

Mr. A. H. Bartlett,
Manager - Exploration,
Comalco Limited,
GPO Box 2773Y,
MELBOURNE, Vic. 3001

Dear Sir,

Further to your letter and the subsequent 'phone discussion with Dr. Scaife, details of our use of fluorspar for steelmaking are as follows:-

Current source:	China
Consumption:	15-20,000 t.p.a.
Cost:	\$63-30 into dump for 85% CaF_2 (penalty/bonus of 81 cents per unit of CaF_2 for variation)
Specification:	CaF_2 83% (min) SiO_2 16% (max) size 6-100 mm (85% min within range)

To be usable by the steel industry, the fluorspar would have to be at least 60% CaF_2.

In regard to the possibility of producing a coal washery grade magnetite from this deposit, please find attached details of the required properties and specifications for such material currently purchased by our company. The price of this magnetite is \$86 per tonne delivered into store.

Should you wish to proceed with this development, the appropriate beneficiation and product evaluation could be carried out at these laboratories on a contractual basis.

Yours faithfully,

K. J. Figgis
MANAGER.

Encl.

excessive settling may result.

Experience has shown that -325 mesh magnetites with coercive forces in the range 50-120 oersteds are suitable.

Maghemite (magnetic Fe_2O_3) has a high coercive force, ie. is difficult to demagnetise. Much of the Fine Flower (N.S.W.) magnetite is high in maghemite.

4. Grindability.

This is a measure of the mechanical hardness and toughness of the magnetite. The magnetite should be hard ~~Generally desirable~~ to minimise sliming during preparation, and degradation in the washing and recovery circuits. Ultrafine magnetite particles are frequently lost in the recovery circuit.

However, excessive hardness of the magnetite will very likely cause excessive wear of the grinding apparatus and for this reason hardness due to impurities such as quartz should not be too high.

5. General.

The above four properties (Specific Gravity, Susceptibility, Coercive Force, and Grindability) are inherent in the particular magnetite of a particular deposit and are readily measured or compared with other magnetites of known performance. Most other properties such as chemical composition, porosity, degree of oxidation, etc. only influence the performance of the magnetite through their effects on one or more of the above properties.

Features imparted to the magnetite by the treatment process are freedom from non-magnetics, and particle size distribution.

Specifications which apply to the product, ie. ground magnetite, are:

Chemical Specification:

Metallic Iron	SiO_2
5% max.	2% max.

Probably the main reason for the 2% SiO_2 limit is that siliceous magnetite is hard and could cause undue wear on grinding apparatus.

Physical Specifications:

S.G. (of the magnetite)	4.6 min.
Moisture content	10% Max.
Magnetics content	90% Min.
Susceptibility (at 800 oersteds)	0.050 e.m.u./gm. Min.
Coercive force	120 oersteds Max.
Size analysis prior to ball milling:	+1/16" 5% Max.
	+30 mesh 10% Max.

Size analysis after ball milling:

+200 mesh		2% max.
−200 mesh	+300 mesh	10% max.
−300 mesh		90% min.

Milling time in Linatex HC 20 ball mill not more than 2.5 hours.

Settling rate at 1.5 (bath) S.G. demagnetised: 0.5 inches per minute maximum

The above details are part of Simonacco Ltd. Specification WTR 61/1/63.

Typical values for Bigginden (Qld.) product are:

Total Fe	Fe_3O_4	Fe_2O_3	TiO_2	SiO_2
66.4	84.8	8.0	0.28	3.4
Al_2O_3	CaO	MgO	MnO	
1.0	1.9	0.3	0.4	

S.G.		4.8
Magnetics content	(Davis Tube)	95.1%
Susceptibility	(at 800 oersteds)	0.055 e.m.u./gm.
Coercive force		62 oersteds
Moisture		8.0%

Sizing Analysis:		
+200 mesh		1%
−200+270	"	2
−270+325	"	2
−325	"	95
+26	microns	24%
+20	"	12
+14	"	14
+10	"	11
+7	"	4
−7	"	24

As with other magnetites produced for use in coal washeries, the Biggenden material is upgraded by wet magnetic separation after grinding. The primary ball mill product is screened at 100 mesh and screen undersize is fed to two magnetic separators to remove non-magnetics. The magnetic fraction is then fed to a secondary ball mill in closed circuit with a cyclone classifier. Undersize material from the classifier is filtered and sent to product storage.

NEWCASTLE: 6/10/71.
WDP:LB

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REQUIRED PROPERTIES & TESTING OF MAGNETITE USED AS HEAVY MEDIUM

1. Specific Gravity

A high S.G. is necessary to ensure a satisfactory S.G. of the medium. Ideal magnetite has a S.G. of 5.17; as the value falls below this figure the proportion of magnetite in the suspension has to be increased, resulting in higher visiosity and less efficient separation. Most specifications call for a minimum S.G. of 4.6.

Low S.G. may be due to porosity, partial oxidation or to the presence of impurities such as ilmenite.

2. Susceptibility

This is a measure of the force with which a particle of magnetite will be attracted to a magnet. It depends on the type and amount of magnetic mineral present, and on grain size and shape. It is reduced by non-magnetic inclusions and by chemical departure from Fe_3O_4, eg. to Fe_2O_3. It diminishes with particle size.

Susceptibility should be high to permit satisfactory recovery and re-use. If susceptibility is low, magnetic losses will be high in the recovery section.

A value in excess of 0.050 e.m.u./gm. in a magnetic field of 800 oersteds is considered satisfactory.

Magnetics Content. A Davis tube may be used. A fixed weight of sample is introduced into the tube as a slurry and a fixed flow of water at constant velocity is passed through the tube which is agitated slowly. A variable electrical control allows the selection of any desired field strength from 500 to 3800 gauss between the poles. (Field strengths of less than 500 gauss are difficult to keep constant.) The test is run for a constant time.

A magnetics content greater than 90% obtained with a field of 700 gauss is regarded as acceptable.

3. Coercive Force.

This indicates the ease with which the magnetite particles can be demagnetised. It is intrinsic to a particular magnetite but also varies with particle size.

Coercive force should be low to permit easy removal of remanent magnetism after recovery from the washing circuit - otherwise unstable bath conditions such as magnetic flocculation and

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AUSTRALIAN COAL INDUSTRY RESEARCH LABORATORIES LTD.

APPLICATIONS OF MAGNETITE IN THE WASHING OF COAL

by

R. G. BURDON *

INTRODUCTION

Australia is seventh in the list of World coal producers, our production (for all coal) representing two per cent. The quantity of coal mined is increasing, especially in relation to our export trade and in this era of increased technological proficiency the degree of coal preparation demanded has increased.

In the concentration of metalliferous minerals Australia has been a leader in scientific investigation and process development. This has not been the case for coal preparation and our contributions have been less dramatic in this field. It is anticipated that developments will occur in this special field of mineral processing in the future.

Coal as mined has two forms of impurity, the first is liberated ash forming constituents, the second inherent mineral matter. The difference in density between "clean coal" and "liberated reject" is usually of the order of 1.0 and standard gravity concentration processes have been adapted for the separation of these constituents. One process, that using dense media, has been most successful since it gives a wide cut point. This is one process which has been developed mainly by coal preparation engineers. Today approximately thirty per cent of the coal cleaned in this country is treated by the dense medium process. The development of the dense medium cyclone will increase the percentage of coal cleaned by this principle of sink and float.

Magnetite is used in sixty five per cent of the dense media units so the importance of this material as a medium is apparent.

This paper is concerned with dense medium separation and the role of magnetite in the process.

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with a density range of 1.2 to 1.8; and the Staatsmignen in 1938 minus 200 micron loess to obtain densities up to 1.6.

Use of magnetite was introduced in the Conklin process (1922) the Tromp process (1938). The magnetite was minus 150 microns and densities of 1.3 to 1.9 were obtained.

Today there is a wide variety of units for dense medium separation. One item of recent interest has been the revival of the air-sand process first introduced in 1923. Aerated solids (sand minus 20 mesh plus 80 mesh) are used for densities of 1.2 to 1.4. When magnetite is used densities of 1.8 to 2.1 are obtained. For intermediate densities a mixture of the two materials (equal settling particles) are required.

ADVANTAGES OF SINK AND FLOAT SEPARATION

The process is easy to control, has a high capacity, low operating cost, causes little degradation of the products and handles a large feed particle size with a wide size range. The lower particle size limit is governed by hindered settling rates.

Medium

The use of sodium chloride solution is limited to an S.G. of 1.14; calcium chloride to 1.4 at forty per cent concentration and 20°C; zinc chloride to 1.96 S.G. at seventy per cent concentration and 20°C. Use of such solutions requires thorough washing of the product (though some chloride may have an advantage as an anti-freeze). They also require thermal concentration and they may be corrosive.

Organic liquids such as perchlorethylene (S.G. 1.60), bromoform (S.G. 2.90), and tetrabromethane (S.G. 2.96) may be used but they are costly.

Solid materials which have been used as a pseudo-liquid with water include:-

Sand	S.G. 2.65	for densities of 1.35 to 1.80,
Galena	S.G. 7.5	for densities of 3.0,
Magnetite	S.G. 4.5 to 5.2	for densities of 1.30 to 2.0,
Ferrosilicon		for densities of 3.0,
Shale		for densities of 1.25 to 1.65,
Barites and clay		for densities of 1.25 to 1.60,
Copper slag	S.G. 3.4,	
Pyrite cinder	S.G. 4.2	
Atomised lead	S.G.11.3	
Litharge and rubber grains.		S.G. 3.7 for densities of 3.0

The last material is manufactured as cubes 0.65 mm. It is claimed to be easy to recover from slimes, to give suspensions up to 2.0 S.G. because low pump wear. It is used in the MIIKE process in Japan.

Medium solids must be non-reactive, non-corrosive and non-toxic and should not slime. During use, the material is subject to abrasion in pumps, to contamination with waste material and to dilution with wash water, therefore a low cost recovery and regenerative process based upon the characteristics of the material, must be possible. The density of the material is important. The separating density (R) of the bath may be expressed as:

$$R = \frac{\text{total weight}}{\text{total volume}}$$

$$\text{or } R = \frac{W}{p + W(1-p)}$$

where W = S.G. of medium solid

P = fraction of solid medium in the pseudo liquid

Alternatively:

$$p = \frac{W\left(1 - \frac{1}{R}\right)}{W - 1}$$

Obviously the higher the density of the solid the lower the percentage of solids and the lower the viscosity of the bath.

The apparent viscosity of the pseudo-liquid may be determined by means of a viscosimeter such as the Stormer. When automatic control is used the viscosity, bath turbulence and flocculation of medium should be determined so that density can be controlled within the limits ± 0.01. Density control may be by bubble tube, differential pressure cells (D.P.) or nucleonic devices.

PROPERTIES OF MAGNETITE MEDIA

The density of pure magnetite is 5.18 and for use in dense medium separators the value should not be less than 4.7. Pure magnetite is strongly magnetic and this physical property is a major reason for its use since it can be recovered and cleaned by magnetic separation. The saturation moment for magnetite is 92 e.m.u. per gram and this property may be used as a measure of purity. Values below 92 indicate the grade, e.g. a value of 65 would indicate $\frac{65}{92} \times \frac{100}{1}$ or 70.5 per cent. magnetite in the sample. The coercive force measurement for magnetite should be low, for example less than 50 oersteds. Higher values indicate that, due to particle orientation, the magnetite may not be recovered by magnetic

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separators. The magnetic content should be greater than 95 per cent as determined by the Davis Magnetic Tube Tester.

Before use the magnetite must be ground to the required size, after which it should not frit, slime or flocculate since deterioration in these regards would influence viscosity and magnetite recovery. Settlin tests may be used to compare new material and material from the washery circuit. Such tests will indicate changes in viscosity due to entrained material, or loss of the fine magnetite fraction. Magnetite should be checked for soluble or chemically reactive material such as marcasite. The magnetite may be ground wet or dry, the latter avoiding the problems of dewatering and drying. While screen analyses may be used as a guide they do not ensure that the material contains the correct proportions of fine material necessary for a stable medium. The following is a guide:-

Separation Density	Percentage minus 44 Microns
1.35 - 1.50	92 - 95
1.50 - 1.80	60 - 65
2.25 - 2.50	45

Typical particle size distribution for two Australian magnetites are compared with an overseas product in the following Table for separations in the range 1.35 to 1.70.

Particle Size (Microns)	Australian Magnetite		Foreign Magnetite
	Sample 1	Sample 2	
208	0.1	0.1	-
147	0.1	0.1	2.0
104	0.4	0.3	2.0
74	1.2	1.3	9.1
44	7.3	16.7	29.4
31	14.7	12.0	10.8
22	12.3	25.8	10.5
16	14.7	12.9	8.3
11	12.0	8.5	8.1
8	7.6	5.3	5.3
- 8	29.6	17.0	16.5
	100.0	100.0	100.0

METHODS OF MAGNETITE RECOVERY

The objects of the magnetite recovery circuit are to:-

1. recover the magnetite in the wash water,
2. separate the slurry from the magnetite,
3. concentrate the magnetite for re-use in the circuit.

Of the methods which have been used the most common is magnetic separation.

Today belt separators are being replaced by drum separators using permanent magnets of Alcomax III. These give a reliable strong field unaffected by stray fields from adjacent cables or by elevated temperatures. Ceramic type magnets (barium ferrite) show little tangible advantage. Field strengths required are of the order of 300-650 gauss, measured two inches from the drum surface.

Capacity of drum separators is given as 2 - 4 tons per foot width per hour for a thirty inch diameter drum with a recovery of 99.8%. Recovery efficiency is reduced by reduction of magnetite content per unit volume of slurry treated so that it is advantageous to thicken first. The grade of the concentrated magnetite is influenced by the quality of the feed. For example an eighty per cent. grade may be made from a feed containing sixty per cent. magnetics or a ninety per cent. grade from a feed of seventy per cent. magnetics. The density of the discharged product is of the order of 2.2 to 2.4 gms. per c.c.

Research on the use of froth flotation has shown this process to be useful as an aid in magnetite recovery. (It is important to note that sharpness of separation is decreased by the loss of fine magnetite, which affects both stability and viscosity)

Flotation may be used at one of two points in the circuit. If much fine coal is influencing magnetic separation efficiency, especially in cyclones, it should be removed prior to magnetic treatment.

Alternatively the effluent from magnetic separation after the addition of M.I.B.C. may be floated to recover clean coal. The residue after treatment with oleic acid is then floated to recover the magnetite which is re-circulated for further use. In a typical case, recoveries and grades were as follows: (U.S.B.M. - R.I. 6821)

	Wt. % Recovery	Magnetite % Recovery	Ash %
Coal concentrate	63.4	8.7	6.5
Magnetite concentrate	11.5	76.8	-
Reject	25.1	14.5	25.1
	100.0	100.0	

Higher recoveries have been obtained.

LOSSES OF MAGNETITE MEDIUM

There are wide variations in the amount of magnetite consumed. A general average appears to be 0.5 to 1.0 pound per ton of coal for heavy medium bath processes and 1.0 to 1.5 poundper ton for cyclones.

Losses of magnetite are normally governed by:

1. the particle size range of the coal being cleaned (effect of surface area),
2. the pollution of the medium circuit, especially by clay,
3. area and action of the washing screens and the disposition of the wash water (e.g. drops from shower boxes better than fine spray),
4. use of wetting agents to improve washing,
5. magnetic properties of the magnetite, e.g. magnetic susceptibility and coercive force (Use of demagnetising coils may be an advantage),
6. composition of the magnetite and the particle size range,
7. the effectiveness of the regeneration circuit.

PROCESSES

The dense medium separation process is characterised by its ability to handle large feed size, variable feed rates, variable feed grades and large feed particle size ranges. The separatory vessel may be of any shape, a more important factor being the method of product removal.

Floats may be displaced by new feed, transported by overflowing medium, or pushed forward by paddles.

Sinks may be discharged via double acting valves; hydraulically, by upward flowing currents; by air lifts; by wheel, drum or bucket elevators; or by belt or chain scrapers.

Processes are usually categorised as deep or shallow baths.

An important consideration is the amount of pseudo-liquid required. This is governed by the shape of the bath and whether the float products are removed by the overflow medium or mechanically. For a feed rate of 100 to 150 tons per hour the quantity of medium in circulation varies from 350 to 1,400 cubic feet depending upon bath geometry, while the volume flow rate of liquid varies between 2,500 and 17,500 cubic feet per hour.

Units in use include:-

Acco Drewboy	M.D.P.A.
Wilmot	Leebar
Humboldt	Teska
McNally - Tromp	Link Belt
McNally - Low Flo	Disa
Wemco, drum	Turpinson
Wemco, cone	de Vooys
Neldco, drum	Barvoys

Stripa

The Stripa is a special type of dense medium separation which makes use of a shallow bath or launder. Sand-sized medium permits a higher S.G. of separation up to 3.4 with magnetite. Desliming of feed is not essential and a higher recovery of coarse medium solids is possible.

CYCLONES

The dense medium cyclone was introduced as a means of extending the principle of sink and float to the separation of fine particles. The limiting feed size for the usual gravity bath is of the order of $\frac{1}{4}$" this being governed by the rate of sedimentation and the rate of flow of the particle through the separator (residence time).

The cyclone making use of centrifugal force (up to 300 times gravitational acceleration in early units but now of the order of forty times) has permitted the dense medium separation process to be used on particles down to 0.5 mm size with top size of some $1\frac{1}{2}$".

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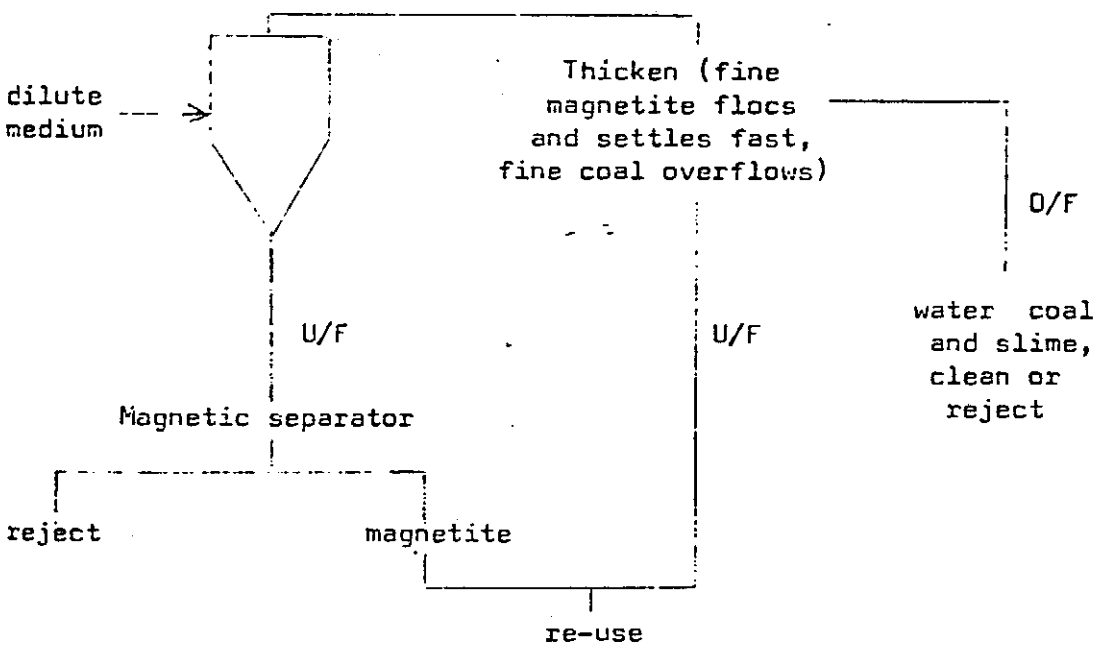
The units are operated at a pressure of some fifteen to eighteen pounds per square inch in a near horizontal position which permits the air core to be maintained at this low pressure, with consequent savings in power and maintenance.

Major problems include medium recovery. Difficulties in this area have been eased by the use of sieve bends and by reduction of rinsing water from over 10,000 cubic feet per 100 tons of coal to some 3,500 cubic feet by use of shower boxes. The use of drum magnetic separators has removed the need for two stages and a densifier.

Density control is rapid because of the low quantity of dense medium required (some 450 cubic feet per 100 tons of coal with circulation of the order of 9,000 cubic feet per hour per 100 tons).

To improve recovery of fine magnetite the medium recovery circuit may be changed to:



There is not much degredation of coal in a cyclone since coal does not come in contact with the wall of the cyclone. Some shale may be broken. Unit sizes are from 24 inches (70 tons per hour) to 14 inches (11 tons per hour) with core angles from 14 to 30 degrees.

Ni-Hard cast iron is an important material of construction.

The costs in operating cyclones have been stated as:

Labour and maintenance ..	30%
Power	20%
Spare parts and magnetite ..	30%
Capital charges	20%
	100%

Variations of the cyclone include:-

1. Dyna-whirlpool process using magnetite to give a density of separation of 1.30 to 2.00,
2. Senidex (or Polish Halder cyclones) in which fines in the feed act as the medium,
3. the Prins which also creates its own medium,
4. Blofif process which uses controlled currents.

PLANT SIZE

Most plants have designed capacities of between 100 and 300 t.p.h. Where larger plants are required they are normally unitised. The 200 t.p.h. unit is fairly usual in coal cleaning practice.

However in considering costs, large plants usually cost less because of savings in the duplication of materials handling equipment. For example two small plants each having 400 t.p.h. capacity would cost more than one plant with a capacity of 800 t.p.h.

Particle size of the feed also influences cost. For example increasing the $-\frac{1}{4}$" content of the feed from 50% to 75% increases the total basic cost of the plant by 12%. Reducing the $-\frac{1}{2}$" content to 25% from 50% decreases the cost by 7%.

Capital costs for plants in U.S.A. are given as $4,000 to $5,000 per ton per hour while operating costs including amortisation range from 22 cents per ton for an 800 t.p.h. plant to 31 cents per ton for a 400 t.p.h. unit, if labour costs are the same.

ACCURACY OF SEPARATION

The dense medium unit separates coal according to its density and so has no control or direct effect on the ash content. For dense media separators the coefficient characterising the intrinsic accuracy of the washing unit is no longer the "imperfection value" which might be expected but the "probable error". The relation between the probable error and the partition density is not simple as in jig washing. The partition density for a given size of particle is the specific gravity of the particle of this size whose velocity of fall in the medium is equal to four cm. per second. (Based on the depth of bath raked by the unit and the rate of travel across the bath, e.g. for Drewboy about ten seconds) The velocity of settling depends upon:-

- particle size and shape,
- particle density,
- bath viscosity,
- influence of hindered settling.

To compare units one may quote the probable error (E) or (E_{pm}), which numerically is half the difference between the densities corresponding to the 75 per cent. and 25 per cent. ordinates as shown in the partition curve. Alternatively one may quote the "imperfection" (I) which is the ratio

$$\frac{\text{probable error}}{\text{partition density} - 1}$$

Various figures have been quoted by operators and manufacturers, for example:-

Drum separators	0.04 E_{pm}
	0.078 I
Cyclones	0.02 E_{pm}

SUMMARY

Heavy medium separation will become the major method of coal preparation. As fine coal is increasingly produced by mechanised mining and as coal is used in smaller particle sizes, cyclones will become of increasing importance. To date no suitable substitute for magnetite has been produced so that we should concentrate on using the best quality magnetite available for the machines and equipment which are best adapted for coal cleaning.

SUMMARY OF QUALITY PARAMETERS

			"Australian "Sources"			
	Fine Flower	Biggenden Concentrate	State "A" Bore Core	State "B" Concentrate	U.S.A.	N.C.B.
Chemical:						
Fe_3O_4	58	78	84	50	90	90
Fe_2O_3	23	15	5	49	3	1
SiO_2	9	3	4	Tr	4	4
Al_2O_3	2	1	2	Tr	1	1
Specific Gravity	4.45	4.05	4.76	5.06	4.88	4.55
Size -53 Microns	96	95	90	88	96	4
Magnetics						
Parbuty Henty	86	98	87	95	94	90
Davis Tube (700 gauss)	59	93	84	61	84	79
Susceptibility (K800) e.m.u./gm	.030	.051	.050	.035	.045	.041
Saturation Moment e.m.u./gm	47	70	71	51	70	65
Coercive Force Oersteds (size 22 x 15 microns)	90	40	40	74	76	47
Settling Rate mm/min.						
As received (pulp S.G. 1.50)	5	21	16	N.D.	10	6
Magnetised (pulp S.G. 1.10)	345	336	390	662	385	215
Rating	8	29	26	20	28	24

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APPENDIX 33.

NEW TECHNIQUES FOR SCHEELITE FLOTATION.

Exploration Department,
P.O. Box 691,
DEVONPORT. TAS.7310.

1st September, 1978.

Mr. H. Wellington,
Chief Chemist & Metallurgist,
Department of Mines,
Wellington Street,
LAUNCESTON. 7250.

Dear Mr. Wellington,

Moina Wrigglite.

I thought that it might be valuable to draw your attention to the attached article, which could be particularly relevant to beneficiation studies of the wrigglite.

Yours sincerely,

Paul Askins,
Senior Geologist.

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How Swedish metallurgists developed

For many years attempts have been made to concentrate scheelite by flotation in the presence of fluorspar, calcite, and apatite which have very similar flotation properties. Until recently, however, it was considered impossible to separate it

At Yxsjöberg in central Sweden (Figure No. 1), AB Statsgruvor mines a low-grade scheelite ore for production of scheelite, chalcopyrite, and fluorspar concentrates. Formerly the scheelite was recovered by gravity concentration. But because of unfavorable prerequisites for this beneficiation method, the operations have given unsatisfactory results until recently. Comprehensive experimental work, however, has resulted in a new selective scheelite flotation process, which is drastically improving the mine economies.

According to laboratory and pilot plant tests the new process yields concentrate which exceeds 65 percent WO_3 in grade and 75 percent in recovery from a feed assaying less than 0.4 percent WO_3. The process came on stream in late 1977.

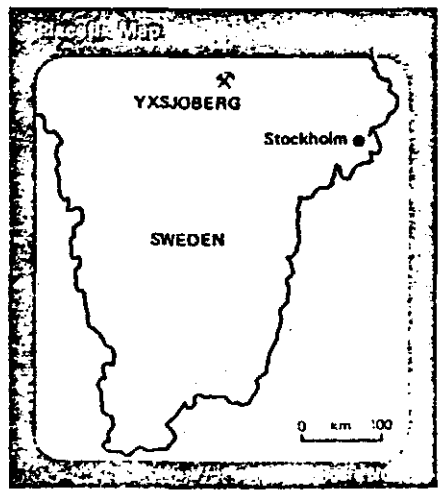
This article describes research and development work which resulted in improvements to the original gravity concentration and finally led to the novel process for selective flotation of scheelite in the presence of fluorspar and calcite. Results from full-scale operation will be presented in a later article.

Statsgruvor is a wholly-owned subsidiary of LKAB, the large, state-owned iron ore mining group. Stoping at the Yxsjöberg underground scheelite mine is mainly cut-and-fill. The run-of-mine ore averages 0.3 to 0.4 percent WO_3. The scheelite in the ore is partly fine-grained and is associated with several times larger quantities of calcite and fluorspar.

