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MICROFILMED

EXPLORATION LICENCE 64/83

PROGRESS REPORT

FOR

YEAR 1984

ABIGNANO LIMITED

G.B. BRINK (B. Sc.)

JANUARY, 1985.

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Scale 1:2,000

1. Introduction

Abignano Limited hold title to an exploration licence on the west coast of Tasmania. The area is prospective for tungsten, tin, copper and gold.

Exploration to-date has been aimed mainly at locating and evaluating veins structures which carry wolframite mineralisation.

This report describes methods of evaluating a potential vein type deposit and includes an evolved computer model of a possible mining operation.

2. Tenements

Abignano Limited are the registered tenants of E.L. 64/83 and four enclosed mining leases M.L.'s 28M/76, 3M/78, 19M/78 and 35M/78. The E.L. covers an area of 44 sq. km. and was previously covered by E.L. 13/81 and E.L. 1/73.

3. Geology

The geology of the licence area has been described in previous reports and this year's activities have not caused any change in the geological interpretation of the subject area. However, diamond drilling, which was concluded in late December, 1984, will add to our geological knowledge, especially when laboratory results have been received and interpreted.

4. Feasibility Studies

Following a mineral liberation study on a bulk sample of the vein ore by Warman International Ltd. (refer Appendix 1), a preliminary conceptual model of a mining and processing operation was designed on the basis of three fault blocks as indicated in a previous report. The model was translated onto a computer spread sheet by C.P. Walker and this resulted in a favourable cash flow forecast.

This led to the design of an exhaustive mining and concentrating model which is capable of coping with a large number of variables, including such unpredictable as the unit price of WO_3 and labour costs.

Some extraordinary items, such as air transport of concentrates from Interview River to Smithton were specifically included and heavily costed, as the remoteness of the prospect has been, and still is, an almost forbidding element.

The resulting model was translated onto the Company's computer by T. Pitt and variables adjusted to realistic values, such values being derived in part from studies of reports on new mining methods and innovations (one such report is reproduced in Appendix 2).

Adopting the current wolframite price of A\$80 per metric tonne unit, a vein width of 0.3 meter, strike length of 1,500 meters and vein persistence to 50 meters depth, a mining and concentrating operation would be a profitable venture if the anticipated grade of 1.4% WO_3 persists.

5. Drilling

To verify the existance of ore resources along these parameters (which were derived from observations and interpretation) a drilling programme was devised. As a consequence a drilling rig was moved onto location and drilling ceased only just before Christmas.

5.1 Drilling Details

The hire of a Mindrill F20C rig, complete with down-the-hole gear, was negotiated with the Tasmanian Mines Department.

Initially transport of the rig from Burnie to the Interview River was to be by means of four wheel drive vehicles, assisted by a Bombardier tractor. However, it was found that the combined weight of the rig and gear was excessive, so a 12 tonne truck was hired together with a Cat. D6 dozer. Although expensive, this seemed to be the safest method of travel along the west coast beaches.

The rig was collected from Hobart on 20th November, 1984 and transported to the Interview River site on the 23rd and 24th (see photograph attached). Three drill sites were prepared at 200 meter intervals at the Kenny's Adit vein, i.e. the northern part of the vein system exposed to-date.

NQ. holes were drilled at 45° declination, at right angles to the vein strike and aimed at intersecting the veins between 40 and 50 meters vertically below the surface.

Only two holes were completed; serious breakdown of the rig prevented drilling of DDH3.

Both DDH 1 and DDH2 have been completed; field logs indicated that a significant zone of alteration and/or greisenisation was intersected below the old workings.

Geological logging of the lower portion of core from each hole confirmed the presence of an altered zone. Copies of the logs are attached as Appendix 3.

DDH1 notably intersected a significant thickness of clay and DDH2 a distinct zone of intense mineralisation with both wolframite and scheelite being present.

Relevant portions of core are being analysed for W, Sn, Cu, Mo and Au.

where core?

6. Expenditure

Expenditure during most of the year was associated with preliminary feasibility considerations and the design and development of a workable computer mine model.

Considerable expense has been incurred to-date on drilling and associated activities, which are continuing.

Details are as follows:-

Staff salaries	\$ 7,500
Consultants	\$ 3,000
Labour	\$ 6,000
Equipment	\$ 5,000
Mobilisation, site establishments	\$ 7,000
Drill hire	\$ 2,000
Camp facilities	\$ 1,700
Vehicles	\$ 2,800
Fuel, oil, parts	\$ 1,000
Travel, accommodation	\$ 1,700
Aircraft charter	\$ 1,200
Demobilisation	\$ 4,500
Office overheads	\$ 2,000
Sundry expenses	\$ <u>1,000</u>
<u>Total</u>	\$47,300

7. Recommendations

The recently completed drilling programme will be an important factor in determining the potential of the area. If the two holes in the Kenny's area confirm the grade and extent of mineralisation, it will be recommended that drilling continue along strike and at various depths. Geophysical methods, such as magnetics, also will be tested against the information obtained by drilling, if significant mineralisation is intersected.

{ Geochemistry has been tried and found unsuccessful,
and therefore no great emphasis will be given to
such methods. }

With increased and improved data from ongoing exploration the computer model will be updated, which should result in an up-to-date and continuous feasibility assessment.

APPENDIX 1

TREATISE ON DRILLING METHOD

IN-THE-HOLE DRILLS:

MAKING THE MOST OF THEM UNDERGROUND

Lane White, international editor

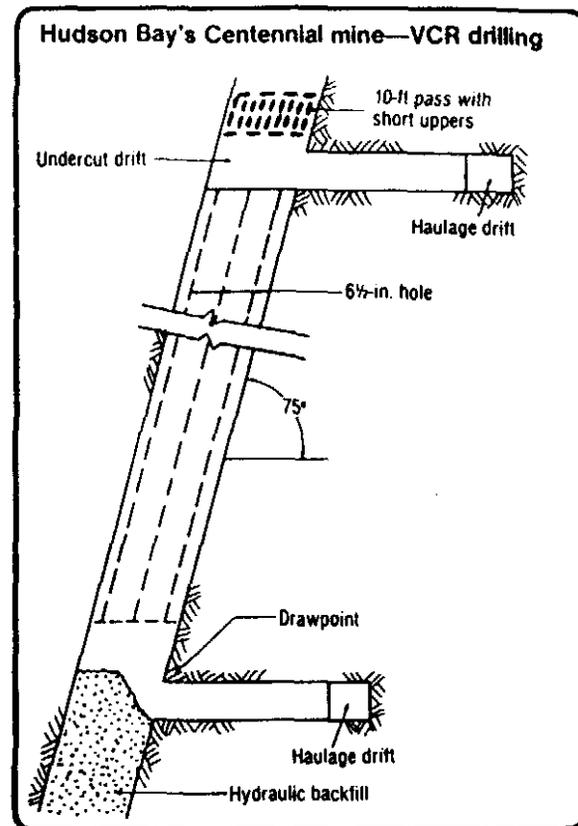
Since their introduction to underground mining operations in the early 1970s, in-the-hole drills drilling 4½- to 8-in.-dia holes have added a significant dimension to mine planners' stope design options. The drills have found use in blasthole open stoping¹⁻⁴, Vertical Crater Retreat mining⁹⁻¹², room-and-pillar benching¹³, and pillar recovery^{14,15}, as well as other applications, at mines both large and small throughout the world.