Skarn Ore

The Yxsjöberg field belongs to the upper, sodium-rich part of the Svecofennian leptite region, which in the area consists of moderately metamorphic sediments and volcanic rocks of intermediate to acidic composition. After folding and uprising, the complex has been tectonized and amply cut by metabasitic dykes. These strike roughly east-west and are in parallel with the folding. Younger granites, traditionally believed to represent the parent magma of the ore-forming solutions, crop out some 4 miles to the east of the mine.

Mineralization occurred along a limestone horizon in the series, preserved in places but mainly transformed into three somewhat distorted bodies of amphibole and pyroxene skarn. The scheelite is associated with pyrrhotite and chalcopyrite as well as garnets, other skarn materials and abundant amounts of fluorspar.



The pyroxene skarn is characterized by xenomorphic hedenbergite normally coarser than 200 microns and seems to be unevenly impregnated with fairly coarse-grained scheelite. The amphibole skarn on the other hand is on average more fine-grained than the pyroxene skarn. It consists mainly of common hornblende with small scheelite grains fairly evenly distributed in the matrix. Some grain counts exemplifying the scheelite grain size distribution in these ore types are given in Figure No. 2. To a minor extent mineralization also occurred in garnet skarn and in pegmatite veinlets. The fluorspar and calcite proportions vary considerably in all ore types.

On average the run-of-mine ore assays 0.38 percent WO_3, 0.25 percent Cu, 2.2 percent S, 0.02 percent P, and 7.5 percent CaF_2, which corresponds to the following mineral composition:

- 0.5 percent scheelite
- 0.7 percent chalcopyrite
- 5.5 percent pyrrhotite
- 0.1 percent apatite
- 7.5 percent fluorspar
- 5.1 percent calcite
- 5.2 percent quartz
- 10 percent biotite
- 15 percent feldspar
- 50 percent skarn minerals

Reopen Mine

In 1963 low tungsten prices forced the former owner, AB Yxsjö Gruvor, to cease operations. In 1969 AB Statsgruvor acquired the mine and after construction of new facilities production recommenced in 1972. Basically the same flowsheet was adopted for beneficiation as in the old plant.[1]

The ore was crushed in three stages to minus-20-millimeters, each time bypassing the fines, and then ground by rod mills in two stages to minus-0.5-millimeter (Figure No. 3). A rougher scheelite concentrate, mainly consisting of pyrrhotite, skarn minerals, and garnet, was obtained on three-deck shaking tables and further upgraded in two stages on single-deck diagonal tables. Most of the pyrrhotite was discarded by wet magnetic separation and then the scheelite product was dried, roasted, and cleaned from residual impurities by dry low and high intensity magnetic separation.

Rejects from the roughing tables were reground in a ball mill operated in closed circuit with a hydrocyclone. After reducing the pyrrhotite content by magnetic separation, chalcopyrite was floated by conventional methods. The copper rougher concentrate was reground and cleaned in four stages.

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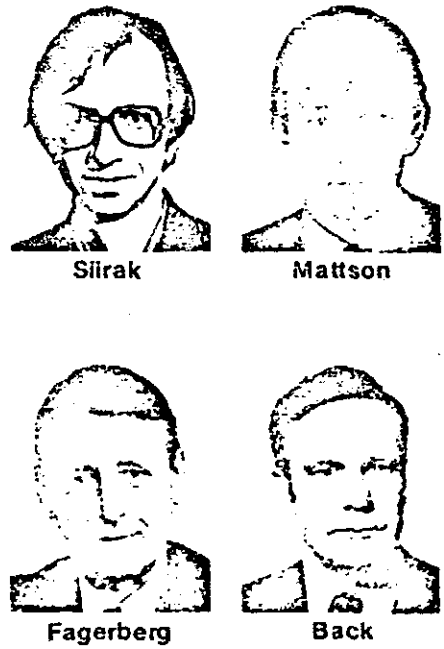
selective scheelite flotation process

from fluorspar, and calcite-bearing ore gave only low grade concentrate or unacceptable recoveries. The new flowsheet described here has successfully solved these problems.

By **Johan Siirak**, research engineer, **Kurt Mattson**, manager Beneficiation Research, **Bengt Fagerberg**, vice president Research and Development LKAB, and **Bo Back**, chief engineer AB Statsgruvor, Sweden.

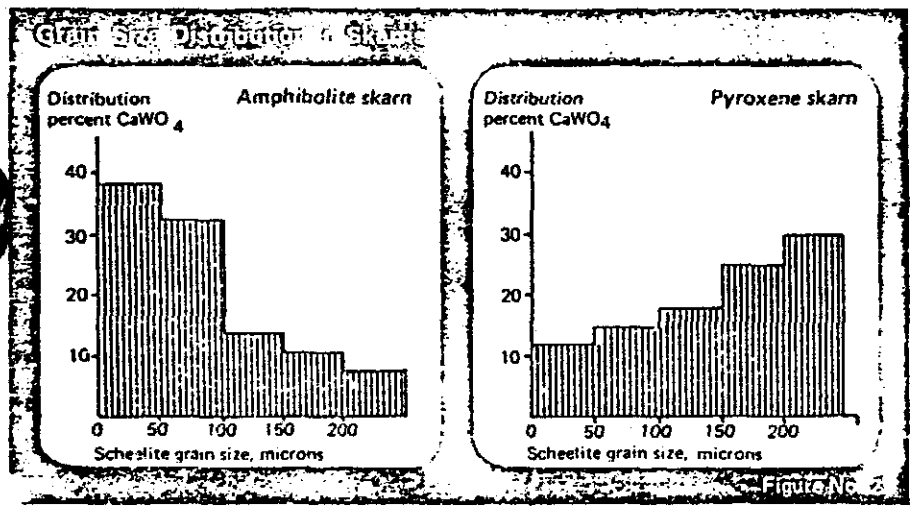


From the copper circuit rejects fluorspar and fine-grained scheelite were floated with tall oil as a collector. Scheelite in the final concentrate and some of the cleaner stage middling was recovered by three-stage tabling and then dried, roasted, and treated as described above.

In spite of great efforts to improve efficiency by adjusting the process, scheelite concentration results remained unsatisfactory for a long time. In 1974 overall scheelite recovery in concentrate assaying 68 to 70 percent WO_3 was still less than 50 percent. Due to this fact, and low tungsten prices, the mine was operating at loss. Therefore a project team with members from AB Statsgruvor and LKAB was appointed in order to develop a beneficiation process for more efficient recovery of scheelite.

Two different approaches were chosen aiming at short and long term improvements respectively. The former comprised modified gravity processes later to be combined with or replaced by flotation, depending on how selective such a process could be made.

Various modifications of the original flowsheet by gravity, magnetic, and flotation methods were developed in bench and pilot plant tests and introduced at the concentrator. These modifications allowed steady progress and together with rising prices helped operations to become profitable. Of particular importance in the development was replacing the three-deck shaking tables by Reichert cone concentrators. This measure raised recovery in the gravity circuit from about 60 percent to about 80 percent. However, the overall scheelite recovery was increased by only some 2 percent. The flowsheet in operation at Yxsjöberg the last year before the switch-over to all flotation is shown in Figure No. 3.



The three-decked tables were originally installed in order to reduce space requirements, but have proven troublesome to operate. Due to their excessive need for mechanical service and the difficulties to control the process, both grade and recovery were inconsistent, thus causing disturbances along the production chain. Replacing the tables with Reichert cone concentrators was a success because the latter stabilized the process, despite the fact that the cone theoretically falls short of tables as a concentrating machine.

Scheelite Flotation Process

For many years attempts have been made to concentrate scheelite by flotation in the presence of fluorspar, calcite, and apatite, which have very similar flotation properties. Until recently, however, it was considered impossible to separate it from fluorspar and only low-grade concentrate or by far inacceptable recoveries were achieved from calcite-bearing ores. Apatite seems to cause comparatively minor troubles.

The primary objective of the flotation tests in this study was to find a method to produce a saleable scheelite concentrate assaying at least 65 percent WO_3 at the highest possible recovery. Otherwise, it was to be investigated

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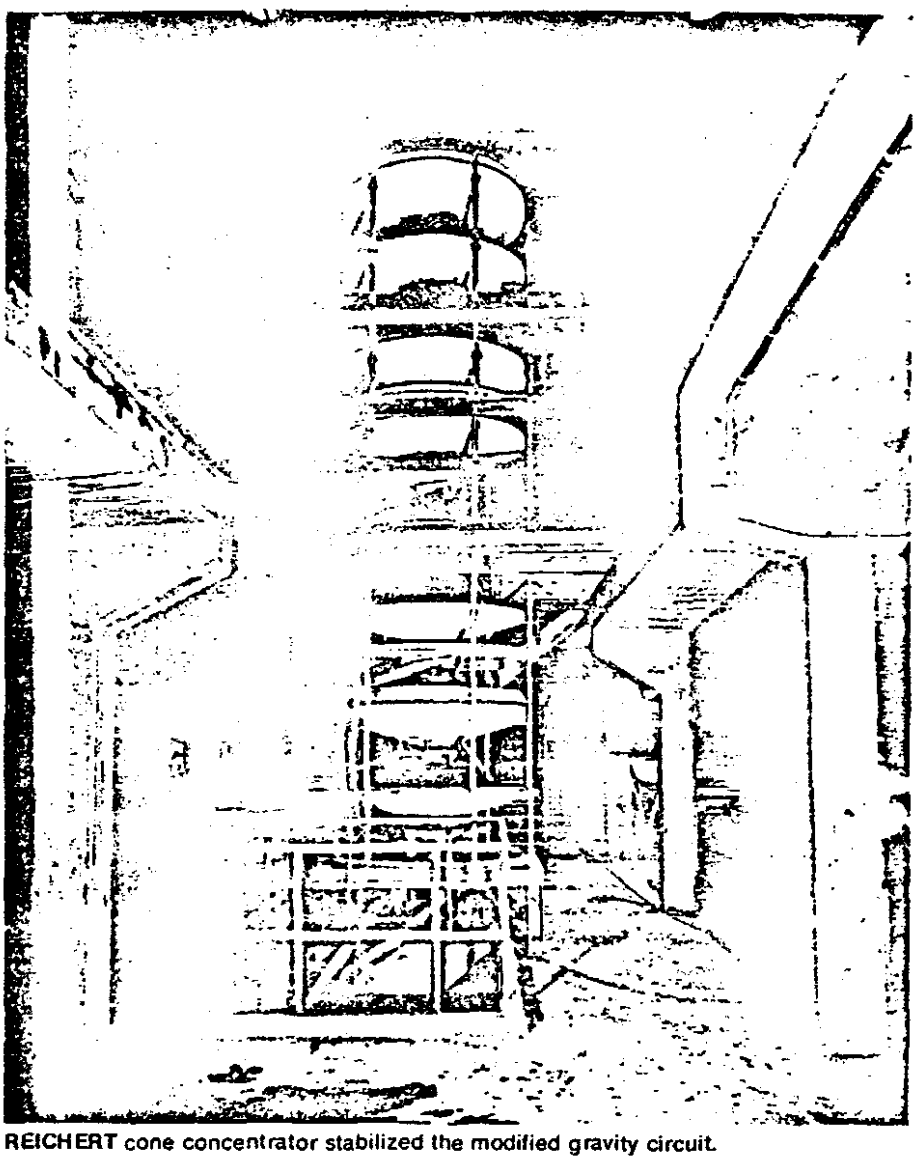


REICHERT cone concentrator stabilized the modified gravity circuit.

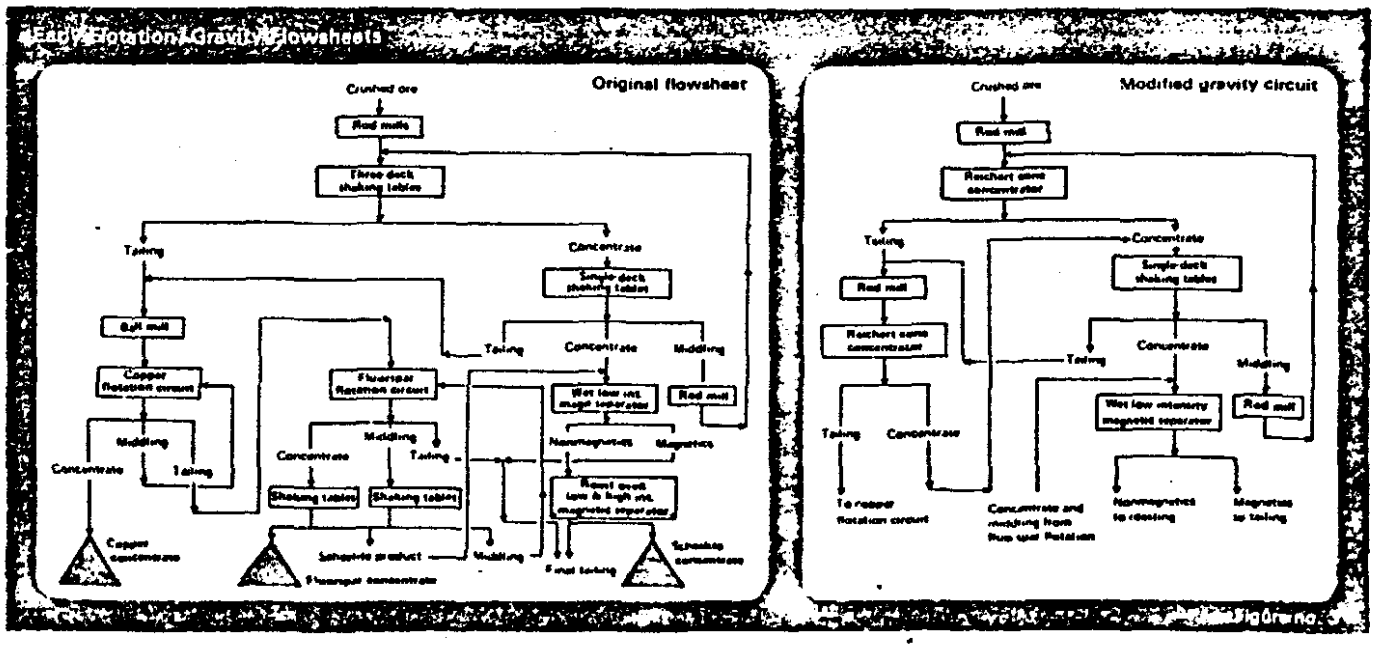
whether low grade concentrates were marketable or could be chemically treated as to give overall economy.

At first, bulk flotation of calcium minerals with subsequent separation was studied. Selective desorption of the collector film was attempted utilizing a method devised by Petrov[2,3] which involves conditioning the bulk concentrate in hot solution with sodium silicate.

With sodium oleate as a collector, desorption was fairly unselective, but it improved if the fatty acid were slightly modified. Very encouraging results were achieved with an amphoteric high molecular unsaturated carboxylic acid (KemaNord Lilaflot OS). In batch tests scheelite concentrate assaying 67 percent WO_3 was produced at an 85 percent overall recovery.

It proved, however, difficult to reproduce these results in the pilot plant as the process is sensitive and difficult to control. Particularly varying calcite contents in the bulk concentrate called for rapid changes of process parameters. Hence it seemed doubtful whether sufficiently steady conditions could be created without using very sophisticated equipment. The flowsheet of this Petrov model process is shown in Figure No. 4.

In parallel to the studies mentioned above tests were also carried out on selective flotation. Here it was tried to find suitable combinations of calcite and fluorspar depressing agents which would permit scheelite to be floated selectively. Chemicals such as soda ash, sodium silicate, and polyphosphates



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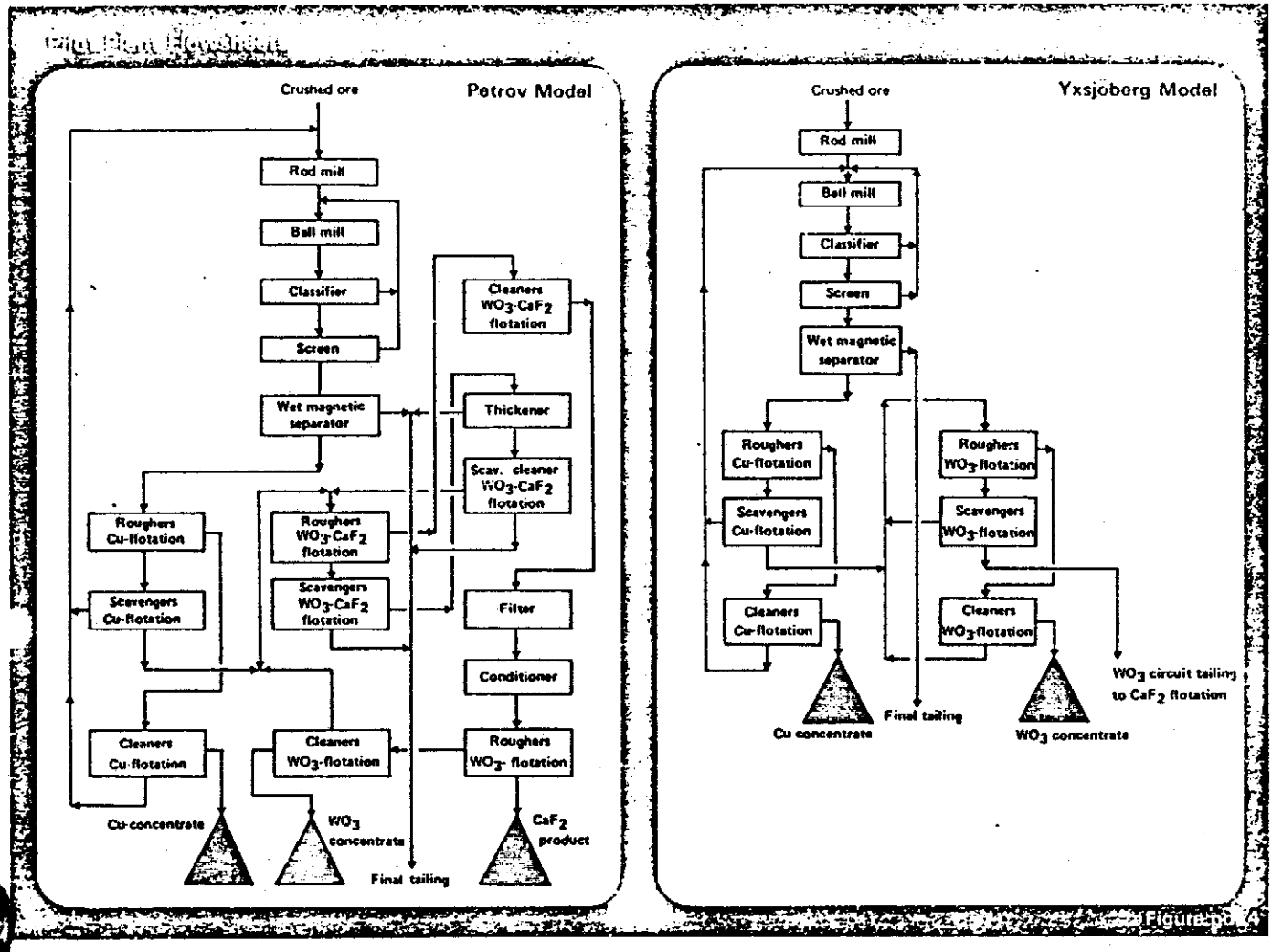

Figure No. 4

Table No. I
Pilot Plant Flotation Tests

Products	Weight percent	Assay, percent Cu	Assay, percent WO_3	Distribution, percent Cu	Distribution, percent WO_3
Magnetics	3.70	0.10	<0.01	2.7	<0.1
Copper concentrate[1]	0.66	16.3	<0.01	75.8	<0.1
Scheelite concentrate	0.53	0.04	68.5	0.3	78.9
Final flotation tailing	95.11	0.03	0.095	21.3	21.1
Feed	100.00	0.14	0.45	100.0	100.0

[1] First cleaner concentrate

Table No. II
Chemical Composition of Scheelite Concentrate, Percent

Component	Percent	Component	Percent
$CaWO_4$	83.2	Mo	0.008
CaF_2	9.2	Mn	0.013
$CaCO_3$	2.5	Fe	2.10
Fe_2O_3	2.64	Cu	0.022
FeO	0.32	Be	0.004
MnO	0.02	Y	0.015
MgO	0.06	Sn	0.0015
Al_2O_3	0.27	Pb	<0.0005
SiO_2	0.60	As	<0.005
TiO_2	0.02	Sb	<0.002
V_2O_5	0.09	Bi	0.023
P_2O_5	0.09	P	0.04
Na_2O	0.04	S	0.085
K_2O	0.03		
	99.08		

with and without addition of metal salts, lignin-sulphonates, and quebracho were tested alone or together in various combinations. As collector sodium oleate was utilized throughout. To obtain optimum results it was also necessary to choose suitable pH and pulp temperature. The best test conditions resulted in high grade rougher concentrate at 80 to 85 percent WO_3 recoveries.

In order to confirm laboratory results, pilot plant tests were run with a capacity of one ton per hour according to the Yxsjöberg model flowsheet given in Figure No. 4. The ore was ground in two stages to 80 percent minus some 140 microns and then pyrrhotite was discarded by magnetic separation. In the first flotation step chalcopyrite and the remainder of pyrrhotite were floated off. In the second stage the rougher scheelite concentrate was cleaned twice. The results of a typical pilot plant test are summarized in Table No. I.

The scheelite concentrate contains extremely small amounts of impurities except some fluorspar, as seen in Table No. II.

In January 1977 it was decided to go ahead with the necessary modifications for a switch-over from gravity concentration to flotation. The concentrator has been reconstructed and production went on stream with the new flowsheet in late 1977. The plant has a rated capacity of 200,000 tons per year.

References

1. Rothelius, E., *Swedish Mineral Dressing Mills*. Stockholm, 1957.
2. Klassen, V. I. and Mokrousov, V. A., *An Introduction to the Theory of Flotation*. London, Butterworth, 1963, 113-114.
3. Mitrofanov, S. I., *Solution of Some Problems Concerning the Theory and Practice of Selective Flotation in the USSR*. Proceedings, 4th International Mineral Congress, Stockholm, Sweden, 1957.

END.

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APPENDIX 34

PRELIMINARY DCF STUDY, MOINA FLUORITE PROSPECT, JULY 1975. G. WESTE

Ore Reserve Tonnage - "Possible Ore"

A ground magnetic survey outlined three highly magnetic areas:-

(1) the main body of skarn, partly exposed at Shepherd and Murphy Mine;

(2) a faulted (down and SE) outlier on the east side of Bismuth Creek (E of the main body);

(3) a small area west of the main body on Brampton Creek.

Possible Ore tonnages are derived from the area of high magnetic values and from the thickness of ore as determined by diamond drilling by Mt. Lyell.

A. Area of Ore:

(1) Main body - 205,100 m^2

(2) Bismuth Creek Outlier - actual anomaly area = 48,000 m^2 but mapping indicates less than this. Assumed that actual area of ore = 20,000 m^2

(3) Brampton Creek Outlier - 19,200 m^2

B. Thickness of Ore:

(1) Main body - 31 m (ML 2)

(2) Bismuth Creek outlier - ML 3A intersected 29.7 vertical m of ore; but entered the ore horizon from below its top. Estimated actual thickness of ore = 40 m

(3) Brampton Creek outlier - no information - assume 25 m

C. Volume of Ore:

(1) Main body	205,100 x 31	=	6.360×10^6 m^3
(2) Bismuth Creek outlier	30,000 x 40	=	1.200×10^6 m^3
(3) Brampton Creek outlier	19,200 x 25	=	0.480×10^6 m^3

D. Possible Tonnes of Ore: S.G. of ore approximately 3.5

(1) Main body	$6.36 \times 10^6 \times 3.5$	= 22.260×10^6 t
(2) Bismuth Creek outlier	$1.20 \times 10^6 \times 3.5$	= 4.200×10^6 t
(3) Brampton Creek outlier	$0.48 \times 10^6 \times 3.5$	= 1.680×10^6 t

TOTAL POSSIBLE ORE = 28.14×10^6 tonnes

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Overburden Tonnage

(1) Main body - basalt SG = 2.6
direct overburden. thickness approximately 25 m
205,100 x 25 x 2.6 = 13.332×10^6 t
additional tonnes for 45° pitslope
over portion (0.7) of area with
circumference 4000 m and cross sectional
area 1800 m^2 = 4000 x 0.7 x 1800 x 2.6 = 13.104×10^6 t
Total = 26.436×10^6 t

(2) Bismuth Creek outlier.
direct overburden: about 3 m. SG = 2.6
30,000 x 3 x 2.6 = 0.234×10^6 t
additional tonnes for 45° pitslope
= 700 m x 1800 x 2.6 = 3.884×10^6 t
Total = 4.118×10^6 t

(3) Brampton Creek outlier.
direct overburden. about 20 m of basalt
and limestone SG 3.0: 19,200 x 20 x 3 = 1.152×10^6 t
additional tonnes for
45° pitslope = 600 x 1012 x 3.0 = 0.822×10^6 t
Total = 2.974×10^6 t

Overall total overburden = 33.528 tonnes

Overburden to Ore ration = 1.2 : 1.0

Mill Head Grade of Ore.

Using C. Ongs revised values for F: Using Amdel's values for Sn

(1) Main body - from 28 analyses ML2 F = 9.75% gives 19.5% CaF_2
from 28 analyses ML2 Sn = 0.14% as SnO_2

(2) Bismuth Creek outlier
from 29 analyses ML3A F = 8.8% gives 17.6% CaF_2
from 29 analyses ML3A Sn = 0.31% as SnO_2

(3) Brampton Creek outlier - no analyses assume as for (1)
CaF_2 = 19.5%
Sn = 0.14%

Total Overall Grade of Ore:

Average grade CaF_2
(2) $4.200 \times 10^6 \times 0.176$ = tonnes pure CaF_2 0.7392×10^6
(1) &(3) $23.94 \times 10^6 \times 0.195$ = tonnes pure CaF_2 4.6683×10^6
total tonnes pure CaF_2 5.4075×10^6
÷ by total tonnes gives overall grade = 19.2% CaF_2

Average grade SnO_2:

(2) $4.200 \times 10^6 \times 0.00031$ = tonnes pure Sn = 0.01302×10^6

(1) & (3) $23.94 \times 10^6 \times 0.00014$ = tonnes pure Sn = 0.03352×10^6

Total tonnes pure Sn = 0.04654×10^6

- by total tonnes gives overall grade = 0.165% Sn

Product Market Price

CaF_2 $70.00 per tonne

SnO_2 $3500.00 per tonne contained Sn

Project Life

19 years at mining rate of 1.5×10^6 tonnes per year plus three years development.

Mill days per year.

300

Mill Recovery

CaF_2 70%

SnO_2 70% (Renison obtain 80% recovery of fine tin)

Concentrate Grade

CaF_2 97%

SnO_2 60% Sn

Concentrate Moisture

10%

Concentrate Freight

47 km mine to Devonport wharf at 8¢ per km	=	$3.76 per t
handling at wharf	=	$0.30 per t
Total:		$4.06 per t

Royalty

At present royalties are charged on quarry materials and iron ore only. However allowance is made in case a royalty is included at a later date. Assume $0.50 per tonne fluorite produced.