Inco Ltd.'s Sudbury, Ont., district nickel mines were among the first to recognize in-the-hole for reducing stoping costs when, in 1973, an in-the-hole drill was tried at the Copper Cliff North mine.¹ Within two years, Inco had 18 in-the-hole rigs in production stopes at its 12 Sudbury district mines. Subsequent cooperative development work by Inco and C.I.L. resulted in the patented Vertical Crater Retreat (VCR) mining procedure using in-the-hole rigs. Sudbury area mines, including those of Falconbridge Nickel as well as Inco's, continue to provide leadership in development of mining systems using the rigs.²

This article draws on earlier reports to review some aspects of in-the-hole drill application at four mines: Hudson Bay Mining and Smelting's Centennial mine near Flin Flon, Manitoba^{9,10}; Homestake Mining Co.'s Homestake mine at Lead, S. Dak.¹¹; Sunshine Mining Co.'s 16-to-1 mine, 40 mi southwest of Tonopah, Nev.⁶; and Ranchers Exploration and Development Corp.'s Escalante mine, 40 mi west of Cedar City, Utah³⁻⁵. All of the articles emphasize the importance of drill hole accuracy to achieve desired fragmentation and to minimize dilution, as well as the need for test stoping to establish optimum drill patterns and blasting procedure.

EVOLVING BIG-HOLE PRACTICE

In a paper, "Controlled Blasting to Minimize Over-

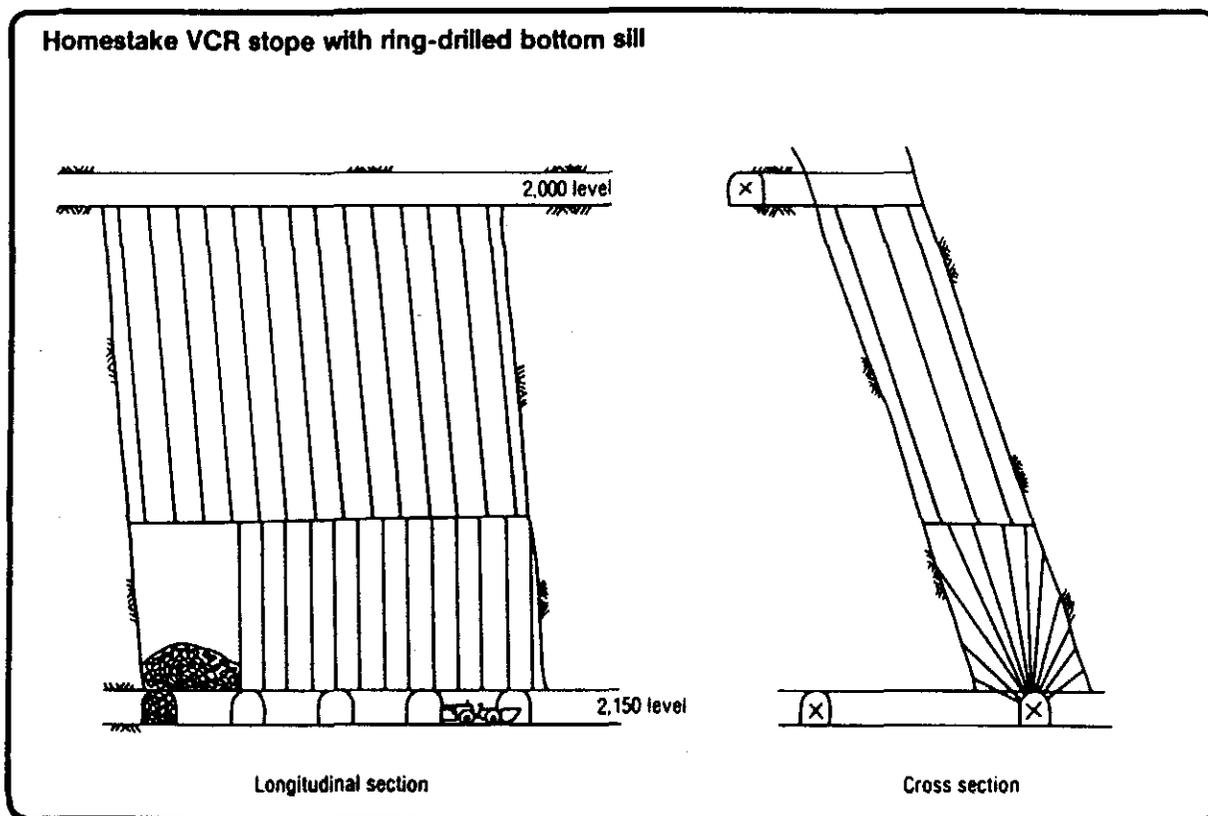


break with Big Boreholes Underground," presented at the CIM annual meeting in 1981, P.R. Day and W.K. Webster of C.I.L.'s Explosives Division introduced their subject by reviewing underground big-hole mining developments to that time.¹⁶ "These big blastholes, which range in size from 4½ in. to 8 in. dia, were originally employed in the same manner as open-pit bench blasts; that is, the stope was silled out to full width, and all holes were drilled vertical and parallel. This was suitable for competent, massive orebodies; however, the popularity of the big drills also saw their usage in more narrow stopes, often with poor ground conditions. Dipping orebodies and small drill sublevels resulted in inclined drilling.

"Today, there is full ring drilling of 4½-in. holes. Some 6½-in. holes are drilled up to 350 ft deep. A 350-ft borehole may contain 6,000 to 7,000 lb of explosive. The detonation of this size charge in the confines of an underground stope can have considerable effect on the stability of stope walls and adjacent pillars. Obviously, precautions must be taken to minimize the damage."