Ore Mining Costs

For 5000 tonnes of ore per day = $1.00 per tonne

Overburden Mining Costs

As for ore = $1.00 per tonne

Concentrating Costs

$2.00 per tonne ore

Overhead Indirect Costs

$1.00 per tonne

Regeneration

$0.15 per tonne ore

Total Capital Cost

Concentrator	20.0×10^6
Mine development and equipment	5.0×10^6
Mine exploration and proving	2.0×10^6
Power, water and other services (high tension power lines 2 km away) (HEC Dam 0.5 km away)	1.5×10^6
Townshite	9.0×10^6
Total Capital Cost	$\$37.5 \times 10^6$

Capital Expenditure

$\$3.0 \times 10^6$ per year

Capital Repreciated

Calculated in a straight lineover mine life

$37.5 \times 10^6 \div 19 = 1.974 \times 10^6$/year

Federal Tax Rate

45%

CALCULATION OF OPERATING PROFITS PER YEAR

Plant Characteristics

Tonnes Ore milled	1.5×10^6
Grade CaF_2 reaching concentrator	19.2%
Grade CaF_2 reaching concentrator	0.165%

532 Product Distribution (Million tonnes)

Tonnes pure CaF_2 in concentrator feed	0.288
Tonnes pure Sn in concentrator feed	0.00247
Recovered tonnes pure CaF_2 at 70% recovery	0.2016
Recovered tonnes pure Sn at 70% recovery	0.00172
Tonnes CaF_2 shipped at 97% grade	0.2078
Tonnes Sn shipped at 60% grade	0.00286

Gross Value of Saleable Products (Million dollars)

CaF_2 at $70 per tonne	14.55
SnO_2 at $3500 per tonne contained Sn	6.02
Total:	20.57

Charges Beyond Concentrator (Million dollars)

Concentrate freight charges (plus 10% moisture) at 4.006 per tonne	0.940
Possible royalty of $10.50 per tonne CaF_2 produced	0.100
Total:	1.040

Receipts after Charges

19.52

Operating Costs

Mining + Overheads + Beneficiation + Regeneration	7.72

Operating Profits

11.80

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	End of Year	Operating Profit	Minus Depreciation	Equals Taxabl Income	Net Profit After Tax	Plus Depreciation	Minus Capital Expenditure	Equals Cash Flow	Cumulative Cash Flow	D.C.F. at 10%	D.C.F. at 9%
		(million dollars)									
DEVEL	0	(12.5000)						(12.500)	(12.500)	(12.500)	(12.500)
	1	(12.500)						(12.500)	(25.000)	(11.364)	(11.468)
	2	(12.500)						(12.500)	(37.500)	(10.330)	(10.521)
PRODUCTION	3	11.800	5.922	5.878	3.233	9.155	3.000	6.155	(31.345)	4.624	4.753
	4		1.974	9.826	5.404	7.378		4.378	(26.967)	2.990	3.101
	5								(22.589)	2.718	2.845
	6								(18.211)	2.471	2.611
	7								(13.833)	2.247	2.395
	8								(9.455)	2.042	2.197
	9								(5.077)	1.857	2.016
	10								(0.699)	1.688	1.849
	11	Payback in Production Year 9							3.679	1.534	1.700
	12								8.057	1.395	1.556
	13								12.435	1.268	1.428
	14								16.813	1.153	1.310
	15								21.191	1.048	1.202
	16								25.569	0.953	1.103
	17								29.947	0.866	1.012
	18								34.325	0.788	0.928
	19								38.703	0.716	0.852
	20								43.081	0.651	0.781
	21								47.459	0.591	0.717
	22								51.837	0.538	0.658
										-2.056	-1.525

534

SENSITIVITY OF MOINA DCF JULY 1975

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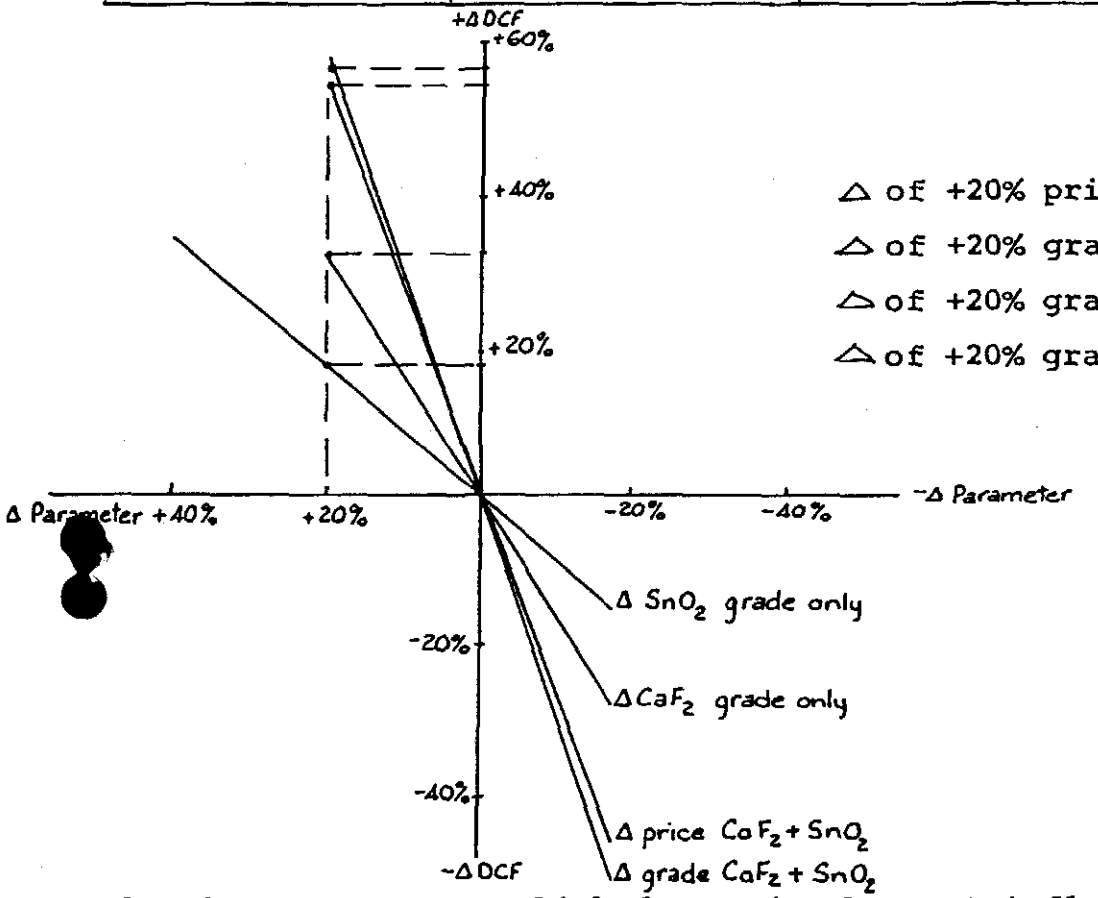
	Price used/tonne	△ +20%	Grade Used	△ +20%
CaF_2	A$70	A$84	19.2%	23.0%
SnO_2	A$3500	A$4200	0.165%	0.198%
DCF $CaF_2 + SnO_2$	9¼%	14¼%	9¼%	14½%
DCF CaF_2 only				12¼%
DCF SnO_2 only				11 %



△ of +20% price (SnO_2+CaF_2) DCF=+54%

△ of +20% grade (SnO_2+CaF_2) DCF=+56%

△ of +20% grade CaF_2 only DCF=+32%

△ of +20% grade SnO_2 only DCF=+18%

The above plot shows which factor is the most influential on the DCF. Both price and grade have a strong influence. An increase in grade of both CaF_2 and SnO_2 of 20% raises the DCF from 9¼% to 14½% - a 56% increase but only slightly more than that caused by a similar increase in the price of both CaF_2 and SnO_2. The lower sensitivity of the DCF to SnO_2 is shown where a 20% increase in the grade of SnO_2 (alone) only raises the DCF by 18%. (Compare with CaF_2 giving 32%).

Therefore we should aim to increase grade and price, but place more emphasis on the CaF_2 than the SnO_2.

NOTE: other factors unchanged:- a CF of 6.3 yr 4 → DCF 14¼%
4.4 → 9¼%
∴ (5.1 → ~ 11 %)
5.6 → 12¼%
∴ (6.5 → DCF 14½%)

CONCLUSIONS.

Annual cash flow $4.4 million

Payback in 9th production year

Capital repaid 2.4 times over the production period

DCF 9%

These figures do not quite meet the returns expected of a mining project for Comalco and so the wrigglite body on the basis of the figures in this study is subeconomic.

The sensitivity analysis however shows that moderate increases in grades can substantially increase the DCF.

The deposit thus warrants serious further evaluation.

APPENDIX 35.

FINANCIAL ANALYSIS, MOINA WRIGGLITE:

SEPT. 1978 - (P. ASKINS)

ASSUMPTIONS:

1. 15 million tonnes 18% CaF_2, 0.1% WO_3, 0.1% Sn
2. Open pittable, ore : overburden 1 : 1
3. Thruput 1.5 million t/yr i.e. 10 yr mine life
4. Assume mill operates for 300 day/yr
5. Recoveries 60% CaF_2 as acid grade
   50% WO_3 as scheelite concentrate
   30% Sn as cassiterite concentrate
   (WO_3 and Sn recoveries are very optimistic).
6. Realized prices (using approximate current prices)

CaF_2	acid grade	\$ 80/t
WO_3	65% WO_3 as scheelite conc.	\$120/m.t.u.
Sn	clean cassiterite conc. say	\$ 90/m.t.u.

7. In-the-ground values, at quoted recoveries:

CaF_2	$\frac{18}{100} \times \frac{60}{100} \times 80$	=	\$ 8.60
WO_3	0.1 x 0.5 x 120	=	\$ 6.00
Sn	0.1 x 0.3 x 90	=	\$ 2.70
			\$17.30

8. Concentrate freight costs:

   for Sn, WO_3 negligible
   for CaF_2 assume concentrate sold F.O.B. Devonport. trucking 100 km to coast at say \$12/tonne.

9. No smelter costs since products sold as concentrates.
10. State Government royalties. Tasmania royalties, Sept. 78, are 2.5% of the gross value of production, or 5% of before tax profit, whichever is less. In this case 5% of before tax profit is less.
11. Ore mining cost \$1.20/tonne
12. Overburden removal cost \$1.20/tonne
13. Milling cost \$10.00/tonne (inflated from Erskine (1975), also approximate Renison cost).
14. Overhead, indirect costs \$1.50/tonne
15. Reclaimation costs negligible

537

16. Capital costs, No townsite necessary

Concentrator (concentrator is complex)	$30 million
Mine development and equipment	$ 4 million
Mine exploration and proving	$ 2 million
	$36 million

17. Capital expenditures each year of operation $ 2 million

18. Assume inflation and commodity prices rise sympathetically.

19. Depreciation of capital costs straight line method $3.6 million/year

OPERATING PROFITS CALCULATION

		Each year of Production.
Tonnes of ore milled		1,500,000
Grade of ore reaching mill		18% CaF_2 0.1% WO_3 0.1% Sn
Tonnes of products in mill feed	CaF_2	270,000
	WO_3	1,500
	Sn	1,500
Tonnes of products recovered	CaF_2	162,000
	WO_3	750
	Sn	450
Gross value of products recovered, $	CaF_2	12,960,000
	WO_3	9,000,000
	Sn	4,050,000
	TOTAL	26,010,000
Tonnes CaF_2 concentrate freighted 97% CaF_2 in concentrate, hence $\frac{162,000 \times 100}{97}$		167,000
Freight charges on CaF_2 concentrate, $ (Sn, WO_3 freight negligible)		2,004,000
No other significant charges, royalty to be figured on operating profit		-
Receipts after charges - equivalent to $16.00/tonne ore milled		say 24,000,000
Operating costs/tonne ore:		
mining ore + overburden	$ 2.40	
milling	$10.00	
overhead/indirect	$ 1.50	
	say $14.00	
Operating profit $16.00 - $14.00 = $2/tonne		$ 3,000,000

CASH FLOW CALCULATION

A detailed cash flow calculation is pointless because the $3 million annual operating profit would be insufficient over the life of the mine, 10 years, to repay the capital costs.

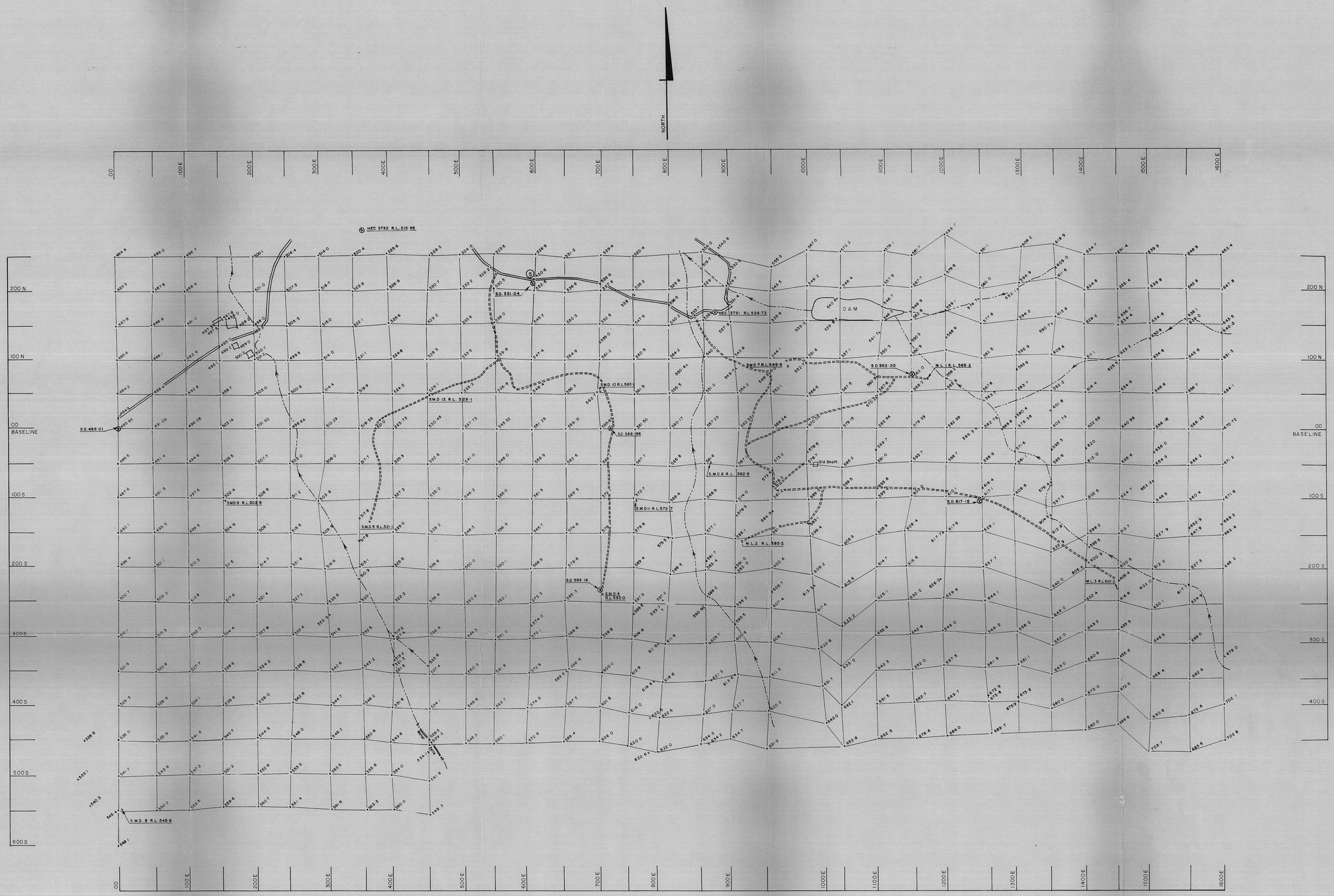
Each year

> Op.profit $3.0 million, less royalty of 5%, gives $2.85 million, minus depreciation $3.6 million means negative taxable income. Net profit after tax is therefore $2.85 million, less annual capital expenditures $2.million gives cash flow of only 0.85 million.

An operating profit exceeding 14 million per year would be necessary to generate a satisfactory cash flow exceeding $6 million per year and a DCF of near 15%.

CONCLUSION

The Moina wrigglite deposit is presently subeconomic.



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5 cm

NOTE:
All R.L.s at ground points are ground level except Base Line where levels are on 100mm x 50mm pegs up to 1250 then on top of 25mm x 25mm stakes to 1600.

ML 1, SMD 12, – Diamond Drill Hole Collars.
SD. – Star Dropper Reference Point.

Revision: Contours south of Baseline 1250 – 1350 amended 15/6/77

Datum for Levels: HEC Bench marks 3731 & 3732

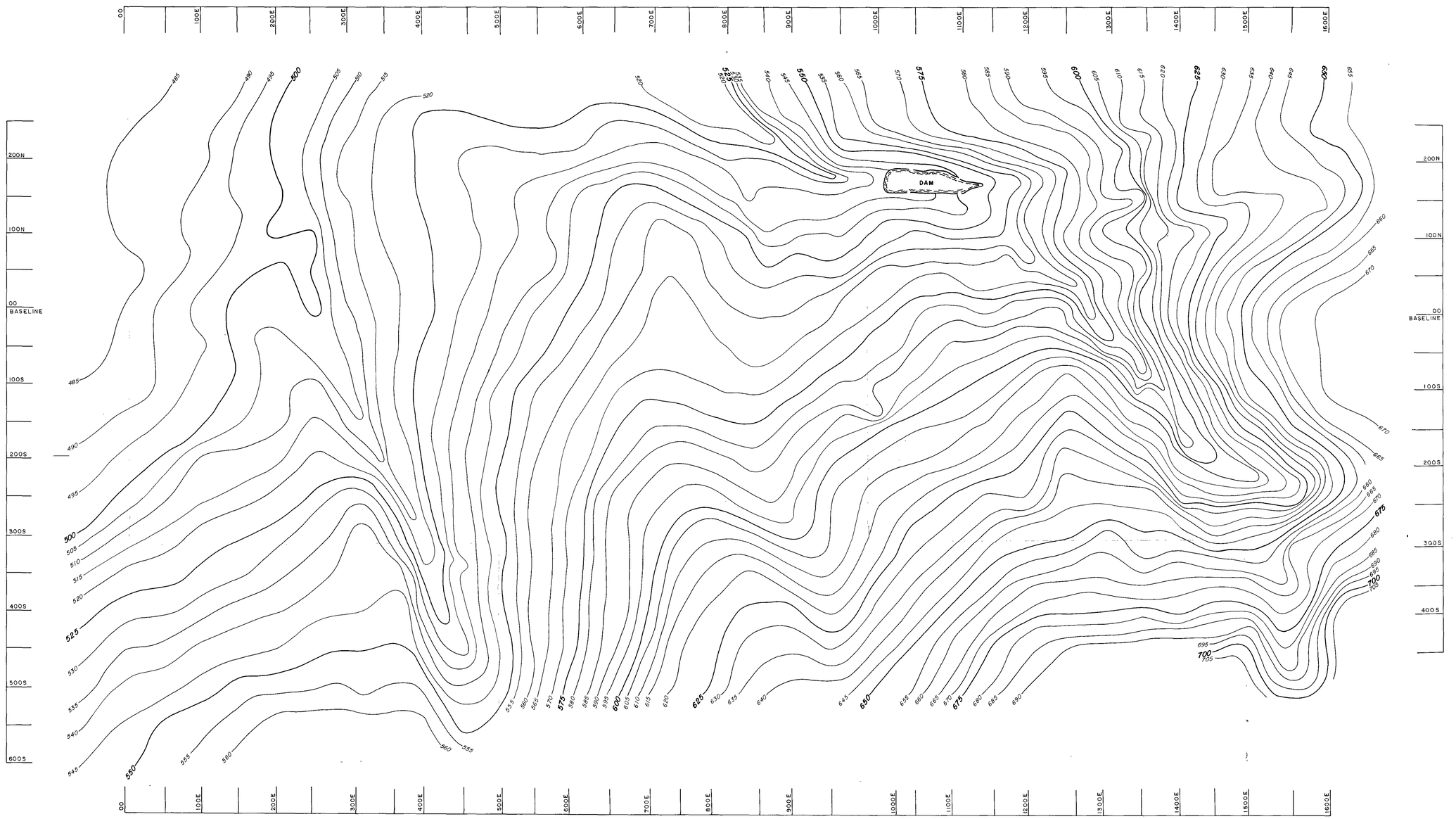
78-1305

COMALCO LIMITED

TASMANIA

CONTOUR & GRID LOCATION SURVEY
E.L.7/74 MOINA TASMANIA
SHEPHERD & MURPHY GRID
DATA SHEET 2424

Compiled: Urban Planners & Surveyors/Tasmania	Revised: August, 1977	Drawn: M. Attwell
Date: 16/9/76	Scale: 1:2,500	Drg N° TAS – 76 – 53



221634

5 cm

Overlay to data plan TAS/76/53.
Datum for Levels, HEC Bench marks 3731 & 3732.

78-1305

COMALCO LIMITED

TASMANIA

CONTOUR & GRID LOCATION SURVEY
E.L.7/74 MOINA TASMANIA
SHEPHERD & MURPHY GRID
CONTOURS

2425

Compiled Urban Planners & Surveyors / Tasm a	Revised: August, 1977	Drawn. M. Attwell
Date 16/9/76	Scale 1:2500	Drg N° TAS-76-53A

Note. Arrows show direction of reading.
Data is incomplete. Information for remainder of grid is missing.

221635

COMALCO LIMITED 78-1305

EL 7/74 TASMANIA

"MOINA" SHEET 2

IRIS RIVER AREA

GRID LINE SLOPES

5 cm

2426

Compiled: P. Askins	Revised:	Drawn: O. Hedditch
Date: Aug 1978	Scale: 1: 2,500	Drg No. TAS-78-162

LEA RIVER

400 S
600 S
800 S
1000 S
1200 S
1400 S
1600 S

3400W 3350W 3300W 3250W 3200W 3150W 3100W 3050W 3000W 2950W 2900W 2850W 2800W 2750W 2700W 2650W 2600W 2550W 2500W 2450W 2400W 2350W 2300W 2250W 2200W 2150W 2100W

Notes: Arrows show direction of reading
Data is incomplete. Information for remainder of grid is missing.

COMALCO LIMITED 78-1305

EL 7/74 TASMANIA
"MQINA" SHEET 3
LEA RIVER AREA
GRID LINE SLOPES

Compiled: P. Askins	Revised:	Drawn: O. Hedditch
Date: Aug. 1978	Scale: 1:2,500	Drg. No. TAS. 78-163

5 cm

221636 2427

200mE 400mE 600mE 800mE 1000mE 1200mE 1400mE 1600mE

600mN 400mN 200mN Base Line 200mS 400mS 600mS 800mS 1000mS 1200mS 1400mS 1600mS

WATER RACE

Bismuth Creek

Brampton Ck

Brampton Creek

20m (grid line error)

LIMITS OF GRIDDING

(SUB BASALT) HUGO'S FAULT

DAM

Tailings

SMD 7

ML 1

50°

Godwins Adit.

SMD 15

North West Branch Lode

SMD 6

No 6 Lode

Main Shaft

SMD 24 (75°)
SMD 25 (65°)

SMD 13

obscure

To Quarry

SMD 12

SMD 10

SMD 11

SMD 9

SMD 5

SMD 4

ML 2
50°

Small veins of qtz, ct, wf.

SMD 16

obscure

Massive Fracture & Resilicified Sandstone

Numerous E-W cuts and stopes

shear zone 275°M qz., ct., wf, mus

No 5 Lode

No 4 Lode

No 2 Lode

ML 3

BISMUTH CREEK FAULT

Pebbly Sst.

Pits quartz-wolfram vein

Zone where bedding obscure. Highly silicified irregular quartz veinlets.

Qtz-wolfram topaz veinlets.

Bedding sub-horizontal.

Zone highly fractured with irregular quartz patches & much pyrite.

slicks

Turbaceous conglomerate

DINES FALLS

LIMIT OF GRIDDING

Cradle Mountain Road

Sub-Crop Roland Conglomerate

Contact with Cambrian rocks possibly just covered by basalt

Qtz, cht, actin magnetite

LEGEND

QUATERNARY

- Alluvium
- bs — Basalt soil

TERTIARY

- (Tb) — Basalt. Massive flows and agglomerates.
- (Tgb) — Greybilly
- (Ts) — Clays, sands and gravels including "Deep Lead"

ORDOVICIAN

- GORDON LIMESTONE (Og) — Fine grained Limestone
- Calcarenite
- MOINA SANDSTONE (Om) — Sandstones. Tubicular, quartz wackes and fine "quartzites." Tends to Siltstone at top of sequence.

METAMORPHOSED, METASOMATISED EQUIVALENTS (DEVONIAN)

- Magnetite chlorite rock with sulphides
- "Wrigglite". Minor calc silicate rocks.
- Calc silicate, garnet pyroxene rocks. Minor wrigglite and metasiltstone ("Transition beds")
- Calc silicate rock, meta siltstone, limestone Finely bedded.
- Meta siltstones

- Bridge
- Shaft
- Diamond drill hole.
- Waste rock pile
- Fault (approx) (Dashed where concealed or inferred)
- Fence
- Shearing
- 070-25 — Strike, Dip of bedding
- Syncline
- Anticline
- Costean or trench.
- Change of slope (scarp)
- Definite geological boundary
- Approximate geological boundary
- Road
- Graded gravel track
- Walking track
- Creek
- Excavation or cliff
- Deep Lead (Approx boundary)
- Sink hole
- 2732 — H.E.C. Bench mark.

221637

5 cm

0 100 200 300 400 metres

SCALE

COMALCO LIMITED 78-1305

EL 7/74 TASMANIA

"MOINA" SHEET 1

SHEPHERD & MURPHY AREA

GEOLOGY

2428

Compiled G. Weste	Revised P. Askins Aug., 1977	Drawn M. Attwell.
Date: August., 1976.	Scale: 1:2,500	Drg. N° TAS-76-46

200E 400E 600E 800E 1000E 1200E 1400E 1600E

1500N 1400N 1300N 1200N 1100N 1000N 900N 800N 700N 600N

BASALT & TERTIARY SEDIMENTS

TO WILMOT DAM

015 30

QUARRY

TO CRADLE MT ROAD

LIMIT OF GRIDDING

285 55

Very weathered calc silicate rock with quartz vein.

TRENCH

Sh

Sh

Sh

Pits in Alluvium

TRENCH

TO CRADLE MT ROAD

LIMIT OF GRIDDING

5 cm

COMALCO LIMITED 78-1305

E.L.7/74 TASMANIA

"MOINA" SHEET 1 EXTENSION

SHEPHERD & MURPHY AREA

GEOLOGY

2429

221638

Compiled: R. Poltock.	Revised:	Drawn M. Attwell.
Date: August, 1977	Scale 1: 2,500	Drg N° TAS-77-65

LAKE GAIRDNER

TO STORMONT LEA RIVER

POSSIBLE ANTICLINE?? PLUNGE EAST

MAYBE GREYBILLY

SOME GREYBILLY??

TERTIARY SEDIMENTS

TERTIARY SEDIMENTS?

SOME SILTY FLOAT

TERTIARY (CLAYEY SILT)

CAMP SITE 1975-76 SEASON

IRIS BRIDGE

CHANNEL

CHANNEL

SINK

Breakaway

LIMIT OF GRIDDING

TO CRADLE MT ROAD

ROAD TO IRIS RIVER

Sh

SMD 14

SMD 17

SMD 21

SMD 20

SMD 22

SMD 23

SMD 8

outcrop

Soil, clay over limestone

lineation 125 30

cleavage 20

Water goes underground in this sink hole.