Among the special problems related to working with large boreholes is that of drilling accuracy, the authors said. "Large-diameter boreholes are being drilled to depths that are greater than any previous production blastholes. This has a multiplier effect on any errors or deviation in drilling accuracy. A 200-ft borehole that is off line by 10% can be 20 ft into a pillar, which may be designed only 20 ft wide."

Maximum deviations as low as 1-2% are possible with modern in-the-hole drills, the authors stated, but few operators achieve such accuracy. "The chief culprit appears to be improper drill set-up," they said, adding that drill hole alignment must be carefully supervised and drillers carefully trained. If an undercut is safe for



inspection, breakthrough holes should be surveyed and plotted.

"The importance of drilling accuracy to effective overbreak control cannot be overemphasized. The best attempts at wall control can be completely undone by improper hole alignment," the authors warned. "Blast layouts should also be designed with wall control in mind. Tight blast configurations should be avoided near the ultimate stope limits. For example, drop raises and slots are often choke-blasted. They should be located away from the final wall, as their confined nature often results in excessive radial cracking and seismic vibrations.

"Blast patterns should be designed in a manner that provides maximum relief area on blastholes. There appears to be a trend toward rectangular patterns rather than square. This reduces the horizontal burden for a given yield and should help in reducing backbreak. Although big boreholes are usually individually delayed, a 2:1 spacing-to-burden ratio is still recommended for ring drilling. . . .

"It is also noted that overbreak and dilution are more prevalent in multiple-row blasts rather than single- or double-row shots. It is expected that millisecond-delay blasting results in some buffering of back rows due to limited muck movements. It would be preferable if drill patterns were oriented in a manner to permit small blasts with the minimum number of rows, especially when approaching final walls and pillars.

"Narrow stopes often have a drill pattern that is only two or three boreholes across. In this case, every hole is a wall-control hole or at least a buffer hole, and special loading precautions must be taken for all the holes in order to minimize wall damage. In these stopes, the drill pattern cannot be expanded to optimum dimensions, and

there becomes an excess of borehole volume. The operator would be well advised to reduce the hole size, such as from 6½ in. to 4½ in. Indeed, there appears to be a trend toward 4½-in. boreholes for this reason and for the ability of some 4½-in. drills to operate at all angles."

VCR AT HUDSON BAY

In converting its Centennial mine to VCR stoping, HBM&S became one of the early users of this big-hole technology underground. Having broken five VCR stopes at Centennial, the company's chief engineer of mines, C.S. Crocker, described the company's VCR activity for the October 1978 American Mining Congress annual meeting in Las Vegas, crediting the earlier Inco-C.I.L. development work^{9,10} as follows:

"Canadian Industries Ltd., working with the International Nickel Co., had evolved a novel mining method to recover a vertical pillar from between two filled stopes at Inco's Levack mine. Briefly, the system took advantage of new technology that now permits drilling of large-diameter holes underground," Crocker reported.

"Their system called for an undercut and a topcut, with parallel 6½-in.-dia holes drilled from the top entry down to the undercut. Breaking was achieved by blasting the toes of the holes, taking advantage of the breaking characteristics of spherical charges and retreating the holes in stages.

"Results were very good and were published by I. C. Lang, the C.I.L. blasting specialist, who designed the system, and were also presented to an Underground Operators' Conference in Winnipeg, February 1977, by George Anderson, the Levack mine superintendent."

HBM&S consulted with Lang on use of VCR. During

the consultation, several specific aspects of VCR procedure were emphasized:

- Cratering depends on the breaking characteristics of the charge, such that charge length must not be more than six times charge diameter.
- In-the-hole drills, usually drilling 6- to 6½-in.-dia holes, enable use of charges of an effective size.
- Not all ores crater successfully.
- Cratering characteristics are unique for each ore, and crater design must be based on test work.
- VCR mining does not damage wall rock. At the Levack mine, the cratered pillar was broken against filled stopes without deleterious effects.
- Any VCR program should first be run as a test, and mine production should not be dependent on its immediate success.

With those considerations in mind, HBM&S began its VCR efforts with a single test stope, using an Atlas Copco ROC 306 carrier and a COP 7 in-the-hole drill to drill an 8 x 8-ft pattern in a stope 45 m high and 35 m along strike. "Blasting in the test stope was quite successful," Crocker reported. "The 8 x 8-ft pattern broke so easily, we decided to try 10 x 10-ft in the next stope. In one case, a burden of 14 ft was measured, and it, too, broke. The muck was excellent, requiring little secondary breaking."

In early VCR drilling at Centennial, drill-hole deviation resulting from striking hard sulphides at very oblique angles initially caused trouble. Stabilizers were tried but mainly resulted in failure of hammer casings. An Ingersoll-Rand Hi-Pac 300 compressor capable of supplying 250 lb/in.² at the drill was purchased, and the higher-pressure air resulted in improved penetration rates, longer runs per bit, better bit life, better hammer life, and straighter holes.

Tonnages broken in the five VCR stopes worked at Centennial up to the time of Crocker's report ranged from 18,123 tons in the smallest stope to 73,000 tons in the largest. A total of 18,518 ft of hole had been blasted in 135 holes, an average of 9.9 tons broken per foot and 1,359 tons per hole.

Explosives used included Hydromex, RPC, Pr-Core, and S.P. caps. An average of 0.75 lb of powder was required per ton broken.

Drilling and blasting costs, including labor, explosives, rod replacement, bit costs, and hammer and drill maintenance, averaged \$13.71 per foot drilled. Overall direct costs for VCR production were comparable to costs previously posted for longhole production, Crocker reported; but with VCR, appreciable savings were recorded in development and reduced secondary breakage. Inherent safety and good working conditions associated with VCR also contributed to a favorable reception, and HBM&S subsequently extended the use of VCR to its Anderson Lake mine, where it proved to be productive while at the same time avoiding ground problems previously encountered using other methods.²

VCR AT HOMESTAKE

Homestake Mining Co. began investigations for possible application of VCR stoping at its Homestake mine in Lead, S. Dak., in the late 1970s, and by 1980, VCR was accounting for about 30% of the mine's annual production of about 1.5 million st. In October 1980, Homestake's bulk mining foreman, Steven T. Mitchell, reported on VCR experience to that time at the Homestake mine—including an overview of crater blasting theory, details of Homestake's VCR blasting procedure, and a review of

some blasting problems encountered and solved—for the SME-AIME annual meeting in Minneapolis, Minn.¹¹

Homestake's VCR tests began in November 1977 using 4-in.-dia holes and continued after March 1978 with 6-in.-dia holes. Mitchell reported that "The main conclusions derived from the small-scale crater tests are:

- 1) Strain energy factors, optimum depth ratios, and crater volumes are dependent on the explosive-rock combination and must be determined by experimentation.
- 2) In hard-rock formations, there is a need to select explosives that exhibit relatively high densities, high detonation velocities, and high bulk strengths.
- 3) ANFO blasting agents are not suitable for true cratering applications due to relatively low density, detonation velocity, and bulk strength values.
- 4) Based upon results of the small-scale crater tests and known powder factors, a 9 x 7-ft pattern was selected for initial production blasting."