LEPTOSPERMUM SWAMP

IRIS RIVER

IRIS RIVER FRACTURE ZONE

Old Saw Mill

DOM 2

DOM 3

TASMANIA TREE CREEK TRACK

BASE LINE IRIS GRID
500mS 1880mW to 000

Qtz chlorite actinolite veins

Qtz veins with >Zn S >Pbs

Qtz, chl, actin magnetite

200m to Roland Conglomerate

COMALCO LIMITED 78-1305

221639 E.L. 7/74 TASMANIA

"MOINA" SHEET 2

IRIS RIVER AREA
GEOLOGY

5 cm

2439

Compiled: R. Poltock.	Revised: P.W. Askins Aug, 1977	Drawn: M. Attwell
Date: August, 1976	Scale: 1: 2,500	Drg N° TAS-76-47

0 100 200 300 400 metres

SCALE

600 W 1400 W 1200 W 1000 W 800 W 600 W 400 W 200 W 000

1500 N 1400 N 1300 N 1200 N 1100 N 1000 N 900 N 800 N 700 N 600 N

ISLAND

S/S TUBICULAR RIPPLE MARKS

295 30

QUARTZ ACTINOLITE BLOCKS
POSSIBLE FAULT ?

TRIG

GOSSAN, PYRITIC SANDSTONE FLOAT

GOSSAN FLOAT

LIMIT OF GRIDDING

Sh

22640

COMALCO LIMITED 78-1305

E.L.7/74 TASMANIA

"MOINA" SHEET 2 EXTENSION

IRIS RIVER AREA
GEOLOGY

5 cm

2431

Compiled: R. Poltock.	Revised:	Drawn: M. Attwell
Date August, 1977	Scale: 1:2,500	Drg N° TAS-77-66

3400mW 3200mW 3000mW 2800mW 2600mW 2400mW 2200mW 2000mW 1800mW

600mN 400mN 200mN 000 200mS 400mS 600mS 800mS 1000mS 1200mS 1400mS 1600mS 1800mS

MAGNETIC NORTH

TO IRIS RIVER BRIDGE

Qtz chlorite Pbs veins

AEROMAG ANOMALY 35 HEREABOUTS

LEA RIVER

IRIS RIVER

FRACTURE ZONE

POSSIBLY COVERS LST OR CS SYN. TROUGH

LEA RIVER

WALKING TRACK TO DEVONPORT MINE

LIMITS OF GRIDDING

Qtz veins with Cp, Py, Pb S, Zn S.

FLETCHER'S ADIT AREA

TRENCHS

TRACK TO FLETCHER'S ADIT

Greywacke or Sandstone erratics?

BASE LINE FOR 1800 W TO 3400mW

LEA RIVER

STORMONT MINE

WATER CHANNEL

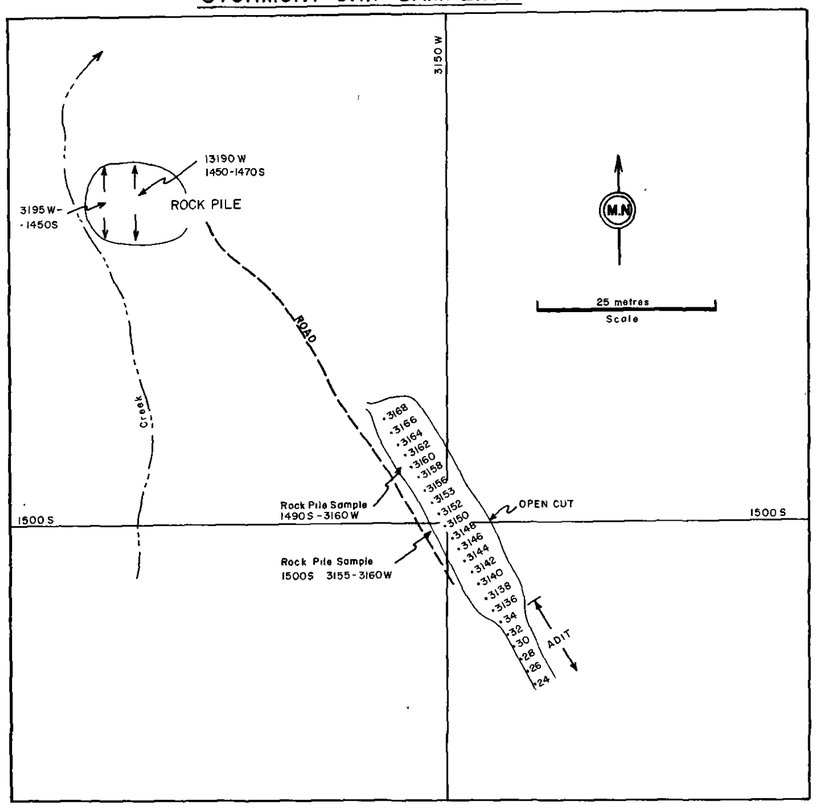
LIMITS OF GRIDDING

BASELINE IRIS GRID
500mS 1880mW to 000

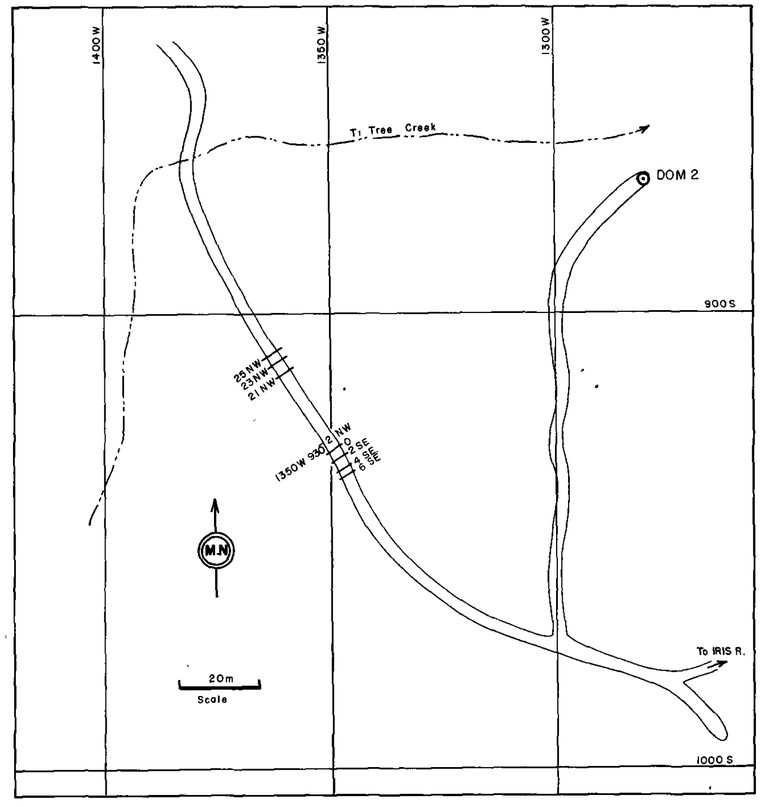
SCALE 0 100 200 300 400 metres

COMALCO LIMITED		
221641 EL 7/74 TASMANIA "MOINA" SHEET 3		78-1305 Vol 6/7
LEA RIVER AREA GEOLOGY		2432
Compiled: R. Poltock	Revised: P.W. Askins Oct. 1976	Drawn M. Attwell
Date August, 1976	Scale: 1: 2,500	Drg. N° TAS-76-48

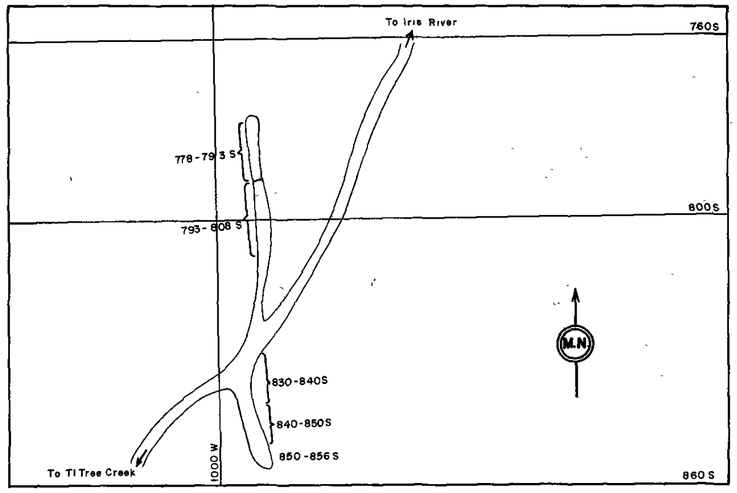
STORMONT CHIP SAMPLING



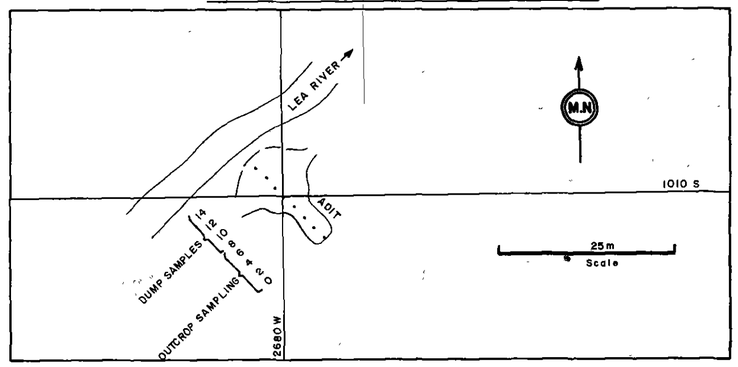
TI TREE CK ROAD & COSTEAN CHIP SAMPLING



COSTEAN IRIS-TI TREE CK. ROAD

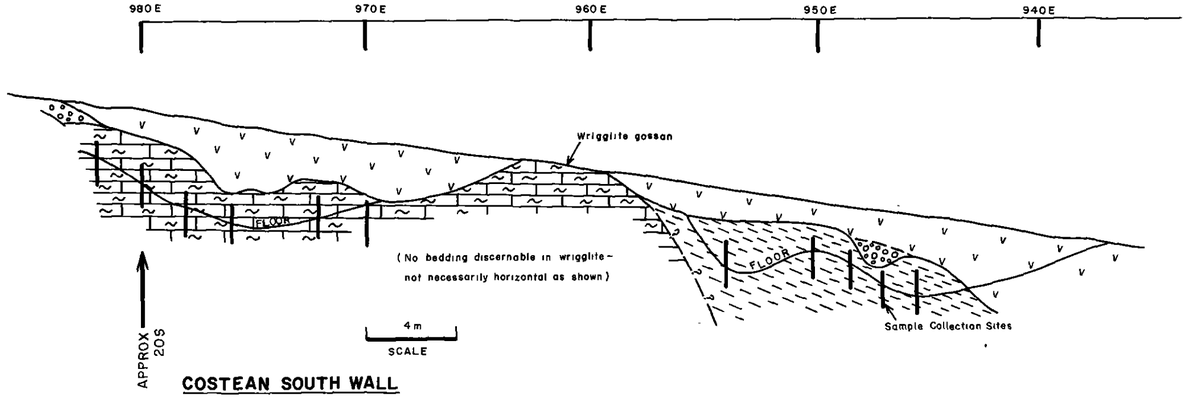


FLETCHERS ADIT CHIP SAMPLING



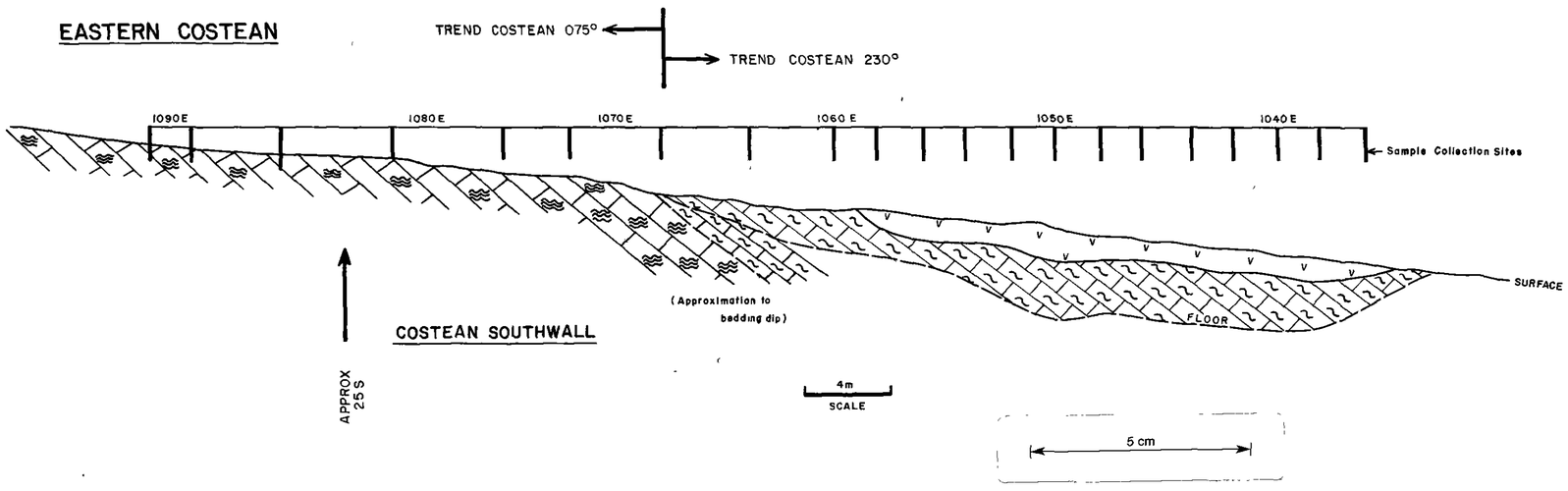
COSTEANS - SHEPHERD & MURPHY AREA

WESTERN COSTEAN



COSTEAN SOUTH WALL

EASTERN COSTEAN



COSTEAN SOUTHWALL

L E G E N D

- *Basalt agglomerate, weathered*
- *Alluvium (deep lead) some partly cemented (greybilly)*
- *Lake sediment*
  Silts, clays, pale grey to black, finely bedded, semi-consolidated some rounded quartz pebbles
- *Wrigglite, mostly highly weathered*
- *Calc-silicate rock*

NOTE- Samples collected are chip samples from costean floor and auger samples from within the lake sediment

221642

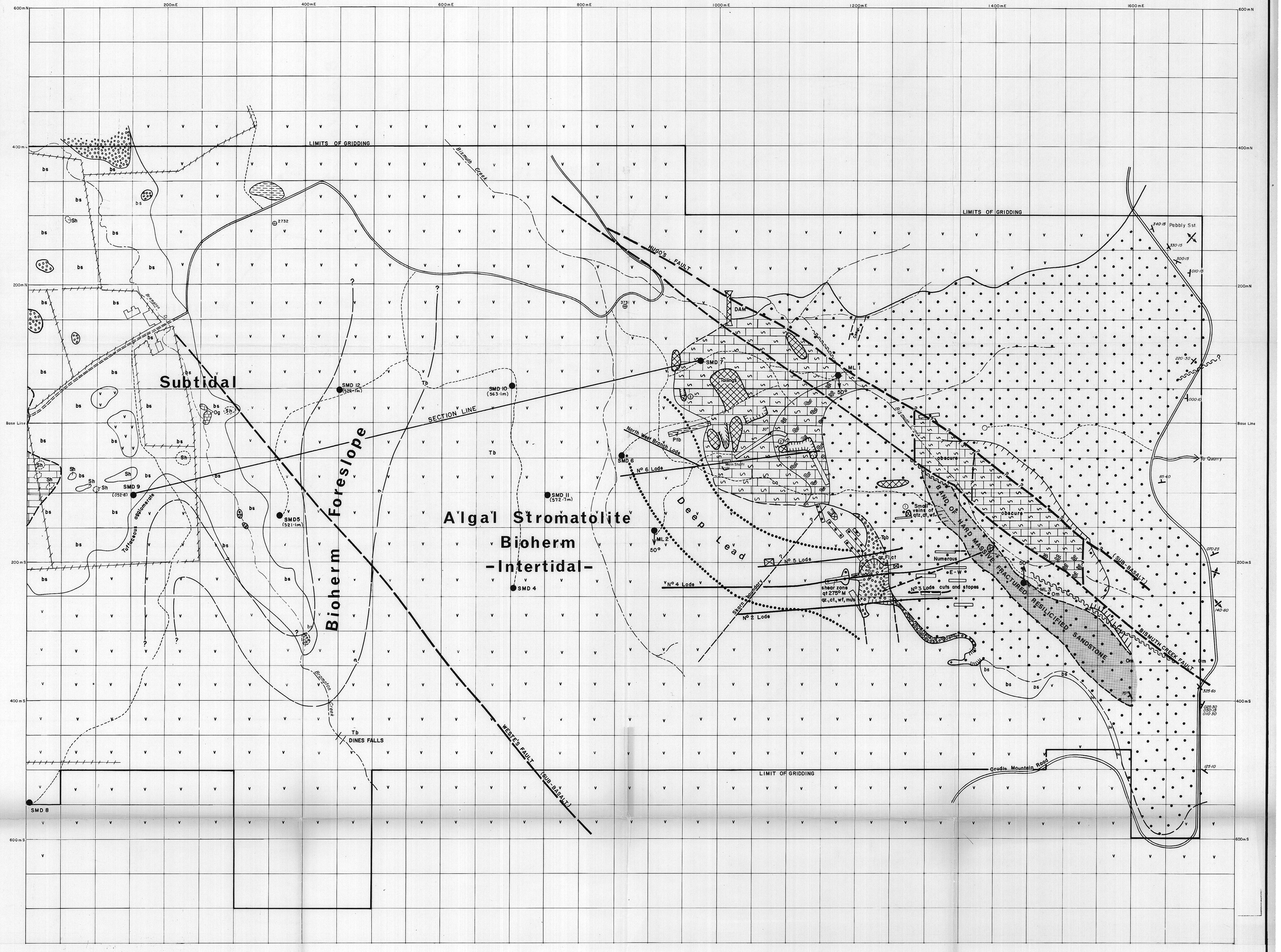
COMALCO LIMITED

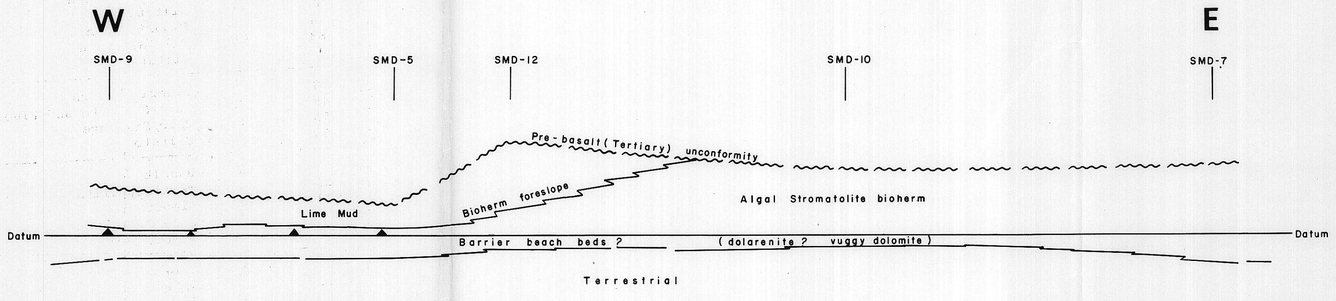
EL 7/74 TASMANIA 78-1305
MOINA AREA

COSTEAN SAMPLING (1976 FIELD SEASON)

2433

Compiled R Pollock P W Askins	Revised	Drawn M Attwell
Date June, 1978	Scale	Drg N° TAS-78-137

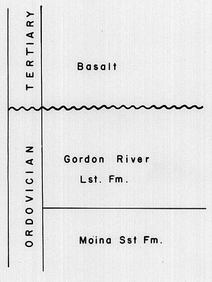




FACIES SECTION

Datum - Base Algal Stromatolite unit.

Scale: 1:2500



Subtidal - Lime mud

Intertidal - Algal Stromatolites; tidal flat(s) with tidal channels.

Supratidal - Dolomitic beds, coarse

▲ Sulphides (Fe-rich)

5 cm

2434

221643 FIG N° 1 2434

COMALCO LIMITED

E.L 7/74 TASMANIA 78-1305

"MOINA" SHEET 1

SHEPHERD & MURPHY AREA GEOLOGY & DEPOSITIONAL ENVIRONMENT OF GORDON RIVER LST FM.

Compiled: G.Weste/P.Askins	Revised: P.W. Stainton	Drawn: M. Attwell.
Date: Aug., 1976 / Jan. 1977	Scale: 1: 2,500	Drg. N° TAS-76-58

78-1305

E.L ~~[illegible]~~ 7/74

221644

76/54 P.1/2

2435

HOLE NO.	GRID REFERENCE	TOTAL DEPTH (metres)	From	To	LOG Geology (Generalized)
SMD 4 ✓	422940 m E 5406240 m N	109	0	66	Basalt
			66	68	Tertiary Sand
			68	90	Fluorite skarn and Calc/Silicate
			90	95	Calc/Silicate
			95	109	Siltstone
SMD 5 ✓	422700 m E 5406400 m N	81	0	15	Basalt
			15	22	Tertiary Silts
			22	38	Limestone
			40	46	Fluorite skarn
			46	61	Calc/Silicate, some Fluorite skarn
			61	81	Siltstone
SMD 6 ✓	423120 m E 5406460 m N	102	0	21	Basalt
			21	67	Fluorite skarn
			67	74	Siltstone
			74	83	Fluorite skarn
			83	91	Calc/Silicate
			91	102	Siltstone
SMD 7 ✓	423260 m E 5406560 m N	71	0	45	Fluorite skarn
			45	55	Calc/Silicate and Fluorite skarn
			55	65	Calc/Silicate
			65	71	Siltstone
SMD 8	422230 m E 5406060 m N	60	0	25	Basalt
			25	33	Tertiary Silts
			33	40	Limestone
			40	46	Calc/Silicate
			46	60	Siltstone
SMD 9 ✓	422450 m E 5406460 m N	130	0	12	Clay
			12	91	Limestone
			91	118	Pyrrhotite Skarn and Skarn
			118	125	Fault Breccia
			125	130	Siltstone

221645

HOLE NO.	GRID REFERENCE	TOTAL DEPTH (metres)	LOG From	To	Geology (Generalized)
SMD 10✓	422970 m E 5406560 m N	117	0	34	Basalt
			34	39	Tertiary Silts, Sand
			39	55	Fluorite Skarn
			55	61	Siltstone
			61	76	Fluorite Skarn
			76	90	Calc/Silicate and some Fluorite Skarn
			90	117	Siltstone
SMD 11	422970 m E 5406400 m N	120	0	31	Basalt
			31	38	Fluorite Skarn and Calc/Silicate
			38	60	Fluorite Skarn
			60	68	Fluorite Skarn and Siltstone
			68	72	Calc/Silicate
			72	120	Siltstone, Sandstone.
SMD 12✓	422810 m E 5406560 m N	124	0	23	Basalt
			23	40	Tertiary muds and sands
			40	91	Limestone
			91	110	Calc/Silicate and fluorite skarn
			110	123	Siltstone, sandstone

2434

221646

79-1305

PROJECT/FIELD SECTION: MOINA (TASMANIA)
BOREHOLE/FIELD LOC'N: ML-2
R.L. (Csg Coll): ..... T.D. 317m (1040ft)
DRILL/FIELD LOG DATE: From ..... To 76/54A

DEPTH(m) or SAMPLE No | GRAPHIC LOG (incl. particles, mud, silt, sand, etc.) 1 2 3 4 5 6 7 8 9 | COLOUR | SEDIMENTARY STRUCTURES & GRAIN SIZE (f, med, crs) | BEDDING ATTITUDE | METAMORPHIC STRUCTURES (cleavage, stylolites, etc.) | MINERALIZATION & ALTERATION — DIAGENETIC & METASOMATIC — METASOMATIC: Pyr, Mgn, Chl, Gnt | LITHO. LOG | REMARKS | ENV. | STRAT. Mbr. Fm.

140 — 150 — 160 — 50 — 170 — 180 — 190 — 60 — 200 — 210 — 220 — 230 — 70 — 240 — 250 — 260 — 80 — 270 — 280 — 290 — 90 — 300 — 310 — 320 — 100 — 330 — 340 — 110

42 —

crs

gy-bf

bf

gy-gn

bf

gy-gn

crs

No cores available from 102 m (335') to total depth at 317 m (1040')

Basalt

Bndst:

Bndst: crs, dk brn

Pkst: dol, alg, vuggy

Mdst: vf, rex, HCl ↑

Pkst: pel, crs, grn, intbd w

Bndst: dk brn, m - crs

Bndst: dk brn, crs

Bndst: dk brn, crs, w tn lam. of Pkst

Pkst: crs, brwd dol, w occ tn lam of alg, vg intbd w

Mdst: gy - gn, chl

Pkst: as above

Mdst: lt brn - lt gy intbd w

Sltst: lt brn - lt gy, calc

No cores available

To T.D. 317m

Tidal ch. levee

Tidal (Algal) Flat

Supra - tidal

Lagoonal

LEGEND

PARTICLES:
- Microbioclastic hash
- Mud lump
- Shell fragments (general)
- Pelletoid
- 'birdseye'
- Spicules
- Fossil - gasteropod (broken)
- Fossil - pelecypod (broken)
- Fossil - brachiopod
- Fossil - arthropod
- Algal stromatolite (Stl.) (undulatory - lateral)
- Algal stromatolite (digitate)
- Fossil, coral
- xx Recrystallized (Rex)

COLOUR:
dk - dark; lt - light; gn - green; r - red; gy - grey; brn - brown; blk - black; wh - white; bf - buff

SEDIMENTARY STRUCTURES:
- Burrows (Brw)
- Bedded (Bd)
- Laminated (Lam)
- Convoluted (Cnvtd)
- 'flame' structure
- Foreset bedding / Lam.
- Ripple - marked

GRAIN SIZE:
f - fine; med - medium; crs - coarse; crs - very coarse; pbl - pebble; congl - conglomerate; aggl - agglomerate; bc - breccia

METAMORPHIC:
clv - cleavage; stylolite

MINERALIZATION:
Pyr - Pyrite; Pyrro - Pyrrhotite; Fl - Fluorite; Chl - Chlorite; Gnt - Garnet; Mgn - Magnetite; Glc - Glauconite; Lmn - Limonite

QUANTITATIVE LOG
present; common; abundant; very abundant; saturated

ATTITUDE:
(a) (b)
70° / (20°)
(a) Dip in relation to vertical (long) axis of core
(b) Dip in relation to horizontal plane.

CARBONATE ROCKS:
Classification (After R.J. Dunham)

a - Lime Mudstone (Mdst): Muddy carbonate, <10% grains
w - Wackestone (Wkst): Mud-supported carbonate, >10% grain bulk
p - Packstone (Pkst): Grain-supported, muddy carbonate
g - Grainstone (Grnst): Grain-supported, mud free, carbonate rocks
b - Boundstone (Bndst): Carbonates showing interconnecting (binding). Skeletal or organic remnants.
e.g. (i) colonial coral reef
(ii) algal mats (undulatory - lat.) and digitate (vert.) algal colonies - stromatolites

ENV. Depositional environment

COMALCO LIMITED

OTHER DATA:

5 cm

Logged by P.W. Stainton

Scale 1:200

Drg N° TAS-76-54 A

2438

78-1305

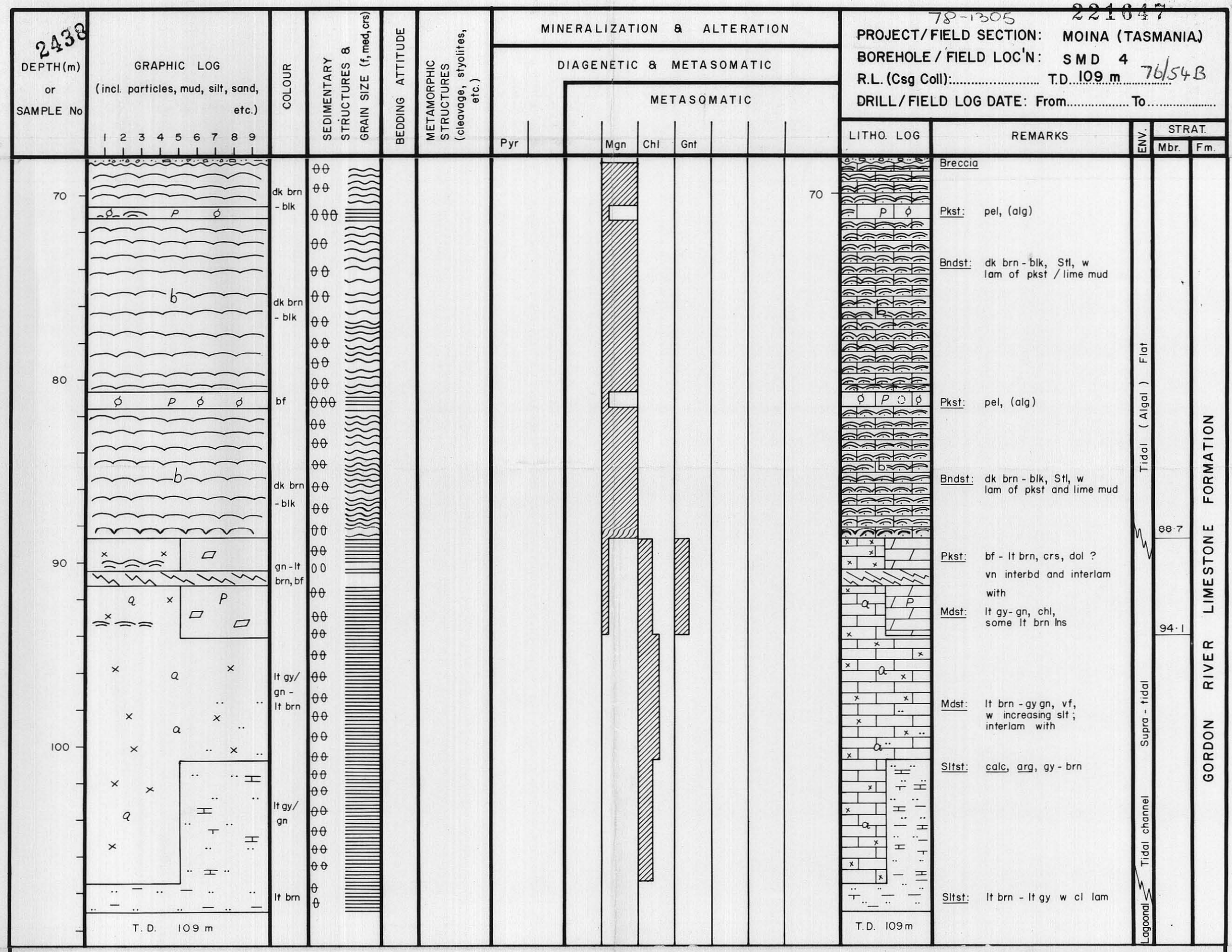
221647

PROJECT/FIELD SECTION: MOINA (TASMANIA)

BOREHOLE / FIELD LOC'N: SMD 4 76/54B

R.L. (Csg Coll):..... T.D. 109 m

DRILL/FIELD LOG DATE: From..... To.....



LEGEND

PARTICLES:
- Microbioclastic hash
- Mud lump
- Shell fragments (general)
- Pelletoid
- 'birdseye'
- Spicules
- Fossil - gasteropod (broken)
- Fossil - pelecypod (broken)
- Fossil - brachiopod
- Fossil - arthropod
- Algal stromatolite (Stl.) (undulatory - lateral)
- Algal stromatolite (digitate)
- Fossil, coral
- Recrystallized (Rex)

COLOUR:
dk - dark; lt - light; gn - green; r - red; gy - grey; brn - brown; blk - black; wh - white; bf - buff

SEDIMENTARY STRUCTURES:
- Burrows (Brw)
- Bedded (Bd)
- Laminated (Lam)
- Convoluted (Cnvtd)
- 'flame' structure
- Foreset bedding / Lam.
- Ripple - marked

GRAIN SIZE:
f - fine; med - medium; crs - coarse; crs - very coarse; pbl - pebble; congl - conglomerate; aggl - agglomerate; bc - breccia

METAMORPHIC:
clv - cleavage; stylolite

MINERALIZATION:
Pyr - Pyrite; Pyrro - Pyrrhotite; Fl - Fluorite; Chl - Chlorite; Gnt - Garnet; Mgn - Magnetite; Glc - Glauconite; Lmn - Limonite

QUANTITATIVE LOG

present; common; abundant; very abundant; saturated

ATTITUDE:
(a) (b)
70° / (20°)
(a) Dip in relation to vertical (long) axis of core
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g - Grainstone (Grnst): Grain-supported, mud free, carbonate rocks

b - Boundstone (Bndst): Carbonates showing interconnecting (binding). Skeletal or organic remnants.
e.g. (i) colonial coral reef
(ii) algal mats (undulatory - lat.) and digitate (vert.) algal colonies - stromatolites

ENV. Depositional environment

COMALCO LIMITED

OTHER DATA:

5 cm

Logged by P.W. Stainton

Scale 1: 200

Drg N° TAS-76-54 B

78-1305 221648

2439

PROJECT/FIELD SECTION: MOINA (TASMANIA)

BOREHOLE / FIELD LOC'N: SMD 5 76/54C

R.L. (Csg Coll):..... T.D.....

DRILL/FIELD LOG DATE: From..... To.....

DEPTH(m) or SAMPLE No	GRAPHIC LOG (incl. particles, mud, silt, sand, etc.)	COLOUR	SEDIMENTARY STRUCTURES & GRAIN SIZE (f, med, crs)	BEDDING ATTITUDE	METAMORPHIC STRUCTURES (cleavage, styolites, etc.)	MINERALIZATION & ALTERATION: DIAGENETIC & METASOMATIC: METASOMATIC: Pyr	Mgn	Chl	Gnt	LITHO. LOG	REMARKS	ENV.	STRAT. Mbr.	STRAT. Fm.
30		gy / wh								22	Wkst: gy wh, rex	Sub-tidal		GORDON RIVER LIMESTONE FM.
		gy-brn									Mdst: gy - wh, rex			
		gy									Pkst: pel, (alg)			
		gy / wh									Mdst: gy - wh, rex			
40		gy / bf									Wkst: gy - wh, rex.			
		dk brn - blk									Mdst:	Tidal (Algal) Flat		
50		mtld	crs								Bndst: dk brn - blk, Stl, w tn lam of Pkst			
		mtld									Sltst: lam, containing org and cl matrix			
		mtld									Pkst: dol, alg (pel) bf - mtld, crs	Supra-tidal		
60												Lagoonal		
70														

LEGEND

PARTICLES:
- Microbioclastic hash
- Mud lump
- Shell fragments (general)
- Pelletoid
- 'birdseye'
- Spicules
- Fossil - gasteropod (broken)
- Fossil - pelecypod (broken)
- Fossil - brachiopod
- Fossil - arthropod
- Algal stromatolite (Stl.) (undulatory - lateral)
- Algal stromatolite (digitate)
- Fossil, coral
- xx Recrystallized (Rex)

SEDIMENTARY STRUCTURES:
- Burrows (Brw)
- Bedded (Bd)
- Laminated (Lam)
- Convoluted (Cnvtd)
- 'flame' structure
- Foreset bedding / Lam.
- Ripple - marked

MINERALIZATION:
- Pyr - Pyrite
- Pyrro - Pyrrhotite
- Fl - Fluorite
- Chl - Chlorite
- Gnt - Garnet
- Mgn - Magnetite
- Glc - Glauconite
- Lmn - Limonite

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- g - Grainstone (Grnst): Grain-supported, mud free, carbonate rocks
- b - Boundstone (Bndst): Carbonates showing interconnecting (binding). Skeletal or organic remnants.
  - e.g. (i) colonial coral reef
  - (ii) algal mats (undulatory - lat.) and digitate (vert.) algal colonies - stromatolites

ENV. Depositional environment

GRAIN SIZE:
- f - fine
- med - medium
- crs - coarse
- crs - very coarse
- pbl - pebble
- congl - conglomerate
- aggl - agglomerate
- bc - breccia

COLOUR:
- dk - dark
- lt - light
- gn - green
- r - red
- gy - grey
- brn - brown
- blk - black
- wh - white
- bf - buff

METAMORPHIC:
- clv - cleavage
- stylolite

QUANTITATIVE LOG
- present
- common
- abundant
- very abundant
- saturated

ATTITUDE:
(a) (b)
70° / (20°)
(a) Dip in relation to vertical (long) axis of core
(b) Dip in relation to horizontal plane.

COMALCO LIMITED

OTHER DATA:

5 cm

Drg N° TAS-76-54C

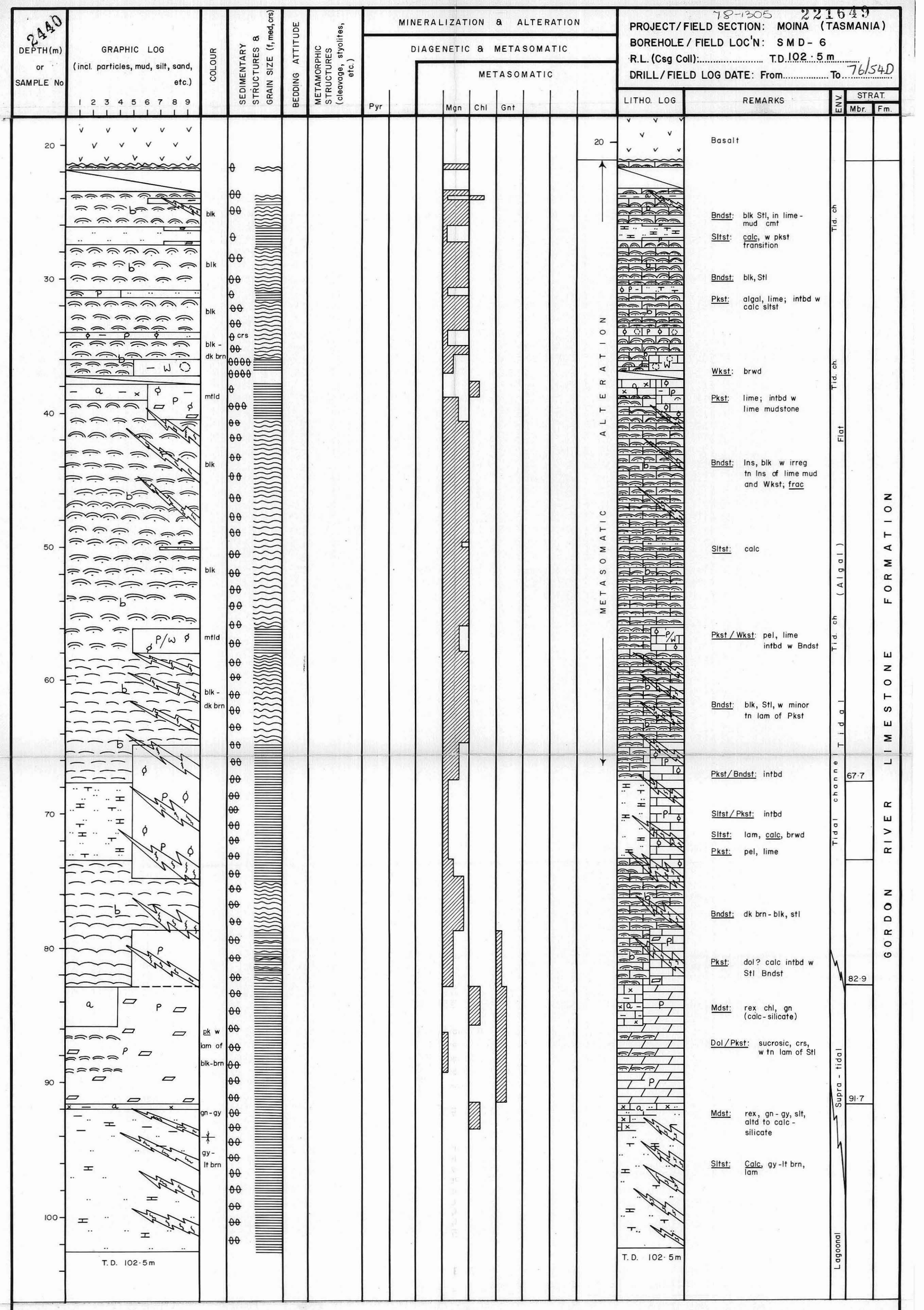
Scale 1:200

Logged by P.W. Stainton

2440

78-1305 221649

PROJECT/FIELD SECTION: MOINA (TASMANIA)
BOREHOLE / FIELD LOC'N: SMD-6
R.L. (Csg Coll): ..... T.D. 102·5 m
DRILL/FIELD LOG DATE: From ..... To 76/54D



LEGEND

PARTICLES:
- Microbioclastic hash
- Mud lump
- Shell fragments (general)
- Pelletoid
- 'birdseye'
- Spicules
- Fossil - gasteropod (broken)
- Fossil - pelecypod (broken)
- Fossil - brachiopod
- Fossil - arthropod
- Algal stromatolite (Stl.) (undulatory - lateral)
- Algal stromatolite (digitate)
- Fossil, coral
- xx Recrystallized (Rex)

COLOUR:
- dk - dark
- lt - light
- gn - green
- r - red
- gy - grey
- brn - brown
- blk - black
- wh - white
- bf - buff

SEDIMENTARY STRUCTURES:
- Burrows (Brw)
- Bedded (Bd)
- Laminated (Lam)
- Convoluted (Cnvtd)
- 'flame' structure
- Foreset bedding / Lam.
- Ripple - marked

GRAIN SIZE:
- f - fine
- med - medium
- crs - coarse
- crs - very coarse
- pbl - pebble
- congl - conglomerate
- aggl - agglomerate
- bc - breccia

METAMORPHIC:
- clv - cleavage
- stylolite

MINERALIZATION:
- Pyr - Pyrite
- Pyrro - Pyrrhotite
- Fl - Fluorite
- Chl - Chlorite
- Gnt - Garnet
- Mgn - Magnetite
- Glc - Glauconite
- Lmn - Limonite

QUANTITATIVE LOG
- present
- common
- abundant
- very abundant
- saturated

ATTITUDE:
(a) (b)
70° / (20°)
(a) Dip in relation to vertical (long) axis of core
(b) Dip in relation to horizontal plane.

CARBONATE ROCKS:
Classification (After R.J. Dunham)

a - Lime Mudstone (Mdst):	Muddy carbonate, <10% grains
w - Wackestone (Wkst):	Mud-supported carbonate, >10% grain bulk
p - Packstone (Pkst):	Grain-supported, muddy carbonate
g - Grainstone (Grnst):	Grain-supported, mud free, carbonate rocks
b - Boundstone (Bndst):	Carbonates showing interconnecting (binding). Skeletal or organic remnants. e.g. (i) colonial coral reef (ii) algal mats (undulatory - lat.) and digitate (vert.) algal colonies - stromatolites

ENV. Depositional environment

COMALCO LIMITED

OTHER DATA:

5 cm

Scale 1:200

Logged by P.W. Stainton

Drg N° TAS-76-54 D

78-1305 221650

241

PROJECT/FIELD SECTION: MOINA (TASMANIA)
BOREHOLE / FIELD LOC'N: SMD 7
R.L. (Csg Coll):..... T.D. 71·5 m
DRILL/FIELD LOG DATE: From..... To 76/54E

DEPTH(m) or SAMPLE No

GRAPHIC LOG (incl. particles, mud, silt, sand, etc.)

COLOUR

SEDIMENTARY STRUCTURES & GRAIN SIZE (f, med, crs)

BEDDING ATTITUDE

METAMORPHIC STRUCTURES (cleavage, styolites, etc.)

MINERALIZATION & ALTERATION

DIAGENETIC & METASOMATIC

METASOMATIC

Pyr | Mgn | Chl | Gnt

LITHO. LOG

REMARKS

ENV.

STRAT. Mbr. | Fm.

10 20 30 40 50 60 70

blk-dk brn
blk-dk brn
mtld
mtld
blk-dk brn
vn
blk-dk brn
lt brn-gn
bf-pk
bf-pk
lt gn
bf-pk
lt gn
lt brn-pk/lt gn
lt brn-lt gy

crs

T.D. 71·5 m

wthd, soil lyr

Bndst: blk-dk brn, Stl, w tn lns of Pkst, brws

Int lam Pkst / Mdst, (dol) cmt, (alg)

Pkst: crs, wh-org, mtld w tn, occ alg lam

Bndst: blk-dk brn vf, org content

Vn

Bndst: blk-dk brn, lam

Pkst: lam, alg

Bndst: blk-dk brn, w tn lam of Pkst.

Pkst: pk, dol, (chl) mtld, Glc ?

Pkst: pk-gy, dol

Mdst: chl, (calc-silicate)

Intbd Mdst / pk dol Pkst

Sltst: lt brn-lt gy, lam, calc, intlam w calc Cl

T. D. 71·5 m

Tidal (Algal) Flat

Tid.ch.

Tid.ch.

50·3

Supra-tidal

Lagoonal

GORDON RIVER LIMESTONE FORMATION

LEGEND

PARTICLES:
- Microbioclastic hash
- Mud lump
- Shell fragments (general)
- Pelletoid
- 'birdseye'
- Spicules
- Fossil - gasteropod (broken)
- Fossil - pelecypod (broken)
- Fossil - brachiopod
- Fossil - arthropod
- Algal stromatolite (Stl.) (undulatory - lateral)
- Algal stromatolite (digitate)
- Fossil, coral
- xx Recrystallized (Rex)

SEDIMENTARY STRUCTURES:
- Burrows (Brw)
- Bedded (Bd)
- Laminated (Lam)
- Convoluted (Cnvtd)
- 'flame' structure
- Foreset bedding / Lam.
- Ripple - marked

MINERALIZATION:
- Pyr - Pyrite
- Pyrro - Pyrrhotite
- Fl - Fluorite
- Chl - Chlorite
- Gnt - Garnet
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  - e.g. (i) colonial coral reef
  - (ii) algal mats (undulatory - lat.) and digitate (vert.) algal colonies - stromatolites

COLOUR:
dk - dark
lt - light
gn - green
r - red
gy - grey
brn - brown
blk - black
wh - white
bf - buff

GRAIN SIZE:
f - fine
med - medium
crs - coarse
crs - very coarse
pbl - pebble
congl - conglomerate
aggl - agglomerate
bc - breccia

METAMORPHIC:
- clv - cleavage
- stylolite

QUANTITATIVE LOG
- present
- common
- abundant
- very abundant
- saturated

ATTITUDE:
(a) (b)
70° / (20°)
(a) Dip in relation to vertical (long) axis of core
(b) Dip in relation to horizontal plane.

ENV. Depositional environment

COMALCO LIMITED

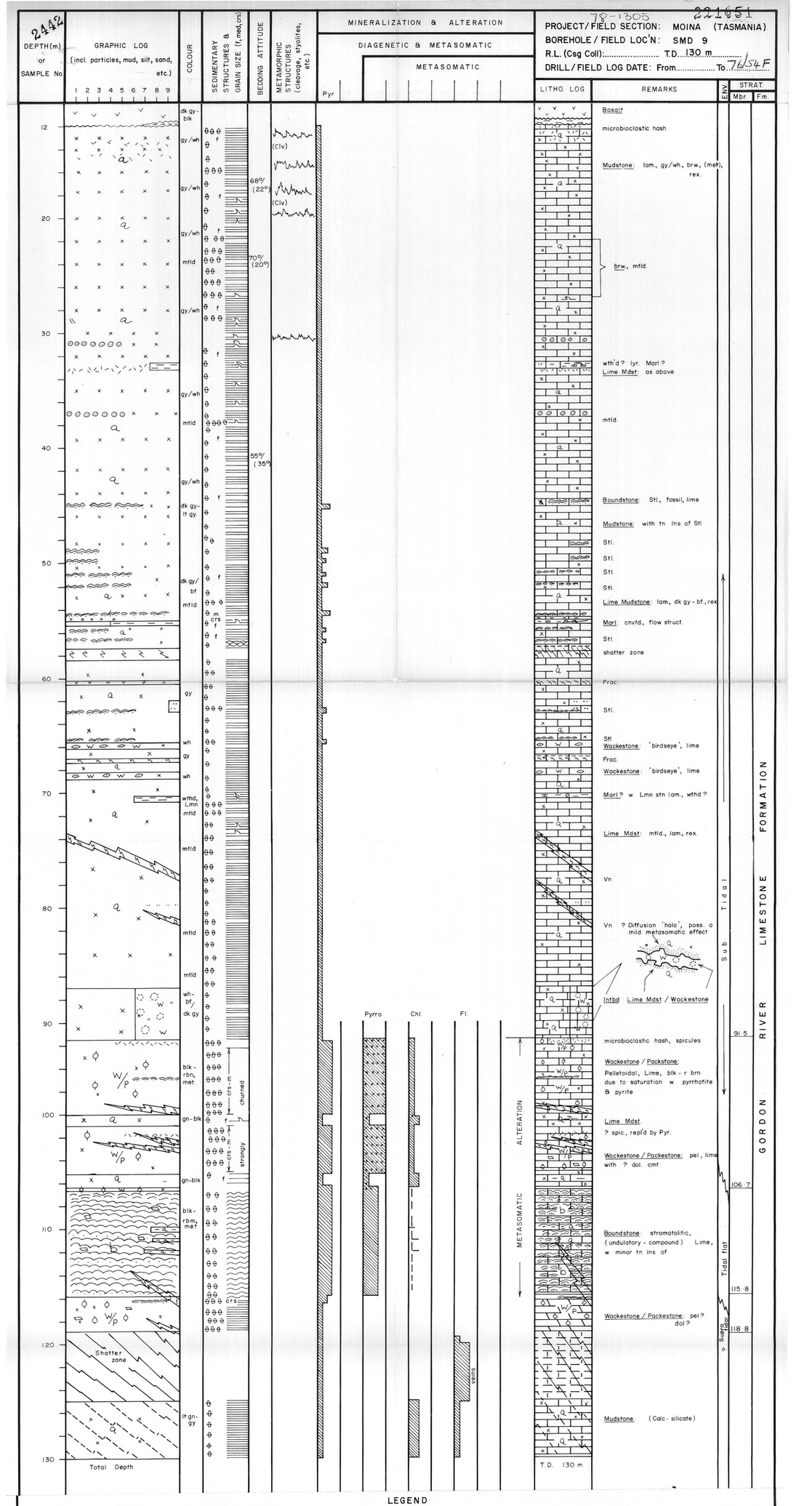
OTHER DATA:

5 cm

Logged by P.W. Stainton

Scale 1:200

Drg N° TAS-76-54 E



LEGEND

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- 'birdseye'
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- Fossil - arthropod
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- Algal stromatolite (digitate)
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- stylolite

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- Fl - Fluorite
- Chl - Chlorite
- Gnt - Garnet
- Mgn - Magnetite
- Glc - Glauconite
- Lmn - Limonite

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- present
- common
- abundant
- very abundant
- saturated

ATTITUDE:
(a) (b)
70° / (20°)
(a) Dip in relation to vertical (long) axis of core
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ENV. Depositional environment

COMALCO LIMITED

OTHER DATA:

5 cm

Scale 1:200

Logged by P.W. Stainton

Drg Nº TAS-76-54 F

78-1305

2443

DEPTH(m) or SAMPLE No | GRAPHIC LOG (incl. particles, mud, silt, sand, etc.) | COLOUR | SEDIMENTARY STRUCTURES & GRAIN SIZE (f, med, crs) | BEDDING ATTITUDE | METAMORPHIC STRUCTURES (cleavage, styolites, etc.) | MINERALIZATION & ALTERATION: DIAGENETIC & METASOMATIC: METASOMATIC (Pyr) | LITHO. LOG | REMARKS | ENV. | STRAT. Mbr. | STRAT. Fm.

PROJECT / FIELD SECTION: MOINA 221652
BOREHOLE / FIELD LOC'N: SMD – 10
R.L. (Csg Coll): ..... T.D. 117 m
DRILL / FIELD LOG DATE: From ..... To 76/54 G

DEPTH (m)	COLOUR	SEDIMENTARY STRUCTURES & GRAIN SIZE	REMARKS
42	bf-gy	m-crs	Basalt
			Dol: prim, m-crs, bf, sucr
	bf	m-crs	Stl.
50			Pkst: dol, alg
	bf	m-crs	Bndst: stl, dol
60	bf-gn	f-m	Bndst: stl, dol
70			Stl.
			Bndst: stl, dol
80	bf-gn		Pkst: dol, slt, alg
			Qz. congl. calc. cast
90			Pkst: dol, slt.
			Siltst / Mdst: lt gy - lt brn, calc, lam, carb
100			
110			Sst: m-f, si, mass
			T.D. 117 m
120			

ENV: Inter-tidal (Algal flat); Supra-Tidal; (Brackish) Lagoonal; Terrestrial

STRAT: GORDON LIMESTONE Fm. / MOINA SANDSTONE Fm

LEGEND

PARTICLES:
- Microbioclastic hash
- Mud lump
- Shell fragments (general)
- Pelletoid
- 'birdseye'
- Spicules
- Fossil - gasteropod (broken)
- Fossil - pelecypod (broken)
- Fossil - brachiopod
- Fossil - arthropod
- Algal stromatolite (Stl.) (undulatory - lateral)
- Algal stromatolite (digitate)
- Fossil, coral
- xx Recrystallized (Rex)

SEDIMENTARY STRUCTURES:
- Burrows (Brw)
- Bedded (Bd)
- Laminated (Lam)
- Convoluted (Cnvtd)
- 'flame' structure
- Foreset bedding / Lam.
- Ripple - marked
- Scour (erosional) surface

MINERALIZATION:
- Pyr - Pyrite
- Pyrro - Pyrrhotite
- Fl - Fluorite
- Chl - Chlorite
- Gnt - Garnet
- Mgn - Magnetite
- Glc - Glauconite
- Lmn - Limonite

CARBONATE ROCKS: Classification (After R.J. Dunham)
- a - Lime Mudstone (Mdst): Muddy carbonate, <10% grains
- w - Wackestone (Wkst): Mud-supported carbonate, >10% grain bulk
- p - Packstone (Pkst): Grain-supported, muddy carbonate
- g - Grainstone (Grnst): Grain-supported, mud free, carbonate rocks
- b - Boundstone (Bndst): Carbonates showing interconnecting (binding). Skeletal or organic remnants.
  - e.g. (i) colonial coral reef
  - (ii) algal mats (undulatory - lat.) and digitate (vert.) algal colonies - stromatolites

COLOUR:
- dk - dark
- lt - light
- gn - green
- r - red
- gy - grey
- brn - brown
- blk - black
- wh - white
- bf - buff
- dpld - dappled

GRAIN SIZE:
- f - fine
- med - medium
- crs - coarse
- crs - very coarse
- pbl - pebble
- congl - conglomerate
- aggl - agglomerate
- bc - breccia

METAMORPHIC:
- clv - cleavage
- stylolite

QUANTITATIVE LOG
- present
- common
- abundant
- very abundant
- saturated

ATTITUDE:
(a) (b)
70° / (20°)
(a) Dip in relation to vertical (long) axis of core
(b) Dip in relation to horizontal plane.