Subsequently, an 8 x 8-ft pattern was adopted for production blasting, with holes varying from 140 to 175 ft long, depending on their attitude. Holes are drilled at dip angles ranging from 45° to 90°.

At the time of Mitchell's presentation, four Gardner-Denver ATD 3100 crawler carriers, each equipped with a TRW Mission mast and rotating head assembly, were in use at Homestake. The rigs used air-powered hydraulics for thrust and rotation and had a total mast-assembly hoisting capacity of 12,500 lb. Air tramping motors were used to drive the carriers.

A Mission Megadrill B53-15 in-the-hole drill was used in conjunction with 6-ft x 5-in.-dia drill pipe. Operating air pressure at the hammers was 225-250 lb/in.² A skid-mounted Gardner-Denver-designed booster compressor package was used to raise mine air from about 100 lb/in.² to as high as 275 lb/in.².

Hole accuracy was a key consideration, because in VCR stoping, "deviations from a desired hole pattern almost always produce substandard blasting results," Mitchell said. "Improper drill alignment (azimuth and dip angle), the attitude and folding of the ore zone, too much feed pressure, and the weight of the drill string itself are the most probable causes of excessive hole deviation. Hole deviation resulting from the static load of the drill string becomes much more pronounced when drilling at dip angles less than 70° from the horizontal.

"Drill operators position the crawler along string-lines that define a drilling grid as marked on the stope walls by mine survey crews. Once the crawler has been situated, the mast is plumbed in the drilling plane and then set on the required dip angle with the aid of a magnetic angle finder (clinometer).

"Apparent hole wander with 160-ft holes averages 3%, but has approached 6-8% on occasion, necessitating fill-in drilling. Generally, the first 100 ft of drilling yields little or no deflection, whereas most of the deviation appears to accumulate in the last 50 ft of the hole. Once drilling has been completed in a section of the stoping area, mine survey crews establish a transit [survey] tie on all holes in the undercut to identify any hole deviation."

During the first half of 1980, VCR stopes at Homestake were producing 34.8 st/man-shift, against 25.1 st/man-shift in blasthole sublevel stopes and 16.7 st/man-shift in open cut-and-fill stopes. As more VCR stopes reached production, the additional tonnage realized was expected to offset development costs and continue to stabilize the overall VCR stoping cost. Ore grades produced from VCR stopes were lower at 0.149 oz/st gold than those from open cut-and-fill stopes at 0.221 oz/st;

however, stoping costs per ounce of contained gold were lower in the VCR stopes.

BIG HOLE STOPING AT 16-TO-1

When Sunshine Mining Co. selected a large-diameter, 6½-in., blasthole stoping method for its 16-to-1 silver mine in Nevada's Silver Peak mountain range, a key consideration was location of the mine in a remote area, where difficulty was anticipated in attracting and keeping skilled workers. The equipment-intensive mining system is now producing 755 st/d, five days a week with a workforce of 28 miners and an underground productivity of 27 st/man-shift. Equipment is available to enable an increase in production to 1,200 st/d if necessary.

"Briefly, this system involves drilling and blasting vertical slices of ore between sublevels into the void created by previous blasts [see illustration], with the broken material being extracted from drawpoints at the bottom of the orebody," said 16-to-1 project manager Allan R. Young at a Northwest Mining Association short course in Spokane, Wash., in November 1983.⁶ The conditions that made selection of the system possible were adequate vein width, exceeding 12 ft in most places and averaging 22 ft; a steep, 70-90° ore dip in most places; vein continuity; and competent wall and vein rocks, with definite hanging wall and footwall contacts.

A dilution factor of 10% was estimated for all reserve blocks, regardless of width. Following the start of stoping operations in October 1982, dilution soon became the most important production concern, with loading and blasting procedures providing the key in efforts to control the problem.

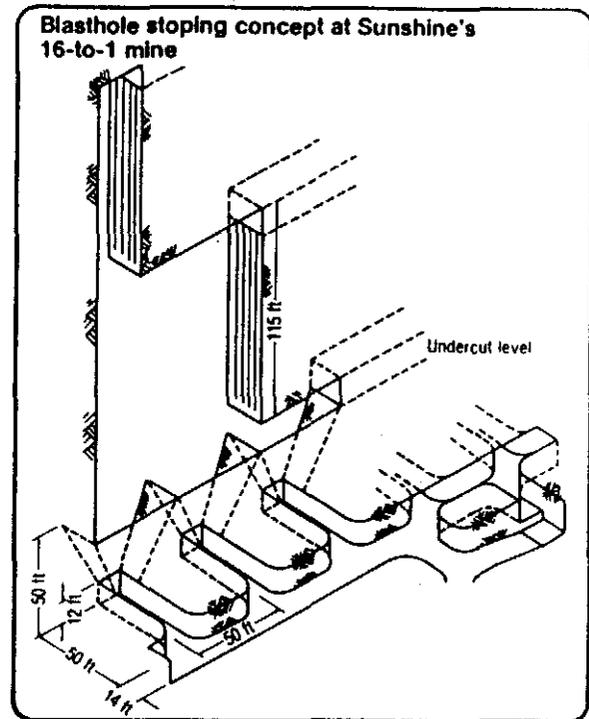
"Damaged walls were usually the result of too much explosive being used, while too little sometimes caused benches and pillars to form in the ore," Young said. "Almost one month of production was lost due to a large bench forming as a result of a missed hole. Following this, the procedure of double-priming was adopted, as it was believed that a primacord may have been cut off in the hole.

"Another cause of dilution was unexpected geologic structures located in the walls of the stope, which weakened the rock surrounding the vein. This was especially a problem in the hanging wall, and random pillars were left in the vein in an attempt to arrest the situation."