ENV. Depositional environment

OTHER DATA:

5 cm

Scale 1:200

Logged by P.W. Stainton

Drg N° TAS-76-54 G

78 1305

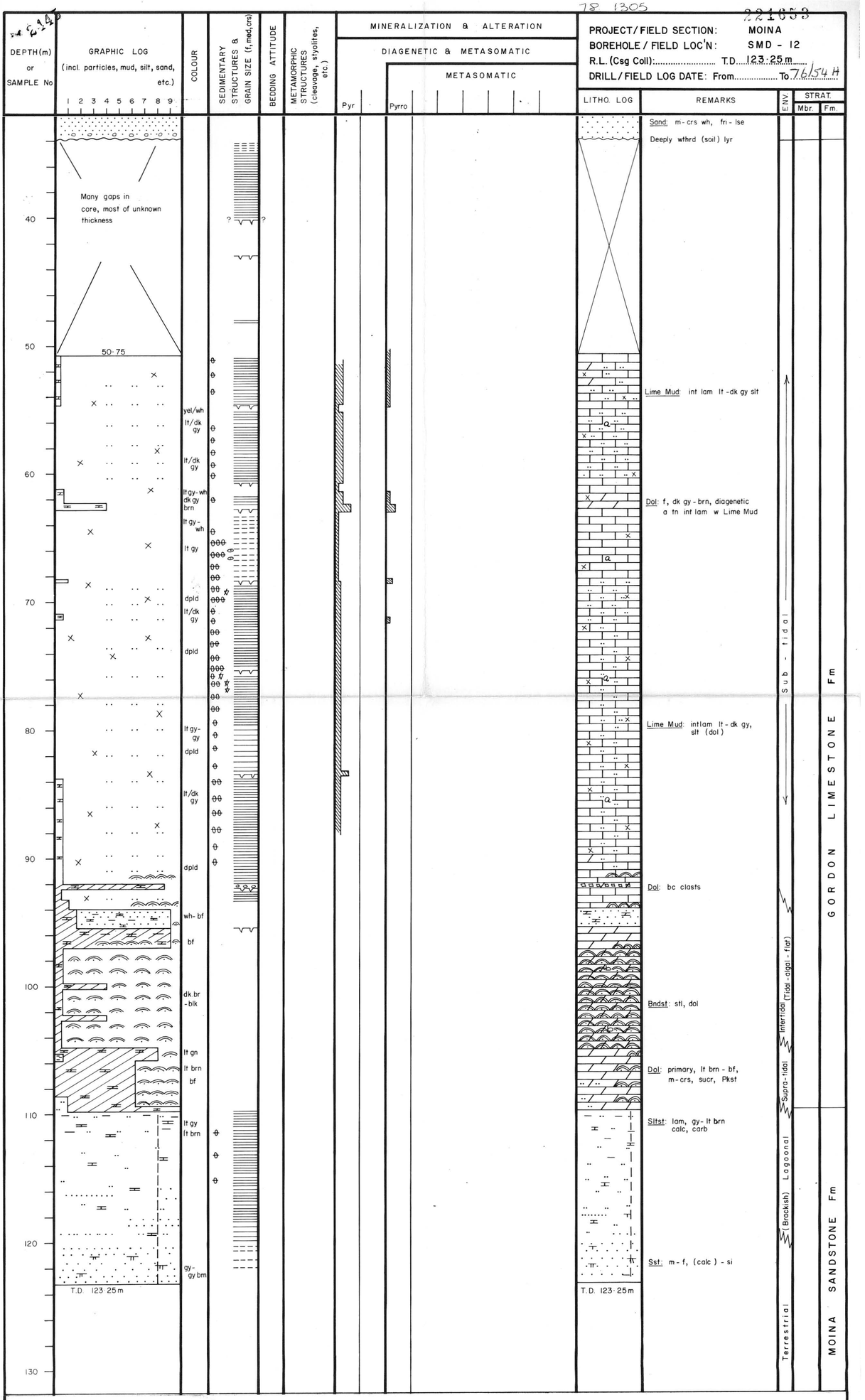
221053

PROJECT/FIELD SECTION: MOINA

BOREHOLE/FIELD LOC'N: SMD - 12

R.L. (Csg Coll): ..... T.D. 123·25 m

DRILL/FIELD LOG DATE: From ..... To 76/54 H



LEGEND

PARTICLES:
- Microbioclastic hash
- Mud lump
- Shell fragments (general)
- Pelletoid
- 'birdseye'
- Spicules
- Fossil - gasteropod (broken)
- Fossil - pelecypod (broken)
- Fossil - brachiopod
- Fossil - arthropod
- Algal stromatolite (Stl.) (undulatory - lateral)
- Algal stromatolite (digitate)
- Fossil, coral
- xx Recrystallized (Rex)

COLOUR:
dk - dark
lt - light
gn - green
r - red
gy - grey
brn - brown
blk - black
wh - white
bf - buff
dpld - dappled

SEDIMENTARY STRUCTURES:
- Burrows (Brw)
- Bedded (Bd)
- Laminated (Lam)
- Convoluted (Cnvtd)
- 'flame' structure
- Foreset bedding / Lam.
- Ripple - marked
- Scour (erosional) surface

GRAIN SIZE:
f - fine
med - medium
crs - coarse
crs - very coarse
pbl - pebble
congl - conglomerate
aggl - agglomerate
bc - breccia

METAMORPHIC:
clv - cleavage
styl - stylolite

MINERALIZATION:
Pyr - Pyrite
Pyrro - Pyrrhotite
Fl - Fluorite
Chl - Chlorite
Gnt - Garnet
Mgn - Magnetite
Glc - Glauconite
Lmn - Limonite

QUANTITATIVE LOG
present
common
abundant
very abundant
saturated

ATTITUDE:
(a) (b)
70° / (20°)
(a) Dip in relation to vertical (long) axis of core
(b) Dip in relation to horizontal plane.

CARBONATE ROCKS:
Classification (After R.J. Dunham)

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e.g. (i) colonial coral reef
(ii) algal mats (undulatory - lat.) and digitate (vert.) algal colonies - stromatolites

ENV. Depositional environment

COMALCO LIMITED

OTHER DATA:

5 cm

Drg No TAS-76-54 H

Scale 1:200

Logged by P.W. Stainton

18-1505 22154

PROJECT/FIELD SECTION: MOINA
BOREHOLE / FIELD LOC'N: SMD 14
R.L. (Csg Coll): ..... T.D. 196·17m
DRILL/FIELD LOG DATE: From ..... To 76/54 I

DEPTH(m) or SAMPLE No

GRAPHIC LOG (incl particles, mud, silt, sand, etc.)

COLOUR

SEDIMENTARY & BIOG. STRUCT. (Lam, Mtd, Vrb)

BEDDING ATTITUDE

METAMORPHIC STRUCTURES (Cleavage, Veins, etc.)

MINERALIZATION & ALTERATION

DIAGENETIC & METASOMATIC

METASOMATIC

Pyr

Pyrro

LITHO LOG

REMARKS

STRAT

Mbr.

Fm.

brn

brn

lt gy

dk gy lam

lt gy - dk gy lam

lt gy

lt gy

lt gy - dk gy lam

lt gy - dk gy mtld

lt gy - dk gy mtld

mtld

lt gy - dk gy

mtld

lt gy

lt gy

lt gy

lt gy - dk gy lam

lt gy

dk gy - dk brn

lt gy

lt gy

lt gy

lt gy

lt gy

lt gy mtld

dk gy

lt gy / wh mtld

gy - gn

gy - dk gy

mtld

mtld

mtld

lt / dk gy mtld

gy - wh

lt / dk gy dpld

gy - wh

lt / dk gy

wh - lt gy dpld

lt / dk gy

lt gy

lt gy

lt / dk gy dpld

lt / dk gy dpld

mtld

mtld

dpld

dk gy / brn

dpld / mtld

lt / dk gy

wh

lt / dk gy

Sand

Lime Mud: dol, lam bndd by algal lyrs & cont granular ppt, w fn alg lam. (N.B. dol prob. diagenetic)

Shell debris (brach?)

Lime Mud: gy - slt sized, containing carb. mtl (in fn micro lam.

N.B. Pyrite aggregate common along carbonaceous & dol laminate

N.B. All bedding features obliterated by stylolitization

bc clasts

Lime Mud: dol, mtld, brws cont calcite

Lime Mud: dol, brws cont more calc. than dol

Lime Mud: dol

Lime Mud: cont ' birdseye' (brws filled w sparry, crs calcite)

Dol: dk gy - dk brn, carb

Lime Mud: (alg)

bc clasts

Lime Mud: (alg) dol

Lime Mud: (alg)

Lime Mud: (alg.) dol

Lime Mud: lam, slt

bc clasts

bc clasts

Lime Mud: slt, lam

bc clasts

bc clasts

Lime Mud: slt, lam

Dol: dk gy - brn, carb, pyr

Lime Mud: slt, dk / lt gy lam

Wollastonite

T.D. 194·17m

GORDON LIMESTONE FORMATION

Seds churned by brws, with sparry calc

some filled

brws filled w crs sparry calc

025° (28°)

025° (25°)

wlist

LEGEND

PARTICLES:
- Microbioclastic hash
- Mud lump
- Shell fragments (general)
- Pelletoid
- Tentaculites
- Spicules
- Fossil - gasteropod (broken)
- Fossil - pelecypod (broken)
- Fossil - brachiopod
- Fossil - orthopod
- Algal stromatolite (SH.) (undulatory - laterol)
- Algal stromatolite (digitate)
- Fossil, coral
- Recrystallized (Rex)

COLOUR:
dk - dark; lt - light; gn - green; r - red; gy - grey; brn - brown; blk - black; wh - white; bf - buff; dpld - dappled

SEDIMENTARY STRUCTURES:
- Burrows (Brw)
- Bedded (Bd)
- Laminated (Lam)
- Convoluted (Cnvtd)
- 'flame' structure
- Foreset bedding / Lam.
- Ripple - marked
- Scour (erosional) surface

GRAIN SIZE:
f - fine; med - medium; crs - coarse; crs - very coarse; pbl - pebble; cong - conglomerate; ogglo - agglomerate; bc - breccia

METAMORPHIC:
clv - cleavage; stylolite

MINERALIZATION:
Pyr - Pyrite; Pyrro - Pyrrhotite; Fl - Fluorite; Chl - Chlorite; Grt - Garnet; Mgn - Magnetite; Glc - Glauconite; Lmn - Limonite

QUANTITATIVE LOG
present; common; abundant; very abundant; saturated

ATTITUDE:
(a) (b)
70°/ (20°)
(a) Dip in relation to vertical (long) axis of core.
(b) Dip in relation to horizontal plane.

CARBONATE ROCKS:
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ENV. Depositional environment

COMALCO LIMITED

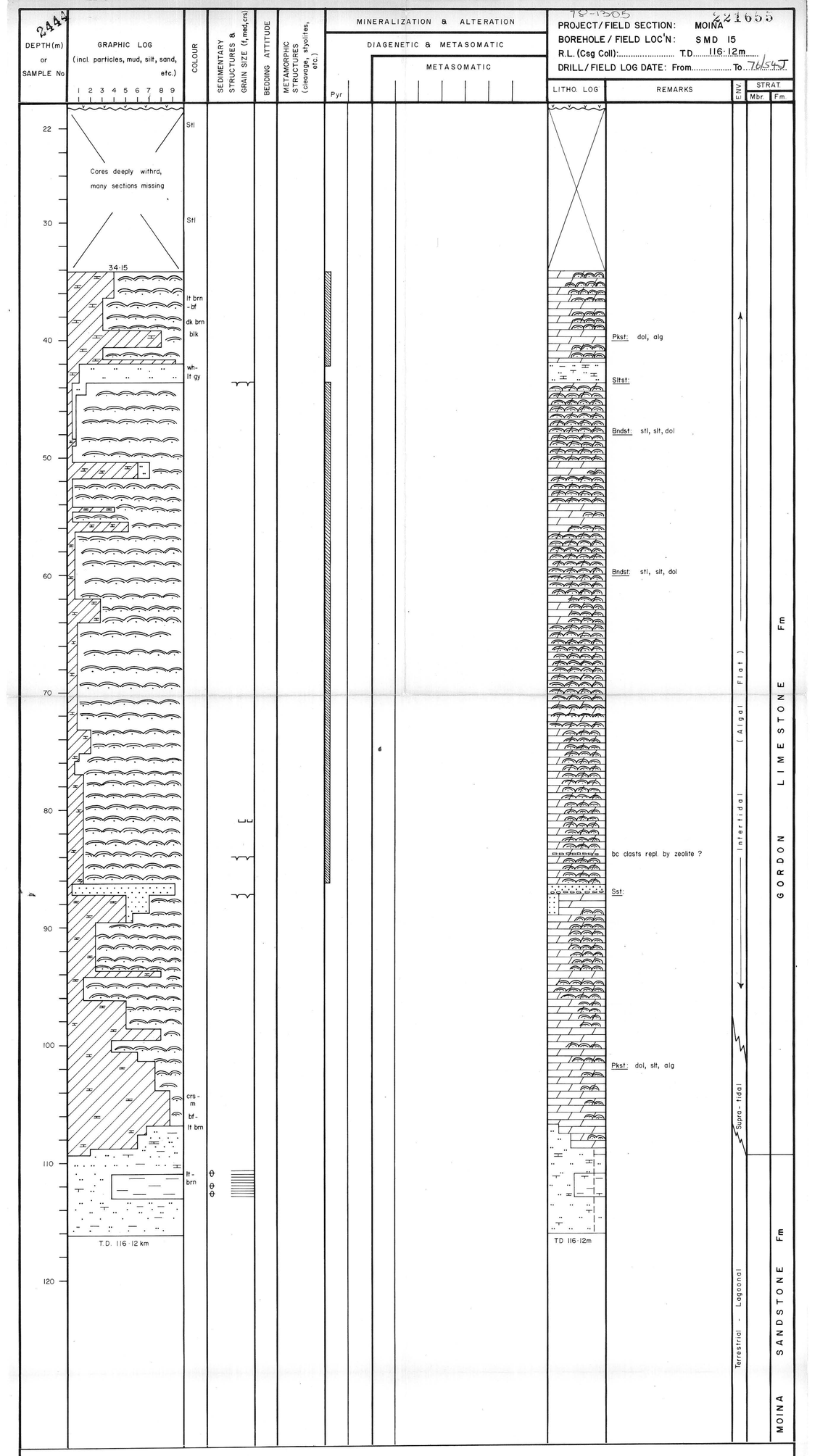
OTHER DATA

5 cm

Logged by P.W. Stainton

Scale 1:200

Drg No TAS-76-542



78-1305 221655

PROJECT/FIELD SECTION: MOINA
BOREHOLE/FIELD LOC'N: SMD 15
R.L. (Csg Coll): ..... T.D. 116·12m
DRILL/FIELD LOG DATE: From ..... To 76/54J

LEGEND

PARTICLES:
- Microbioclastic hash
- Mud lump
- Shell fragments (general)
- Pelletoid
- 'birdseye'
- Spicules
- Fossil - gasteropod (broken)
- Fossil - pelecypod (broken)
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ATTITUDE:
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ENV. Depositional environment

COMALCO LIMITED

OTHER DATA:

5 cm

Scale 1:200

Logged by P.W. Stainton

Drg N° TAS-76-54-J

12447

DEPTH(m) or SAMPLE No | GRAPHIC LOG (incl. particles, mud, silt, sand, etc.) 1 2 3 4 5 6 7 8 9 | COLOUR | SEDIMENTARY STRUCTURES & GRAIN SIZE (f, med, crs) | BEDDING ATTITUDE | METAMORPHIC STRUCTURES (cleavage, styolites, etc.) | MINERALIZATION & ALTERATION | DIAGENETIC & METASOMATIC | METASOMATIC | Fyr | Pyrro

TS-1305

221656

PROJECT/FIELD SECTION: MOINA

BOREHOLE / FIELD LOC'N: SMD - 17

R.L. (Csg Coll): ..... T.D. 74·12 m

DRILL/FIELD LOG DATE: From ..... To 76/54K

LITHO. LOG	REMARKS	ENV.	STRAT. Mbr.	STRAT. Fm.
a	Lime Mud: lam, lt/dk gy (dol)	sub-tidal		GORDON LIMESTONE Fm
p	Dol: m grn lt gy - bf, sucr, si, Pkst			
b	Bndst: st, dol			
P	Dol: crs, lt brn rsns, sucr. (prim), bf Pkst.	Supra - tidal — Intertidal		
	Bndst: stl, dol			
P	Dol: m- crs, lt brn - bf, sucr, Pkst			
	Sh: slt, carb ? intbd w sltst/sst f, gy- brn	? Lagoonal ?		
	Sst: f, arg			
	Sst: gy - lt gy, m - f	Terrestrial		MOINA SANDSTONE Fm

20 30 40 50 60 70 80

lt/dk gy; lt/gy - bf; lt/dk gy; bf; dk brn - blk; lt brn - bf; dk gy - brn

m - crs

T.D. 74·12 m

LEGEND

PARTICLES:
- Microbioclastic hash
- Mud lump
- Shell fragments (general)
- Pelletoid
- 'birdseye'
- Spicules
- Fossil - gasteropod (broken)
- Fossil - pelecypod (broken)
- Fossil - brachiopod
- Fossil - arthropod
- Algal stromatolite (Stl.) (undulatory - lateral)
- Algal stromatolite (digitate)
- Fossil, coral
- xx Recrystallized (Rex)

COLOUR:
dk - dark; lt - light; gn - green; r - red; gy - grey; brn - brown; blk - black; wh - white; bf - buff; dpld - dappled

SEDIMENTARY STRUCTURES:
- Burrows (Brw)
- Bedded (Bd)
- Laminated (Lam)
- Convoluted (Cnvtd)
- 'flame' structure
- Foreset bedding / Lam.
- Ripple - marked
- Scour (erosional) surface

GRAIN SIZE:
f - fine; med - medium; crs - coarse; crs - very coarse; pbl - pebble; congl - conglomerate; aggl - agglomerate; bc - breccia

METAMORPHIC:
clv - cleavage; stylolite

MINERALIZATION:
Pyr - Pyrite; Pyrro - Pyrrhotite; Fl - Fluorite; Chl - Chlorite; Gnt - Garnet; Mgn - Magnetite; Glc - Glauconite; Lmn - Limonite

QUANTITATIVE LOG
present; common; abundant; very abundant; saturated

ATTITUDE:
(a) (b)
70° / (20°)
(a) Dip in relation to vertical (long) axis of core
(b) Dip in relation to horizontal plane.

CARBONATE ROCKS:
Classification (After R.J. Dunham)
- a - Lime Mudstone (Mdst): Muddy carbonate, <10% grains
- w - Wackestone (Wkst): Mud-supported carbonate, >10% grain bulk
- p - Packstone (Pkst): Grain-supported, muddy carbonate
- g - Grainstone (Grnst.): Grain-supported, mud free, carbonate rocks
- b - Boundstone (Bndst): Carbonates showing interconnecting (binding). Skeletal or organic remnants.
  e.g. (i) colonial coral reef
  (ii) algal mats (undulatory - lat.) and digitate (vert.) algal colonies - stromatolites

ENV. Depositional environment

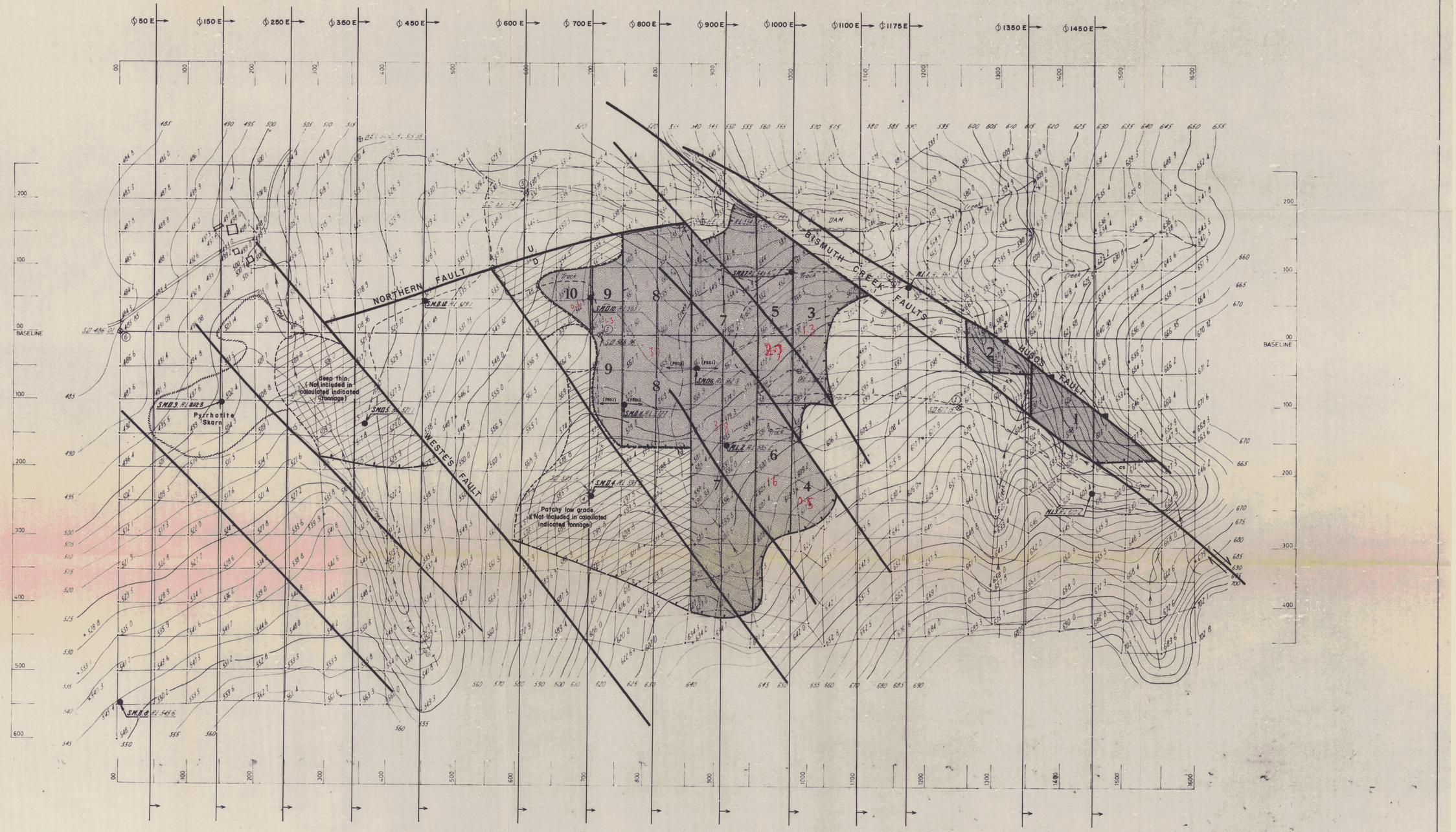
OTHER DATA:

5 cm

Scale 1:200

Logged by P. W. Stainton

TS-76-54 K



LEGEND

- Section Lines
- Limits of block
- 8 — Indicated or Inferred Wrigglite, Block Number
- Surface or Pre Basalt surface edge of Wrigglite, with dip
- Grade boundary
- Faults, most Inferred
- Drill holes
- Inferred boundary

5 cm

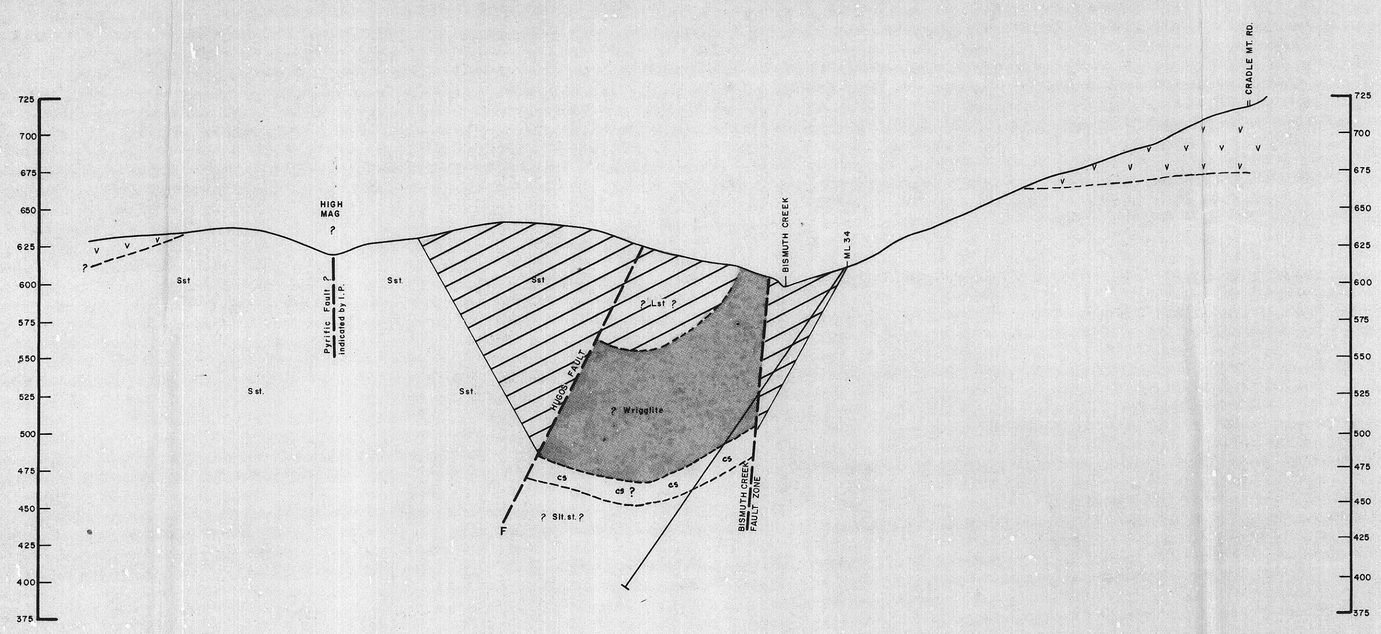
NOTE: All R.L.s at grid points are ground level, except Base line where levels are on 100mmx 50mm pegs up to 1250. then on top of 25mmx 25mm stakes to 1600.