Selection of sublevel interval was a prime concern during mine planning, with major considerations being hole length vs. the amount of supervision required to maintain accuracy and minimization of dilution resulting from changes in dip. These and other factors contributed to a decision to establish the sublevel interval at 115 ft, sill to sill. Drawpoints on the loading level at the bottom of the mine were established on 50-ft centers and coned upward to an undercut level in the ore, about 35 ft above the drawpoint brows.

Two TRW Mission 6200-U in-the-hole drills, each with an 800-ft³/min, 250-lb/in.² booster compressor, do the drilling. Under normal conditions, penetrations rates of 50-70 ft/hr are common in quartz calcite vein. Accuracy is good, with deviations usually no more than 1 ft per 100 ft of hole.

Mining at the 16-to-1 begins by excavating a slot raise between sublevels at the extremity of the ore block. Production blastholes are drilled at a minimum of three holes across from footwall to hanging wall, parallel to the true dip of the vein. Row burden varies from 7 to 10 ft, depending on vein width. Powder factors are about 1.7



lb/st in narrow areas and less than that in wider zones.

Production blastholes are deck-loaded with an emulsion explosive in cartridge form and double-primed using time-delay sliding boosters. Blasting retreats from the raises. Broken ore is drawn down, shrinkage-fashion, to provide free space for following blasts, with remaining ore left to provide wall support until the orebody has been stoped out.

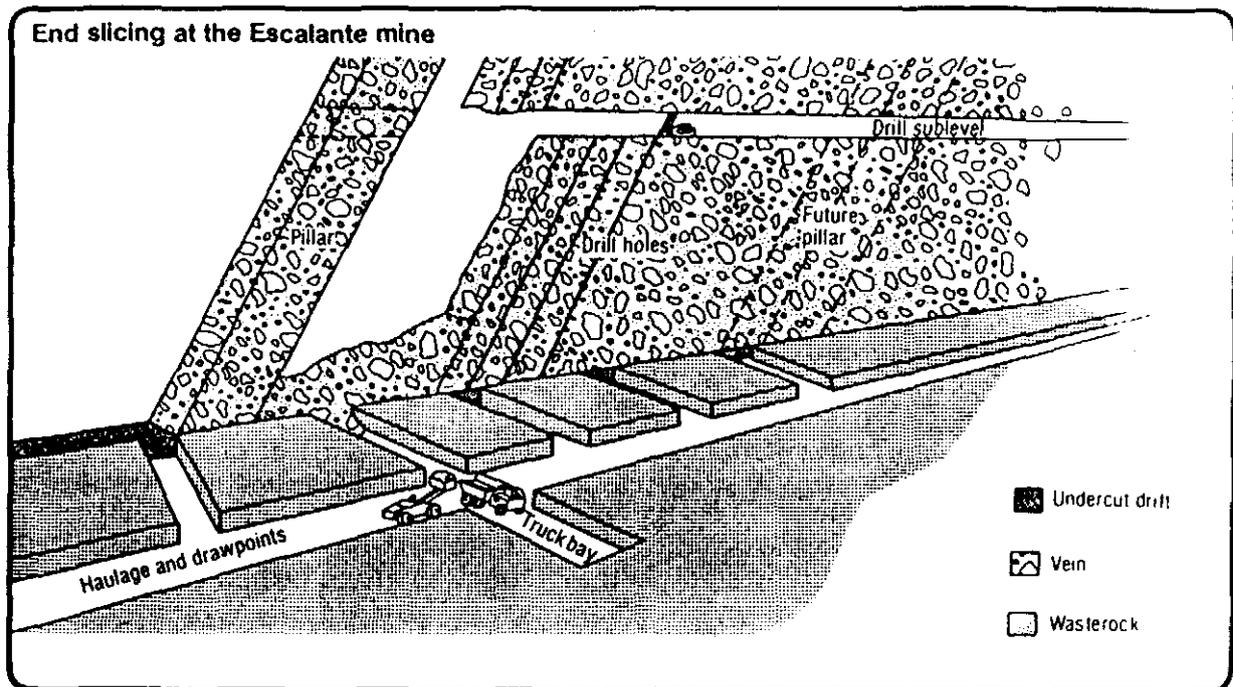
END SLICING AT ESCALANTE

At Ranchers Exploration and Development's Escalante silver mine, in-the-hole drills drilling 6½-in.-dia holes have been the main production drill rigs since the mine started up in 1981. However, the VCR stoping system used at startup has been converted to a blasthole open stoping system—called end slicing at Escalante. Advantages cited as a result of the conversion include reduced hole cleaning; elimination of blowback during blasting, which might damage timbered drill sublevels; and lower powder consumption.

Escalante silver mineralization occurs in a relatively wide, continuous quartz vein that is economic over a strike length of 3,500 ft, reaches depths of more than 800 ft, varies in thickness from 5 to 45 ft, and has an average true width of 19 ft. The vein dips about 73° west and has well-defined contacts.

Testing for VCR stoping was done by drilling from the surface outcrop to an undercut about 120 ft below surface, then retreat-blasting upward to reach a 30 ft crown pillar left between the surface and the stope. In the drill pattern, center holes are placed at 10-ft intervals along a center line on the strike of the ore, and side holes were placed at the hanging wall and footwall contacts and aligned with the center holes. Vein width was 5-10 ft at the surface in the test area and about twice that width at the undercut.

Regarding VCR results at Escalante, E.L. Catbagan, senior mine engineer, told a Society of Explosives Engineers conference in February 1983⁷ that "Fragmentation



was good, and practically no overbreak occurred on both walls. However, 33-50% of the holes blasted were plugged. These plugged holes were caused by 1) stemming freezing in the hole with polyethylene rope still hanging, and 2) vuggy holes further aggravated by concussion due to the blast. Usually, a blowpipe or in-the-hole drill is used to clean the holes. After the holes were cleaned, measurements showed only an average 7 ft of advance per blast."

Some changes in blasting practice brought improved results, but after consultation with Falconbridge and Inco indicated that similar problems had been encountered using VCR at some Sudbury area mines, a decision was made to test end slicing. A test stope was drilled from surface, with three-hole rows of increasing burdens at 7, 8, 9, 10, 11, and 12 ft. These were deck-loaded and shot one row at a time to determine fragmentation, with the 10-ft burden being the maximum to produce acceptable results. An additional test shooting of two rows at a time with appropriate delays produced "a beautiful blast. The break was right on the pillar, and the walls broke exactly on the contacts. Fragmentation was very good. Muck produced was 1,993 st, with a powder factor of 1.0 lb/st," Catbagan said.

Subsequent success in adapting the end slicing method to Escalante production stopes relied on a number of additional factors. Among these, Catbagan said, were the readiness of the staff to accept change, and diligence on the part of drillers and blasters toward their work. Daily observation and evaluation of production drilling and blasting results are done meticulously by the engineers. To blast the full width of the vein, holes are drilled right on the contacts, or as close as possible. Underground, drilling on the contacts was made possible by modifying the Ingersoll-Rand CMM-1 drill carrier so that the tower can be positioned 3 ft outside the tracks. Drilling is done with DHD 360 hammers and 4-ft rods.

Blasting sequences are carefully planned, with resulting excellent fragmentation and almost no overbreak. After 10 months and blasting of four stopes in an area 25 ft wide x 230 ft high x 450 ft long, stope walls were still

stable, with no sloughing at all.

End slicing also offered a further advantage in that major mine development is the same as for VCR. It is a simple matter to return to VCR when bad ground is met or where unusual pillar configurations are required. ■

REFERENCES

- 1) White, L. "Open-pit concepts boost Inco productivity underground." E&MJ, May 1975, p 101.
- 2) Lewis, A., "Inco: The World's No. 1 Ni Producer." E&MJ, November 1981, p 86.
- 3) Catbagan, E.L., "Production Drilling and Blasting Techniques at the Escalante Silver Mine." Society of Explosives Engineers Conference, February 1983.
- 4) Anderson, M.C., "Underground Mining Practice at the Escalante Silver Mine." International Gold/Silver Conference III, Reno, Nevada, June 1983.
- 5) Burger, J. R., "Ranchers end slices Escalante silver deposit." E&MJ, January 1984, p 48.
- 6) Young, A.R., "The 16-to-1 Project." presented at Northwest Mining Association short course, Spokane, Wash, November 1983 (Abstracted in E&MJ, May 1984, p 46).
- 7) White, L., "Mount Lyell passes 90th anniversary." E&MJ, November 1983, p 56.
- 8) Gomez, C. B., "Elura adds to EZ mine output." E&MJ, November 1983, p 60.
- 9) Crocker, C. S., "Vertical Crater Retreat Mining." American Mining Congress Meeting, Las Vegas, Nevada, October 1978.
- 10) White, L., "Diverse Mines Feed Hudson Bay's Flin Flon Zinc-Copper Complex." E&MJ, November 1981, p 168.
- 11) Mitchell, S. T., "Vertical Crater Retreat Stopping as Applied at the Homestake Mine." SME-AIME Fall Meeting, Minneapolis, Minn., October 1980.
- 12) Lang, R. C., Roach R. J., and Osoko, M.N., "Vertical Crater Retreat an Important New Mining Method." CANADIAN MINING JOURNAL, September 1977, p 69.
- 13) Dayton, S.H., "Asarco's Troy mine." E&MJ, February 1983, p 40.
- 14) Monahan, C.J., "The Crater Blasting Method Applied to Pillar Recovery at Falconbridge Nickel Mines Limited." Underground Operators Conference, February 19-21, 1979.
- 15) Meyer, K.V.S., "Improvements in Blasthole Drilling Productivity at the Sullivan Mine." Northwest Mining Association Convention, Spokane, Washington, December 1982.
- 16) Day, P.R. and W.K. Webster, "Controlled Blasting to Minimize Overbreak with Big Boreholes Underground." CIM Annual General Meeting, Calgary, May 1981.

APPENDIX 2

ORE LIBERATION STUDY

WARMAN INTERNATIONAL LTD.

(Incorporated in Queensland)

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ML:sh

RESEARCH & DEVELOPMENT DIVISION REPORT

83/3944

Report 83/157396

Phase 1 Testwork - Liberation Study
Tasmanian Tungsten/Tin Ore

for

Abignano Limited

M. Lanyon

September 16, 1983

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SUMMARY

- * A sample of tungsten-bearing ore was examined to assess its liberation characteristics relative to gravity concentration.
- * The sample contained 0.74% WO_3 , 0.06% As and less than 0.01% SnO_2 .
- * The tungsten was present as wolframite and also scheelite.
- * Heavy liquid separation produced a concentrate containing 93.7% of the tungsten at a grade of 42.2% WO_3 .
- * The tungsten minerals present were coarse and well-suited to primary concentration by gravity means.
- * The upgrading of a primary concentrate to sales grade is likely to be a complex process. The presence of scheelite limits the extent to which high intensity magnetic separation can be used.
- * There was no significant concentration of any tin minerals.

1. INTRODUCTION

The sample comes from a tungsten/tin prospect on Tasmania's west coast. Run-of-mine ore was said to consist of 3 parts greisenised granite, containing some sulphides, to one part mineralised quartz reef, the quartz reef consisting of 75% quartz, 10% tourmaline, 5% pyritic sulphides, 5% mica and 5% coarse wolframite.

The relative friability and low specific gravity of the reef and greisen material compared to the coarse wolframite suggested crushing followed by cycloning to remove low specific gravity waste prior to standard gravity concentration techniques.

2. AIM

This testwork looked at the liberation characteristics of the tungsten minerals in the ore, with the aim of indicating the tungsten recovery that might be obtained by gravity concentration processes. High intensity magnetic separation was added to improve the concentrate grade by separating the pyritic sulphides from the slightly magnetic wolframite.

3. PROCEDURE

The test procedure is shown diagrammatically in Figure 1.

3.1 Sample Preparation

Approximately 68 kg of ore arrived on August 12, 1983 (Warman Laboratory Number R3882). After drying at 100°C it was crushed to pass a 6.8 mm screen, then riffle blended. A 5.5 kg sample was split out, from which 500 g was roll crushed to minus 2 mm to provide 150 g for the head assay. It was pulverised before analysis.

3.2 Heavy Liquid Separation

The 5 kg parcel of nominally minus 7 mm ore was wet screened on 0.3 mm. The undersize material was then deslimed by decantation and the slimes were collected, dried, weighed and assayed.

After drying, the +300 µm material was separated in 200 g batches in tetrabromoethane (TBE, sp. gr. 2.9), with a little agitation to reduce entrainment during the separation. The LIGHTS (floating material) were screened into a filter, drained and repeatedly washed with acetone. Any material not definitely settled within 5 minutes was classified as LIGHTS. The HEAVIES (sinking material) were similarly recovered, washed with acetone on a filter and dried, before repeating the heavy liquid separation procedure using diiodomethane (DIM, sp. gr. 3.3).

A similar procedure was applied to the deslimed minus 300 µm material, with smaller quantities being separated each time.