Datum for Levels: H.E.C. Bench marks 3731 & 3732

COMALCO LIMITED 76P-1305		
EL.7/74 MOINA TASMANIA		
WRIGGLITE RESOURCE ESTIMATION PLAN (PRELIMINARY) 221657		
Compiled: P. Askins.	Revised: Oct. 1976	Drawn: M. Attwell
Date: Sept., 1976.	Scale 1:2500	Drg. N° TAS-76-57

INDICATED Nil
INFERRED (Because of lack of understanding of geology here)
Area of Section 17x25x25 ≈ 1,0600m²
BLOCK 1 ≈ 175m across (NW-SE)
Volume 175x 10600 = 1,855,000m³
Mass ≈ 1,800,000 x 3·3 ≈ 5·9 million tonnes

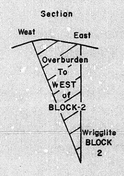
OVERBUREN
Area of Section 26x25x25 ≈ 16,250m²
BLOCK 1 Assume south contact dips parallel to open pit
Assume north west contact incorporated into block 2
Volume ≈ 175x 16,000² = 2,800,000m³
Mass ≈ 2,800,000 x 2·5 ≈ 7·0 million tonnes

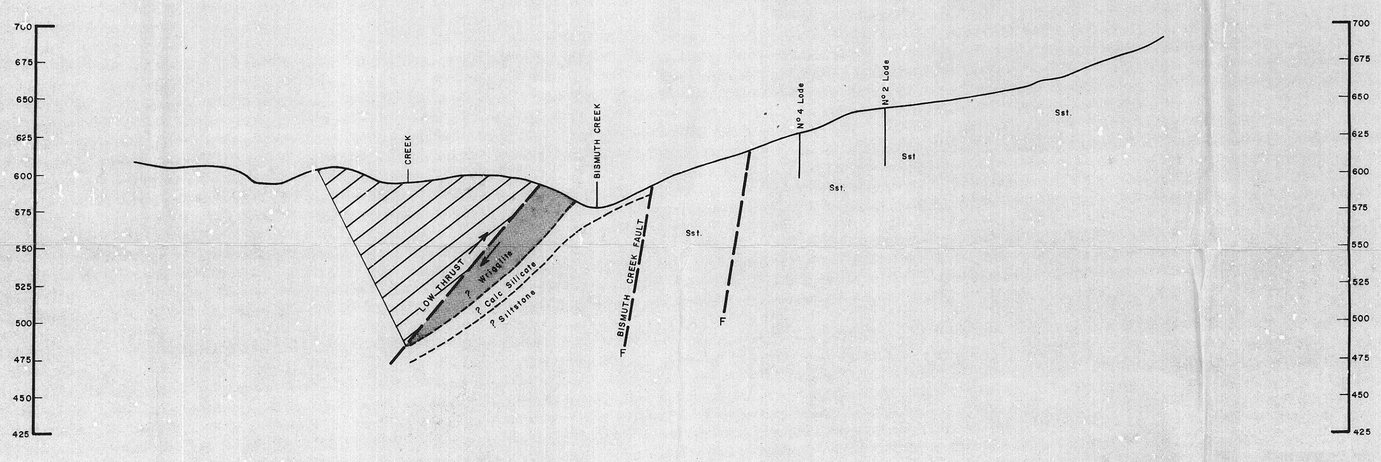


CROSS SECTION 1450 E

INDICATED Nil
INFERRED
Area of Section 14x12·5x 12·5 ≈ 2,000m²
BLOCK 2 ≈ 100m across
Volume 100x 2,000 = 200,000m²
Mass 200,000 x 3·3 = 660,000 tonnes

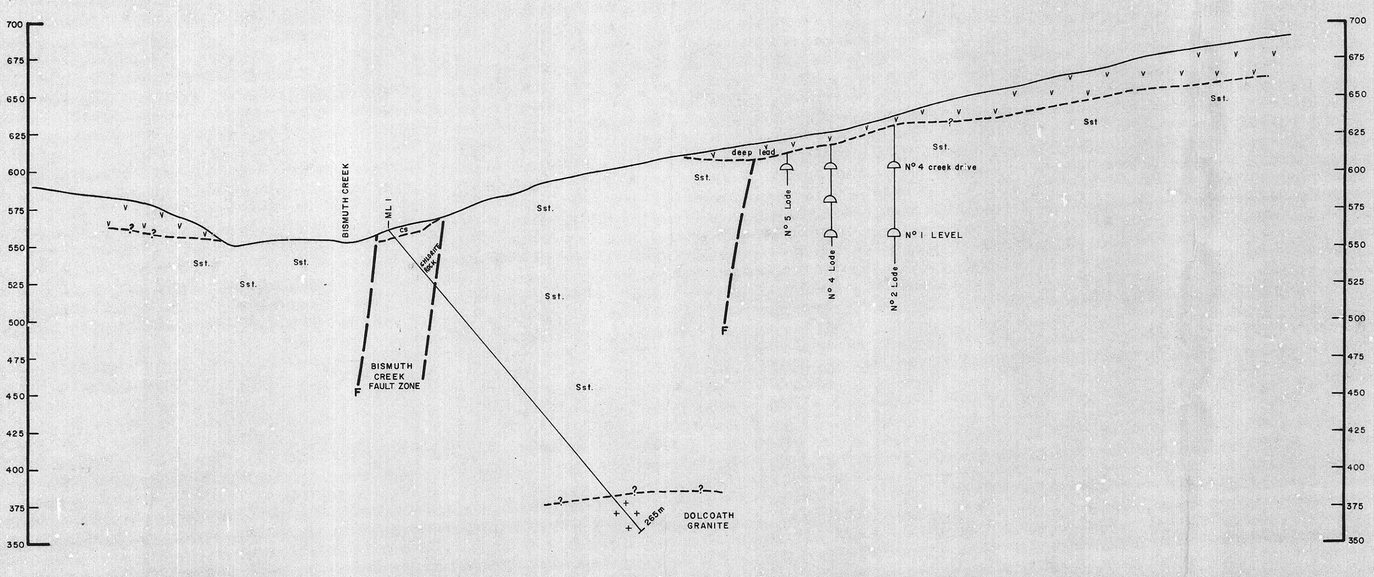
OVERBURDEN
BLOCK 2 Assume S.E. edge incorporatnd into block 1
N.W. edge has additional batter.
Area of Section = ½ x 130x 130 = 8450
Volume = 8450 x 100 = 845,000m³ plus batter N.W. edge
see W – E. section.
Approx Volume ½ x 120x 70x 100 = 420,000 m²
Total Volume = 1,265,000 m³
Mass = 1·265 x 2·5 ≈ 3·2 million tonnes





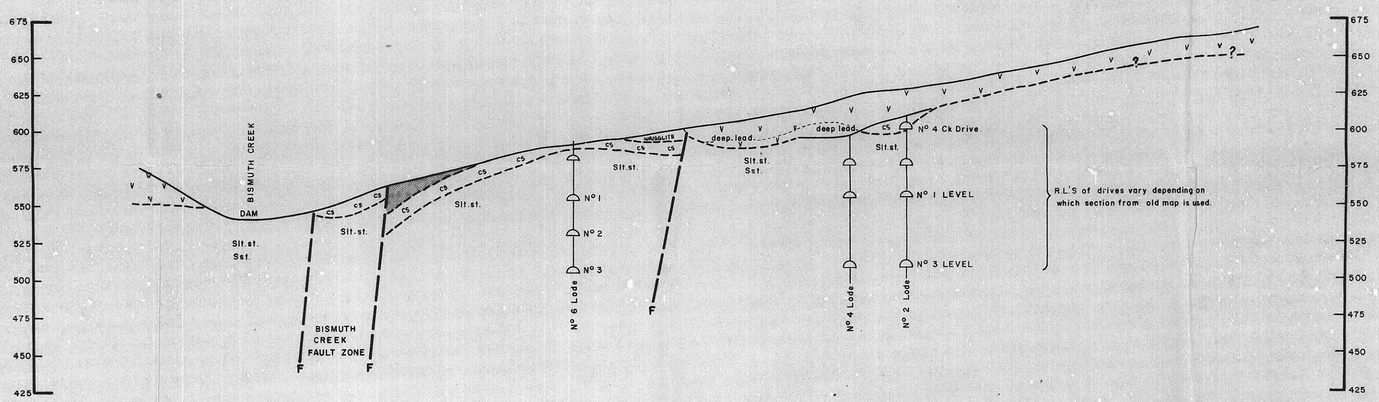
CROSS SECTION 1350 E

INFERRED Nil
INDICATED Nil



CROSS SECTION 1175 E

INFERRED Negligible
INDICATED Negligible



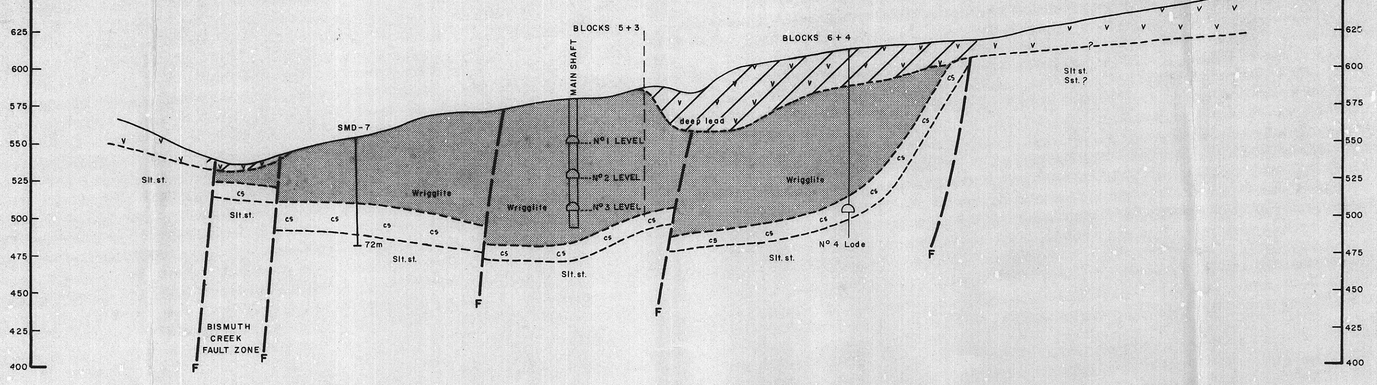
CROSS SECTION 1100 E

INDICATED
BLOCK 3
Area of Section = 16250m²
Because of dipping beds, half this area is assumed representative of whole block
Average width of block is close to 50m
Volume = 16250 x ½ x 50 = 406,250 m²
Mass 406,250 x 3·3 ≈ 1,340,500 tonnes
OVERBURDEN NIL

INDICATED
BLOCK 4
Area of Section = 9700m²
Because of irregular east edge of block, half this area is assumed representative of whole block. Average width of block about 30m.
Volume = 9700 x ½ x 30 = 145,500m³
Mass = 145,500 x 3·3 ≈ 480,000 tonnes
OVERBURDEN
Batter also along eastern edge, length 175m, depth to wrigglite average 30m, with 60° slope
Volume 20 x 30 x ½ x 175 = 52,500m³ plus area of section x average width of block
ie. 4840 x 30 = 145,200m³
TOTAL VOLUME = 197,700m³
Mass = 197,700 x 2·5 = 494,250 tonnes



INDICATED
BLOCK 5
Area of Section 19 x 25 x 25+28 x 12·5 x 12·5 = 11875+4375
= 16250 m²
This area assumed representative of whole block
Volume = 16250 x 50 = 812,500m³
Mass = 812,500 x 3·3 = 2,681,000 tonnes
OVERBURDEN NIL



CROSS SECTION 1000E

INFERRED Negligible
INDICATED Negligible

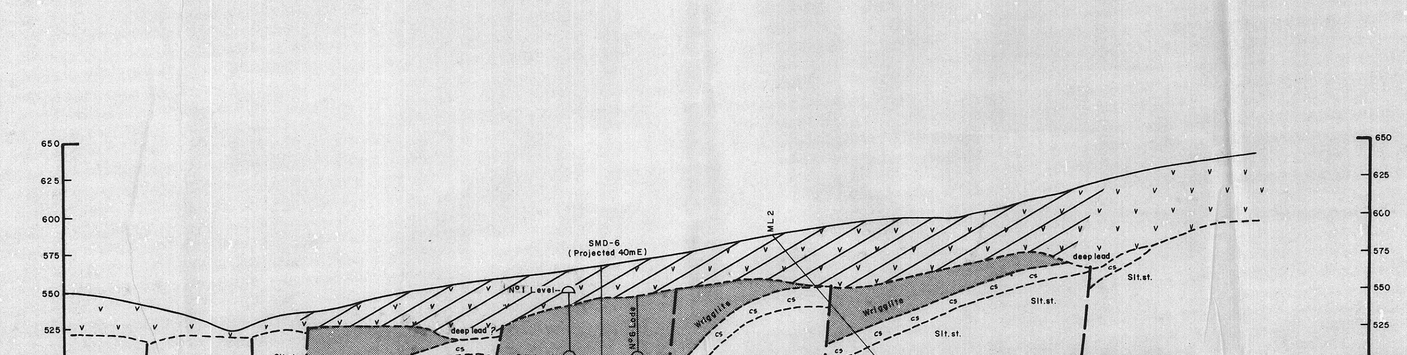
BLOCK 6
Area of Section 12 x 25 x 25+ 14 x 12·5 x 12·5 = 7500+2188 = 9688m² say 9700m²
This area assumed representative of whole block
Volume = 9700 x 50 = 485,000m³
Mass = 485000 x 3·3 = 1600,500 tonnes
OVERBURDEN
Area of Section 31 x 12·5 x 12·5 ≈ 4840m²
Volume 4840 x 50 = 242,000 m³
Mass 242,000 x 2·5 = 1,210,000 tonnes

INDICATED
BLOCK 7
Area of Section 54 x 12·5 x 12·5 + 5 x 25 x 25 ≈ 8440+3125
= 11,565 m²
Volume 11565 x 100 = 1,156500m³
Mass 1,156,00 x 3·3 ≈ 3,816,000 tonnes
OVERBURDEN
Area of Section 99 x 12·5 x 12·5 ≈ 15,470m²
Volume 15,470 x 100 = 1,547,000m²
Plus batter on western side, south end of block, length 250m
average depth 35m.
Volume 250 x 20 x ½ x 35 = 87500 m³
plus northern edge, 12·5 x ½ x 9 x 125 = 7030m³
plus southern edge 55 x ½ x 35 x 100 = 96205m³
TOTAL VOLUME 1,737,500m³
Mass 173 500 x 2·5 = 4,343,750 tonnes
say 4,344,000 tonnes



CROSS SECTION 900 E

INDICATED
BLOCK 8
Area of Section 45 x 12·5 x 12·5 + 7 x 25 x 25 = 7030+4375
= 11405 m²
Volume 11405 x 100 = 1,140,500 m³
Mass 1140 500 x 3·3 = 3,764,000 tonnes
OVERBURDEN
Area of Section 13 x 25 x 25 + 22 x 12·5 x 12·5 = 8125+3438
= 11563 m²
Volume 11563 x 100 = 1,156,300 m²
plus northern edge 15 x ½ x 10 x 100 = 7500 m³
plus southern edge 50 x ½ x 25 x 100 = 62500 m³
TOTAL VOLUME 1,226,300 m³
Mass 1,226,300 x 2·5 = 3,065,750 tonnes
say 3,066,000 tonnes



CROSS SECTION 800 E

SUMMARY

INDICATED WRIGGLITE TOTAL 15,501,000 tonnes
INFERRED WRIGGLITE TOTAL: 6,560,000 tonnes
TOTAL OVERBURDEN OVER INDICATED + INFERRED WRIGGLITE TOTAL: 21,443,000 tonnes
TOTAL OVERBURDEN OVER INDICATED WRIGGLITE TOTAL: 11,243,000 tonnes.
INDICATED WRIGGLITE / OVERBURDEN, RATIO = 1·38 : 1

L E G E N D

Basalt
Calc silicate rock
Indicated or Inferred Wrigglite Assumed ρ = 3·3
Overburden Assumed ρ = 2·5 basalt, limestone, sandstone Assumed 60° batter slope.

NOTE: For Block Areas see seperate plan.

221658

5 cm

COMALCO LIMITED 78-1305

E.L 7/74 "MOINA" TASMANIA
SHEPHERD & MURPHY MINE AREA
GEOLOGICAL SECTIONS
PRELIMINARY WRIGGLITE RESOURCE CALCULATIONS
SHEET-A

Compiled P.W. Askins	Revised	Drawn M. Attwell
Date October, 1976	Scale 1:2,500	Drg Nº TAS-76-56

400N 300N 200N 100N 00 100S 200S 300S 400S 500S 600S

INDICATED

BLOCK - 9

Area of Section 49 x 12·5 x 12·5 = 7656 m²

Volume 7656 x 50 : 382,800 m³

Mass 382800 x 3·3 ≈ 1,263,000 tonnes

BLOCK - 10

Area of Section 24 x 12·5 x 12·5 = 3750 m²

Average width about 45 m

Volume 45 x 3750 = 168750 m²

Mass 168750 x 3·3 ≈ 557,000 tonnes

OVERBURDEN

BLOCK - 9

Area of Section 42 x 12·5 x 12·5 + 4 x 25 x 25 = 6562 + 2500

= 9062 m²

Volume 9062 x 50 = 453100 m³

plus northern batter 50 x 30 x ½ x 50 = 37500 m²

plus southern and western batter length 175 m

Average depth 35 m

Volume = 175 x 20 x ½ x 35 = 61,250 m³ (20 / 35)

Total Volume 551850 m³

Mass 551850 x 2·5 = 1,379,625 say 1,380,000 tonnes

BLOCK - 10

Area of section 3 x 25 x 23 + 17 x 12·5 = 1875 + 2656 = 4530 m²

Volume = 45 x 4530 = 203850 m²

plus edge batter all sides except east, length 170 m or depth 40 m

Volume = 170 x 45 x ½ x 25 = 95625 m³ (25 / 45)

Total Volume 299475 m³

Mass 299475 x 2·5 = 748687 say 749,000 tonnes

L E G E N D

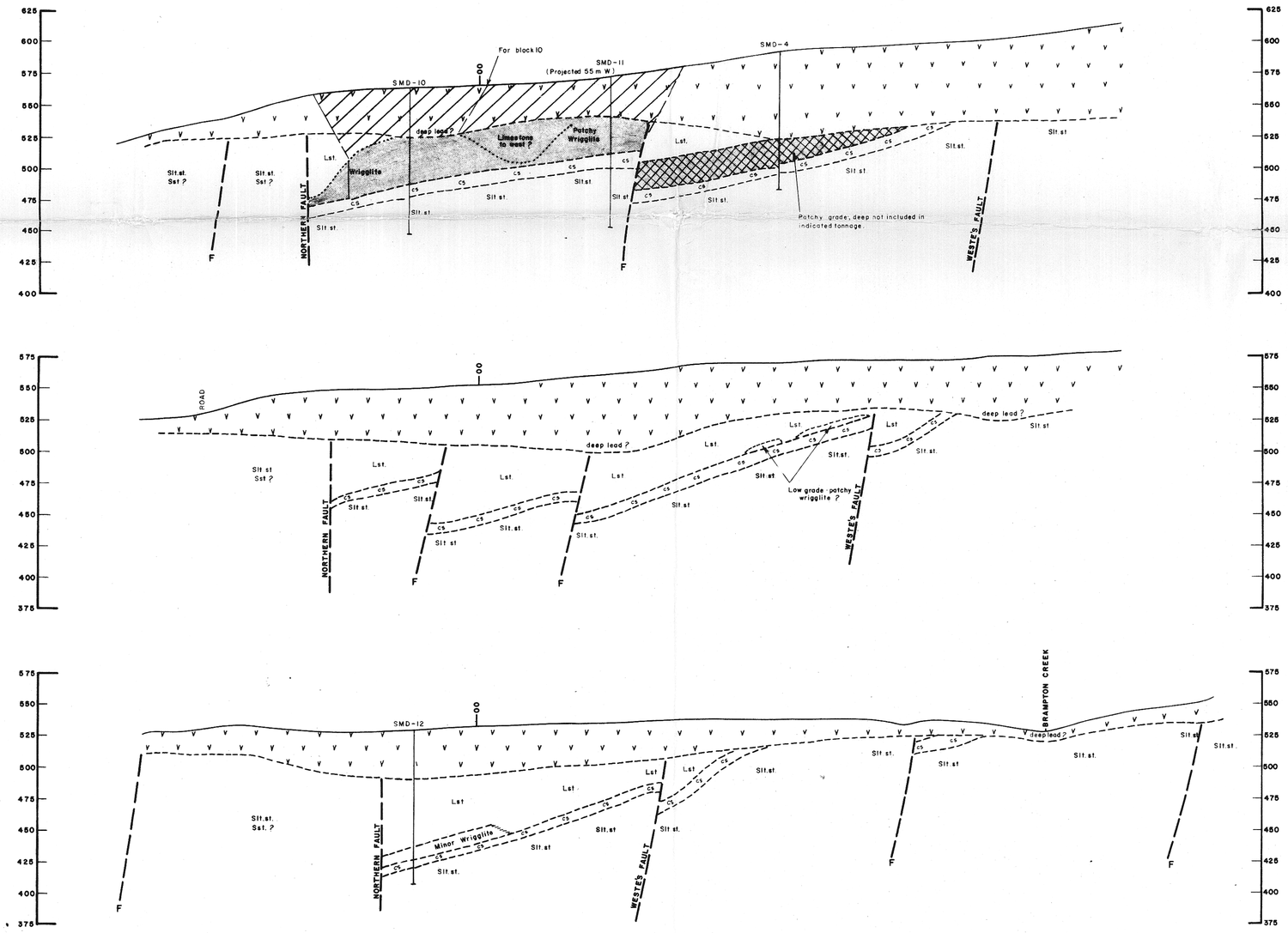
Indicated or inferred Wrigglite Asumed P = 3·3

Overburden Asumed P = 2·5 — *Limestone, Basalt, Sandstone*

Asumed 60° batter slope

Basalt

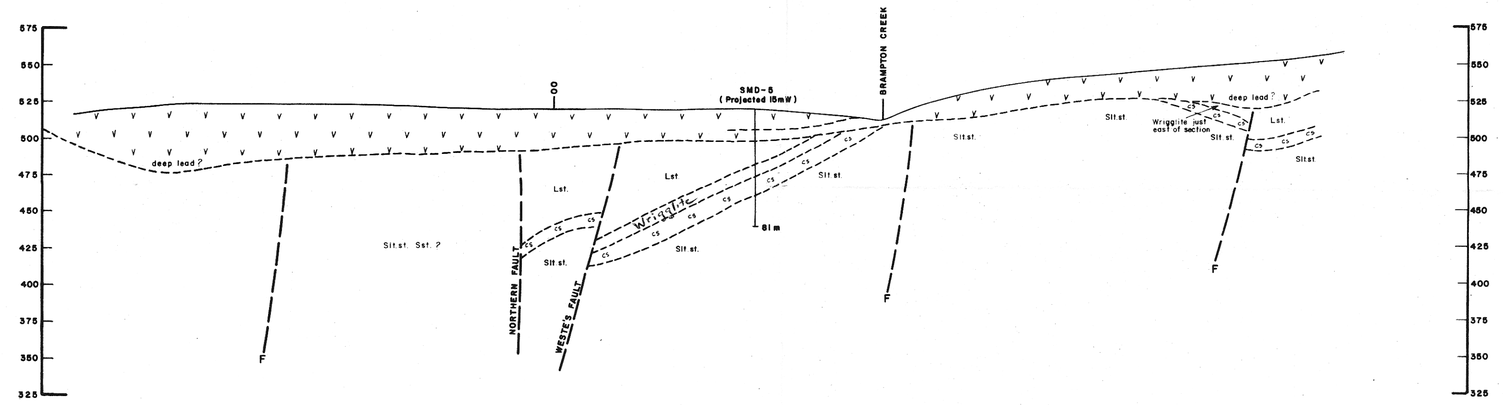
cs cs *Calc Silicate rock*



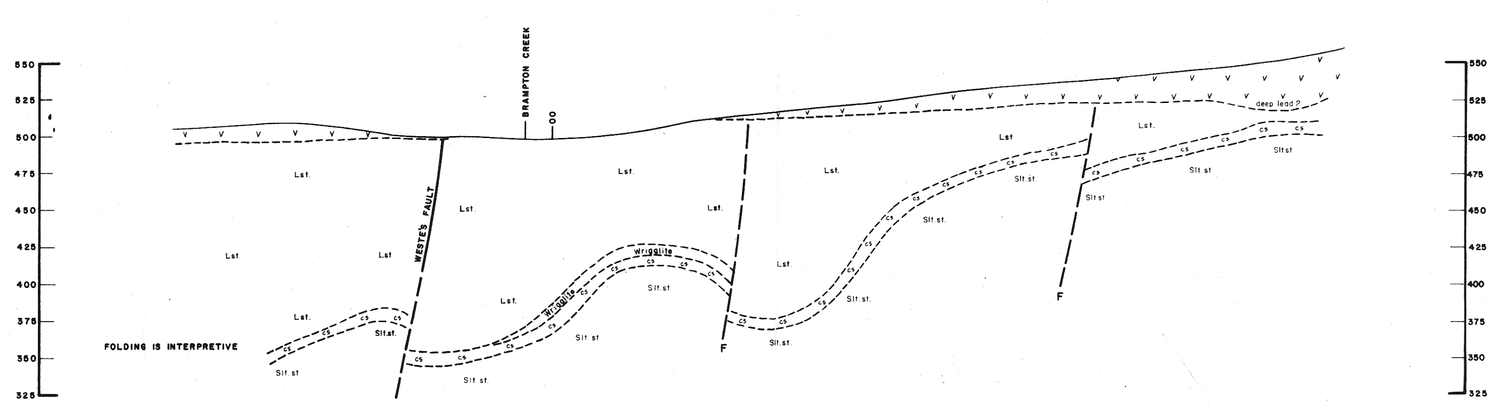
CROSS-SECTION 700 E

CROSS-SECTION 600E

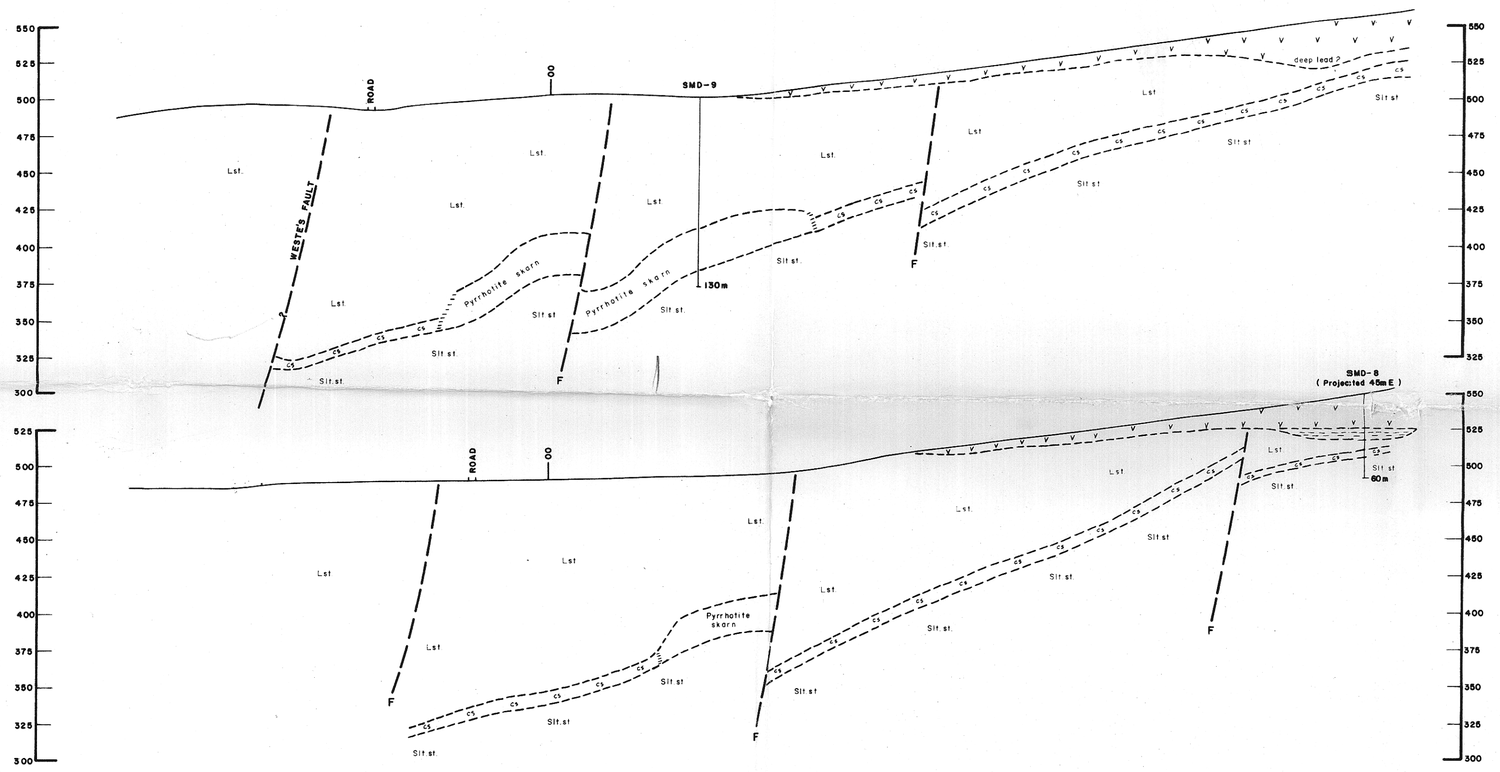
CROSS-SECTION 450 E



CROSS-SECTION 350 E



CROSS-SECTION 250 E



CROSS-SECTION 150 E

CROSS-SECTION 50 E

5 cm

221659 2450

COMALCO LIMITED 78-1305

E.L 7/74 "MOINA" TASMANIA

SHEPHERD & MURPHY MINE AREA

GEOLOGICAL SECTIONS

PRELIMINARY WRIGGLITE RESOURCE CALCULATIONS

SHEET - B

Compiled P.W. Askins.	Revised:	Drawn M. Attwell.
Date October, 1976	Scale 1:2,500	Drg. N° TAS-76-55