The TBE LIGHTS and DIM LIGHTS were screened at 2 mm, 1 mm, 850 µm, 500 µm, 250 µm and 106 µm.

3.3 High Intensity Magnetic Separation

The DIM HEAVIES product was screened at 2 mm, 1 mm and 500 μm . The +1 mm fractions were observed under ultra-violet light and then assayed for tungsten. The minus 1 mm DIM HEAVIES were separated into magnetics and non-magnetics using a high intensity induced roll magnetic separator. The non-magnetics tailing and the magnetic concentrates were each screened at 850 μm , 500 μm , 250 μm and 106 μm and the fractions pulverised for analysis.

All heavy liquid separation product size fractions were weighed and assayed for tungsten. Tin analyses were conducted on particular fractions of each product, representing those richest in tin.

4. RESULTS AND DISCUSSION4.1 The Ore

The ore received from the tungsten/tin deposit was said to consist of one-quarter quartz reef containing 5% coarse wolframite, and a little tin. The ore assayed 0.74% WO_3 , less than 0.01% SnO_2 and 0.06% As. The tungsten assay corresponds to approximately 0.9% wolframite.

The ore was found to contain scheelite, (calcium tungstate) as well as wolframite (iron-manganese tungstate), the scheelite being observable as fluorescent grains under ultra-violet light. Scheelite is generally considered a friable mineral whereas wolframite is less friable than the host rock. Both minerals, as shown below, are of higher specific gravity than the host rock.

<u>Mineral</u>	<u>Specific Gravity</u>	<u>Hardness</u>
wolframite	7.0 to 7.5	5 to 5.5
scheelite	5.9 to 6.1	4.5 to 5
pyritic sulphides	5	6
tourmaline	3.0 to 3.25	7 to 7.5
mica	2.76 to 3.10	2 to 2.5
quartz	2.66	7.0
greisenised granite	2.6 to 2.9	-
ore average	2.8	-
TBE	2.9	
DIM	3.3	

4.2 Heavy Liquid Separation - Theory

The densities of the heavy liquids tetrabromoethane (TBE) and diiodomethane (DIM) are 2.9 and 3.3 respectively. Thus on the basis of separating grains of pure minerals, a TBE separation of this ore's constituents will remove as TBE LIGHTS the quartz, granite and some mica. Separation of the TBE HEAVIES in DIM will then remove as DIM LIGHTS the tourmaline and remaining mica, leaving as DIM HEAVIES a concentrate containing wolframite, scheelite, pyritic sulphides, and if present, minerals of the garnet family.

Composite particles, depending on the ratio of each mineral, will report to the sink or float product, losing recovery of tungsten to the LIGHTS or decreasing the grade of the HEAVIES, until the particles are fine enough to consist of the pure minerals. Thus the information on the coarsest size at which breaking of composites and liberation of individual grains is significant, is obtained by observation and assay of the product size fractions after TBE and DIM separations of the crushed ore.

The results obtained are presented in four tables illustrating different aspects which will be discussed in more detail.

4.3 Liberation

Table 1 presents the size analyses of the 5 kg test parcel and the tungsten distribution according to size fraction. This table was constructed from the weights and assays of the various test products and shows that the tungsten mineralisation has tended to remain in the coarser fractions and is relatively more resistant to grinding compared to the gangue minerals. The head assay calculated from all the products came out at 1.08% WO_3 , which is significantly higher than the actual assay of 0.74% WO_3 . The actual assay is the more reliable figure.

Table 2 presents the distribution of tungsten to the various specific gravity fractions and the distribution of tungsten in the DIM HEAVIES

to several sub-fractions. The particular feature of these results is that over 90% of the tungsten reports to the +3.3 sp. gr. fraction at a grade of 42% WO₃; also 70% of the tungsten is associated with +1 mm particles.

Further liberation data is presented in Table 3 which gives the sizing and assay for each specific gravity fraction and the tungsten distribution within each specific gravity fraction relative to particle size. Also given in Table 3 is the sizing and assays for the magnetic and non-magnetic portions of the minus 1 mm DIM HEAVIES.

Table 4 is an alternative presentation of the Table 3 data but calculated to show the weight and tungsten distribution of the specific gravity fractions within each size fraction.

Significant points brought out in Tables 3 and 4 are:

- the preference for the tungsten distribution in the DIM HEAVIES to be biased to the coarser size fractions. Thus 88.2% of the tungsten in the DIM HEAVIES is in particles coarser than 500 µm. This characteristic is consistent with wolframite mineralisation.
- the assays of the TBE LIGHTS presents the liberation in a different way. Each fraction except the fine sands contains only 0.02 to 0.04% WO₃ indicative of almost complete liberation of tungsten from 90% of the gangue.
- the DIM HEAVIES decrease in grade in the finer sizes due apparently to increased liberation of heavy non-tungsten minerals.
- the tungsten assays of the NON-MAGNETIC heavies are relatively high indicative of the presence of scheelite.
- the tungsten assays of the MAGNETIC heavies are between 17% and 50% WO₃ which means that the wolframite contents range 20 to 60% so there must be present a significant quantity of composite particles and/or heavy non-tungsten magnetic minerals.

4.4 Mineralogical Examination

The +0.5 mm grains of DIM HEAVIES were observed to contain many composites. As well as wolframite and pyritic sulphides there was a considerable amount of a white mineral which fluoresced under ultra-violet light and was obviously scheelite. It is estimated that the scheelite could account for 30% of the tungsten present. Composites were similarly visible in the fine high intensity magnetic separator products.

Samples of the coarse DIM HEAVIES and minus 1 mm MAGNETICS AND NON-MAGNETICS have been retained. A full mineralogical examination should be conducted on these to identify the minerals and composite proportions present before more tests are proposed to upgrade the DIM HEAVIES.

4.5 Tin Distribution

Tin assays of the specific gravity fractions were 0.01% Sn or less. There is thus no tin of commercial significance in the sample. (See Table 5).

4.6 Process for Tungsten Recovery

The practical implications of the results of the liberation study are that the ore, as represented by this sample, would be well-suited to gravity processing for beneficiation of the tungsten minerals. The test results indicate that one could expect to recover close to 90% of the tungsten in a primary gravity circuit. As a significant proportion of the tungsten mineralisation is coarse, a process involving jigs and table concentrators would seem to be the preferred approach.