1600 mW | 1400 mW | 1200 mW | 1000 mW | 800 mW | 600 mW | 400 mW | 200 mW | 00

600 mN
400 mN
200 mN
00
200 mS
400 mS
600 mS
800 mS
1000 mS
1200 mS
1400 mS
1600 mS

Note. - Samples collected in C horizon by hand auger
- -80 mesh fraction analysed by Comalco Research Centre using AAS after hot HCl & HNO_3 digestion
- Key, ppm Cu | F / Pb | Bi / Zn | Mo
- ND indicates below limits of detection
  Cu 5 ppm
  Pb 10 ppm
  Zn 5 ppm
  Bi 20 ppm

Note Overlay to geology plan No. Tas / 76 / 47

5 cm

COMALCO LIMITED 78-1305

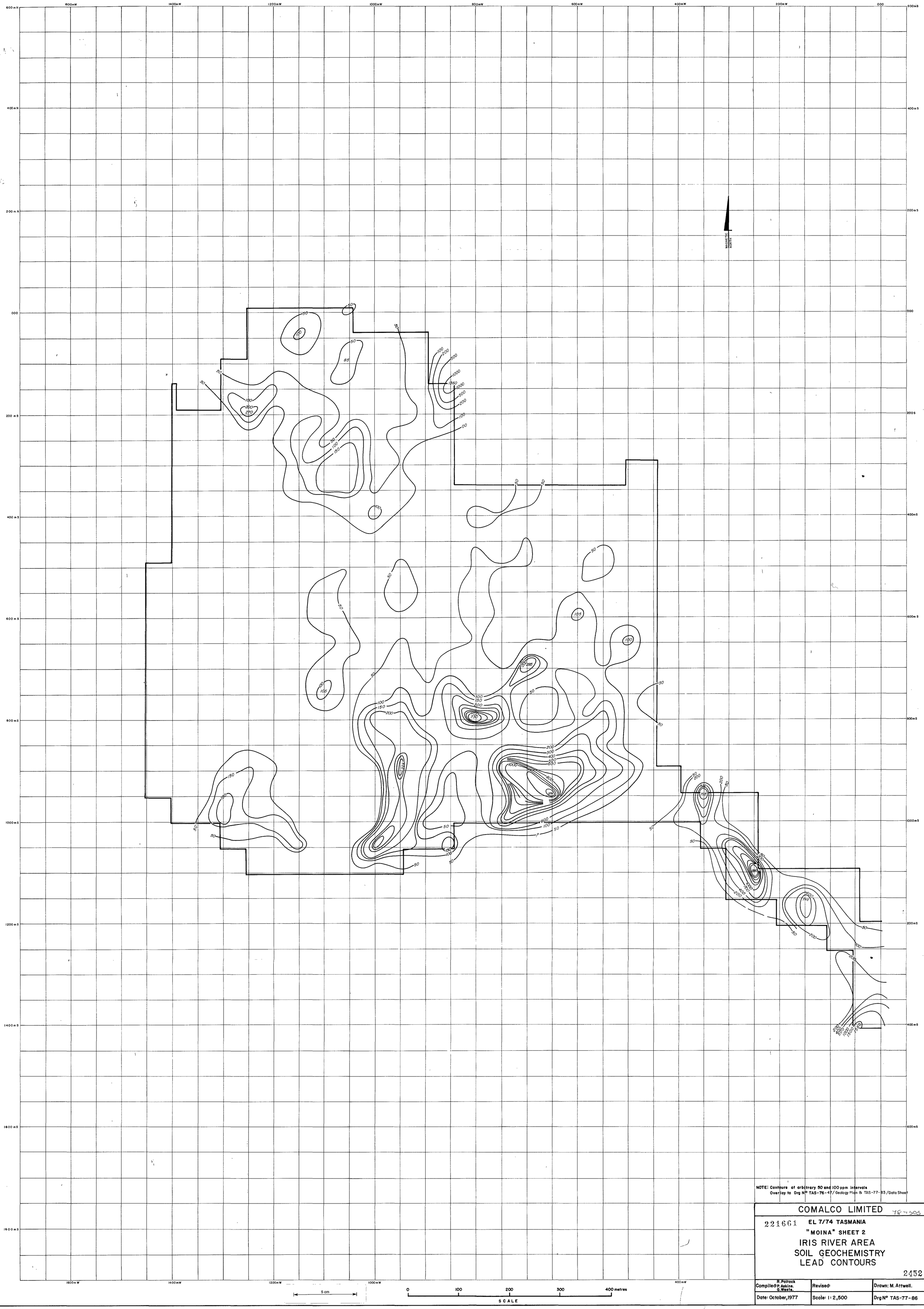
E.L. 7/74 TASMANIA

221660

"MOINA" SHEET 2
IRIS RIVER AREA
SOIL GEOCHEMISTRY
DATA SHEET

2451

Comp. R. Pollock, P.W. Askins, G. Weste	Revised:	Drawn: S. Hutchison
Date: October, 1977	Scale: 1:2500	Drg. No: TAS / 77 / 83



COMALCO LIMITED 78-1305

221661 EL 7/74 TASMANIA

"MOINA" SHEET 2

IRIS RIVER AREA
SOIL GEOCHEMISTRY
LEAD CONTOURS

2452

Compiled: R. Poltock, P. Askins, G. Weste.	Revised:	Drawn: M. Attwell.
Date: October, 1977	Scale: 1:2,500	Drg N° TAS-77-86

1600mW 1400mW 1200mW 1000mW 800mW 600mW 400mW 200mW 000

600mN 400mN 200mN 000 200mS 400mS 600mS 800mS 1000mS 1200mS 1400mS 1600mS 1800mS

MAGNETIC NORTH

5 cm

NOTE:
- Contours at arbitrary 50 and 100 ppm intervals

X 0·05 Indicates site of rock sample with Au analysis, ppm.
- This plan overlays Drg N° TAS-76-47/Geology Map & TAS-77-83/Data Sheet

SCALE 0 100 200 300 400 metres

COMALCO LIMITED KP-1385

221662 E.L. 7/74 TASMANIA

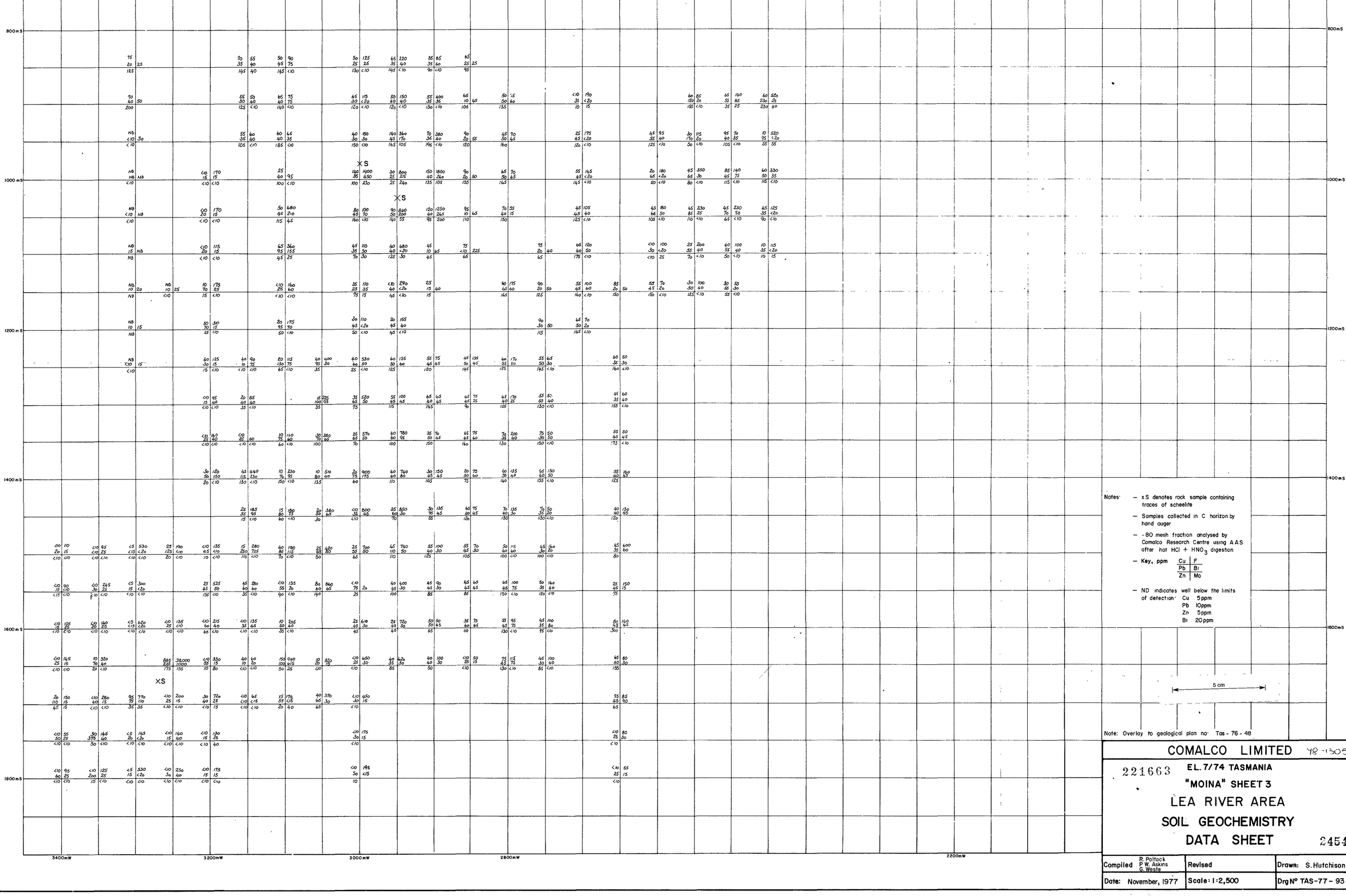
"MOINA" SHEET 2
IRIS RIVER AREA
SOIL GEOCHEMISTRY
BISMUTH CONTOURS
GOLD IN ROCK SAMPLES

2453

Compiled: R. Poltock, P. Askins, G. Weste.	Revised:	Drawn: M. Attwell
Date October, 1977	Scale: 1: 2,500	Drg N° TAS-77-89

3400mW 3200mW 3000mW 2800mW 2600mW 2400mW 2200mW 2000mW 1800mW

600mN 400mN 200mN 000 200mS 400mS 600mS 800mS 1000mS 1200mS 1400mS 1600mS 1800mS



Notes:
- xS denotes rock sample containing traces of scheelite
- Samples collected in C horizon by hand auger
- -80 mesh fraction analysed by Comalco Research Centre using AAS after hot HCl + HNO_3 digestion
- Key, ppm: Cu | F / Pb | Bi / Zn | Mo
- ND indicates well below the limits of detection: Cu 5ppm, Pb 10ppm, Zn 5ppm, Bi 20ppm

Note: Overlay to geological plan no: Tas - 76 - 48

COMALCO LIMITED 48-1305

221663

EL. 7/74 TASMANIA

"MOINA" SHEET 3

LEA RIVER AREA

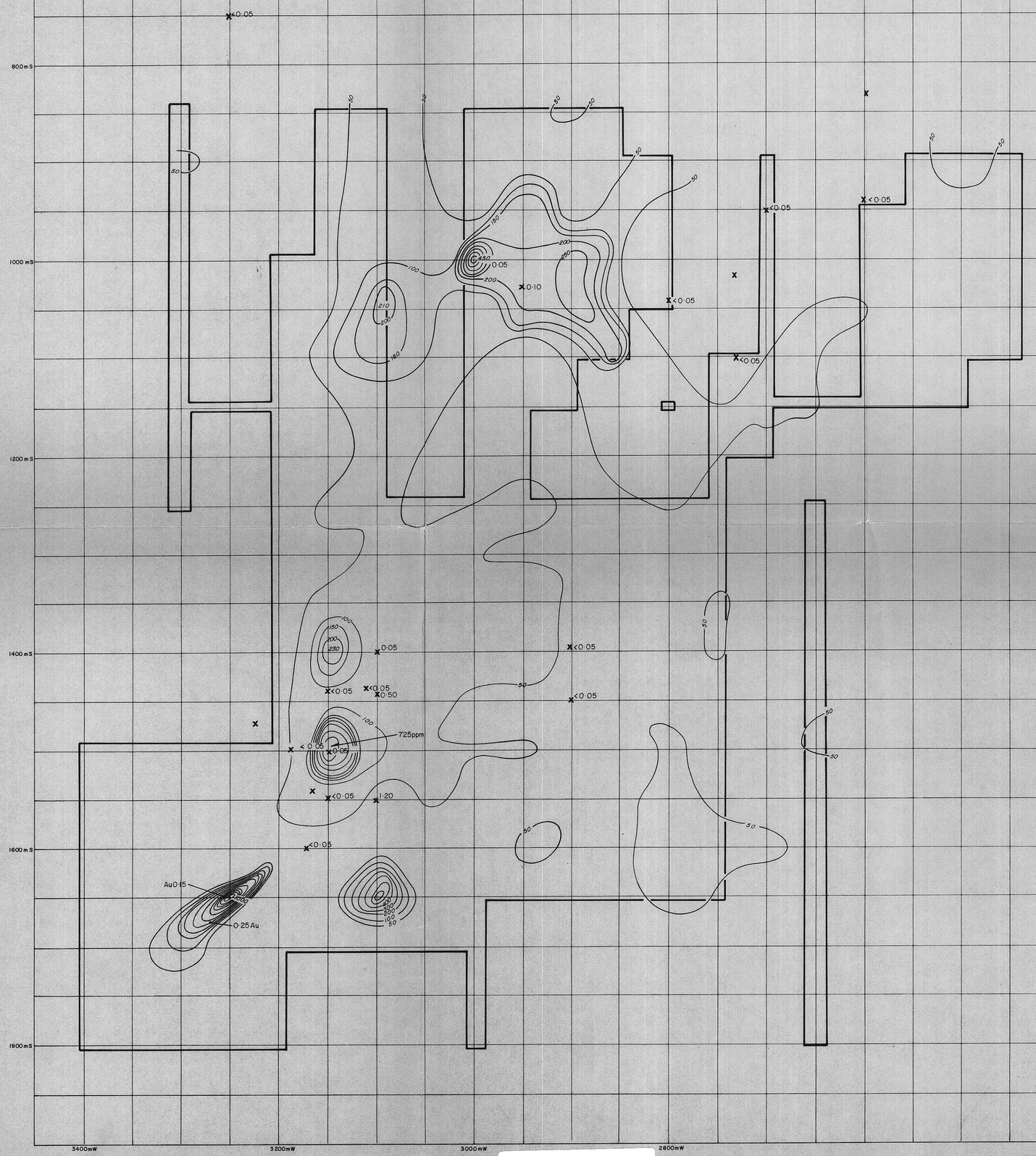
SOIL GEOCHEMISTRY

DATA SHEET 2454

Compiled: R. Pollock, P.W. Askins, G. Weste	Revised	Drawn: S. Hutchison
Date: November, 1977	Scale: 1:2,500	Drg N° TAS-77-93

3400mW 3200mW 3000mW 2800mW 2600mW 2400mW 2200mW 2000mW 1800mW

600mN 400mN 200mN 000 200mS 400mS 600mS 800mS 1000mS 1200mS 1400mS 1600mS 1800mS



NOTE:
- Contours at arbitrary 50 or 100ppm intervals
- 0·05 X - Indicates site of rock sample with Au analysis, ppm.
- This plan overlays Drg TAS-76-48/Geology Plan. & TAS-77-93/Data Sheet.

5 cm

0 100 200 300 400 metres
SCALE

COMALCO LIMITED 78-1305

221664 EL. 7/74 TASMANIA Vol 6/7

"MOINA" SHEET 3

LEA RIVER AREA
SOIL GEOCHEMISTRY
BISMUTH CONTOURS
GOLD IN ROCK SAMPLES

2455

Compiled R. Pollock, P. Askins, G. Weste	Revised	Drawn: M. Attwell
Date: October, 1977	Scale: 1:2,500	Drg N° TAS-77-92

3400mW 3200mW 3000mW 2800mW 2600mW 2400mW 2200mW 2000mW 1800mW

600mN 400mN 200mN 000 200mS 400mS 600mS 800mS 1000mS 1200mS 1400mS 1600mS 1800mS

MAGNETIC NORTH

NOTE: Contours at arbitrary 50ppm intervals
Overlay to Drg N° TAS-76-48 / Geology Plan & TAS-77-93 / Data Sheet

COMALCO LIMITED 78-1305

EL 7/74 TASMANIA Vol 6/7

221665 "MOINA" SHEET 3

LEA RIVER AREA
SOIL GEOCHEMISTRY
LEAD CONTOURS

2456

Compiled: R. Pollock. P. Askins. G. Weste.	Revised	Drawn M. Attwell
Date October, 1977	Scale: 1: 2,500	Drg. N° TAS-77-85

5 cm

0 100 200 300 400 metres
SCALE

3400mW 3200mW 3000mW 2800mW 2600mW 2400mW 2200mW 2000mW 1800mW

600 mN
400 mN
200 mN
000
200 mS
400mS
600mS
800 mS
1000mS
1200mS
1400mS
1600mS
1800mS

MAGNETIC NORTH

MAGNETIC NORTH

50 25 75 100 125

NOTE: Contours at arbitrary 25ppm. intervals.
Overlay to plan Drg N° TAS-76-48/Geology Plan & TAS-77-93/Data Sheet

COMALCO LIMITED 78-1305

221666 EL 7/74 TASMANIA

"MOINA" SHEET 3

LEA RIVER AREA
SOIL GEOCHEMISTRY
COPPER CONTOURS 2457

Compiled: R Poltock, G Weste, P W Askins	Revised	Drawn M Attwell
Date October, 1977	Scale 1 2,500	Drg N° TAS-77-84

5 cm

0 100 200 300 400 metres
SCALE

3400mW 3200mW 3000mW 2800mW 2600mW 2400mW 2200mW 2000mW 1800mW

600mN 400mN 200mN 000 200mS 400mS 600mS 800mS 1000mS 1200mS 1400mS 1600mS 1800mS

100 100 100 100 200 200 100 100 100 100 100 100 100 100 200 300 300 200 100 100 100 100

3400mW 3200mW 3000mW 2800mW 2200mW

5 cm

NOTE:
- Contours at arbitrary 100ppm intervals
- Overlay to Drg N° TAS-76-48/Geology Plan & TAS-77-93 Data Sheet

COMALCO LIMITED 78-1305

221667 EL. 7/74 TASMANIA

"MOINA" SHEET 3

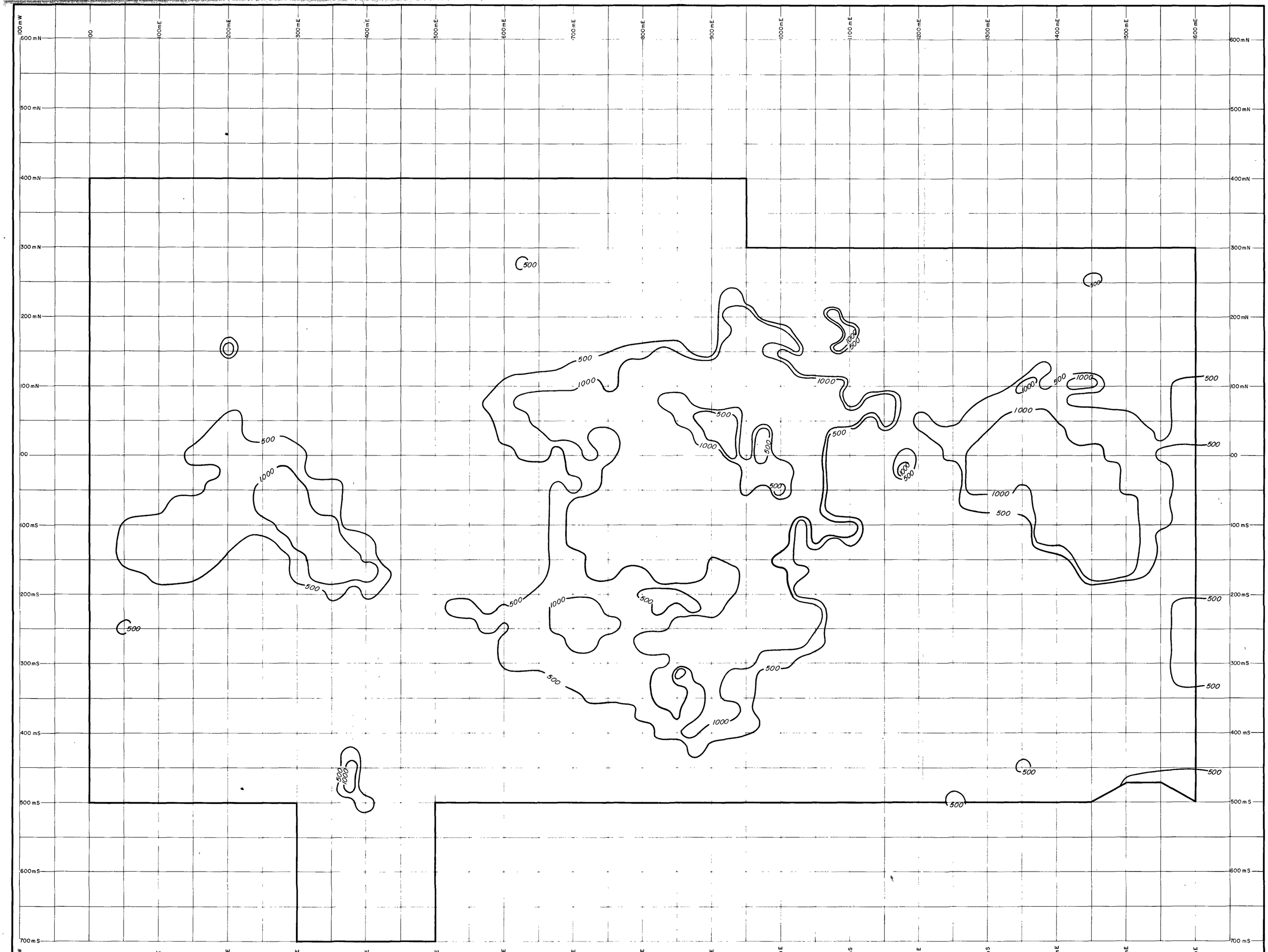
LEA RIVER AREA

SOIL GEOCHEMISTRY

ZINC CONTOURS

2438

Compiled R Pollock, P Askins, G Weste	Revised	Drawn: M. Attwell.
Date: October, 1977	Scale: 1:2,500	Drg N° TAS-77-94



SCINTREX MF-2 FLUXGATE MAGNETOMETER
Contours plus 500 and plus 1000 gammas

5 cm

S C A L E
0 100 200 300 400 metres

221668

COMALCO LIMITED 78-1305

E.L. 7/74 MOINA, TASMANIA
SHEPHERD AND MURPHY GRID 2459

MAGNETIC CONTOURS IN GAMMAS - 50 METRE GRID LINES

Compiled: G. Weste	Revised	Drawn: S Hutchison
Date: April & May, 1975	Scale: 1:2500	Drg. No. Tas/M/75/21

MAGNETIC NORTH

SURVEYED IN 1977

SURVEYED IN 1976

LIMITS OF GRIDDING

LIMITS OF GRIDDING

No readings taken along this line

LIMIT OF GRIDDING

NOTE: No Readings taken on lines 1625mE and 1675mE

NOTE: From 500S to 1550S readings taken only on 50m grid lines every 25m
ie: No readings on 25W, 25E, 75E, 125E, 175E

SCALE

NOTE
1. Contour interval 500 gammas
2. Instrument Scintrex MF 2 Fluxgate Magnetometer measures vertical component of field
3. Readings taken at 25m grid points (on and between 50m spaced lines) except where shown
4. Overlay to data plan TAS-75-19A

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221669 E.L. 7/74 TASMANIA 78-1505

"MOINA" - SHEET 1

SHEPHERD AND MURPHY AREA FLUXGATE MAGNETIC CONTOURS IN GAMMAS 2460

(Vertical component of field)

Compiled: G. Weste	Revised: P.W. Askins Sept'77	Drawn: M. Attwell
Date: September, 1976	Scale: 1:2500	Drg N° TAS-76-51