While the primary recovery aspects of the sample appear well defined by this preliminary test programme, the matter of concentrate grade is

not so clear. For tungsten concentrate to be marketable without penalty, a minimum grade of 65% WO₃ is usually required and specific elements such as molybdenum, tin, arsenic, sulphur and phosphorus attract penalties. The DIM HEAVIES averaged 42% WO₃, so there is obviously need for further up-grading and it would be a matter of additional testwork to determine the best approach. It is clear from the testwork that the application of magnetic separation to further up-grading is not clear-cut since the tungsten mineralisation is both scheelite and wolframite. Ultimately the concentrate cleaning process is likely to be relatively complex and would include re-crushing or regrinding the coarse primary concentrate, screening, further gravity concentration, magnetic separation, high tension separation and perhaps flotation. There would inevitably be some loss of tungsten values in concentrate cleaning and this could amount to 10% or more of the tungsten.

FIGURE 1
Schematic Test Procedure

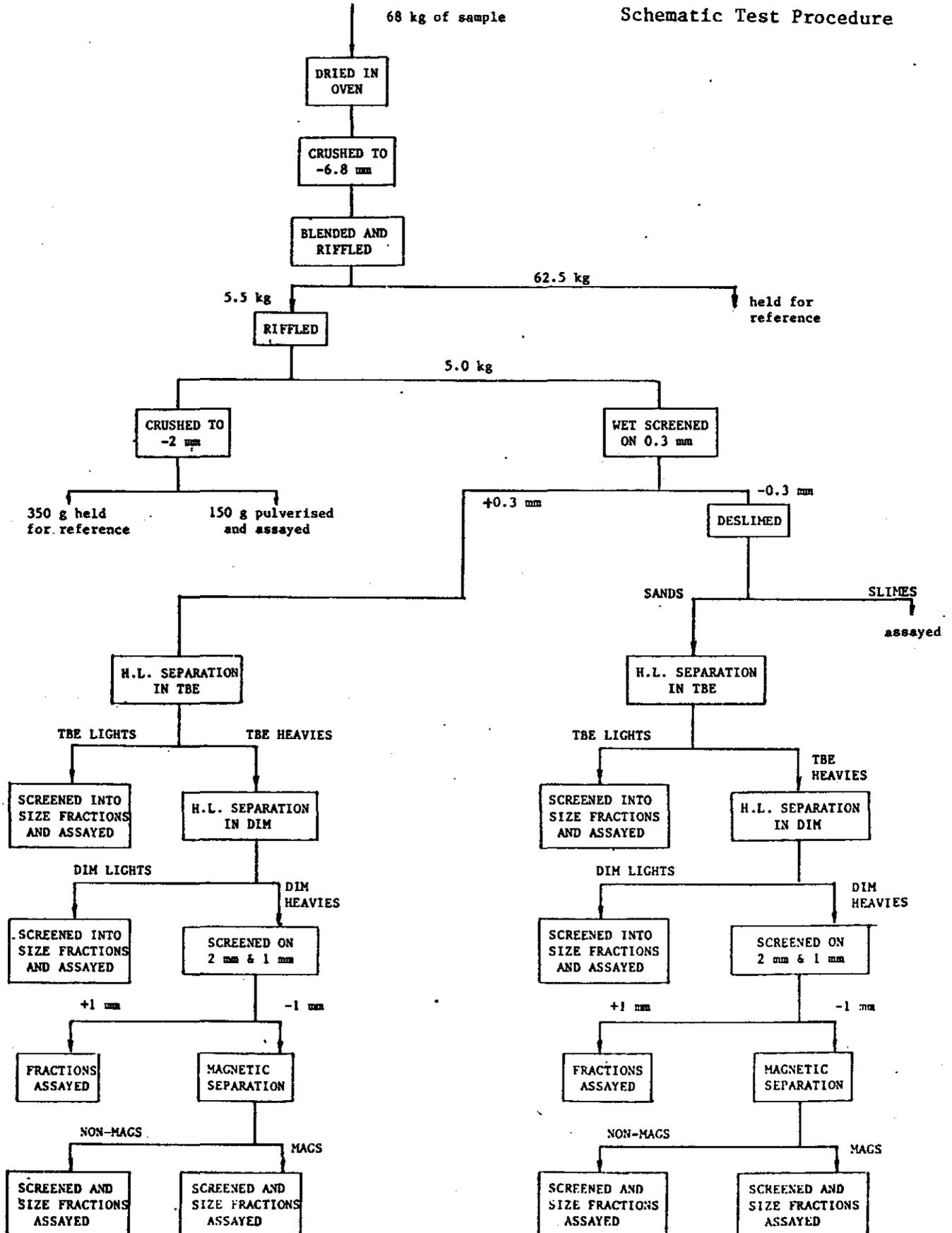


TABLE 1

Distribution of tungsten in the crushed sample with respect to particle size

size fraction µm	weight % retained	cumulative weight % passing	assay % WO ₃ (calc)	tungsten distribution %	
				fractional	cumulative passing
-7000 +2000	43.6	56.4	1.44	58.4	41.6
-2000 +1000	17.5	38.9	1.03	16.7	24.9
-1000 + 500	12.2	26.7	1.00	11.3	13.6
- 500 + 250	8.4	18.3	0.63	4.9	8.7
- 250 + 106	7.5	10.8	0.57	3.9	4.8
- 106 +slimes	3.9	6.9	0.89	3.2	1.6
slimes	6.9	-	0.25	1.6	-
total	100	-	1.08	100	-

TABLE 2

Distribution of tungsten with respect to
specific gravity fractions.
(Sample crushed to -6.8 mm)

fraction	sp. gr.	weight %	% WO ₃	distribution WO ₃ %
DIM heavies *	+3.3	2.39	42.2	93.74
DIM lights (mids)	-3.3 +2.9	2.65	0.63	1.56
TBE lights	-2.9	88.06	0.04	3.10
slimes	not separated	6.90	0.25	1.60
feed	100	100	(1.08)	100

<u>* DIM HEAVIES</u>				
+2 mm		1.06	56.8	55.68
+1 mm		0.38	46.0	16.02
magnetics		0.36	27.2	9.01
non-magnetics		0.60	23.2	13.03
total DIM heavies		2.39	(42.2)	93.74

brackets indicate calculated assays

TABLE 3

Sizing of the specific gravity fractions
and the distribution of tungsten within
the specific gravity fractions

size fraction µm	FEED			TBE LIGHTS			DIM LIGHTS (TBE HEAVIES)			DIM HEAVIES			NON-MAGNETICS (DIM HEAVIES)			MAGNETICS (DIM HEAVIES)		
	weight %	assay % WO ₃	WO ₃ dist %	wt % of TBE lights	assay % WO ₃	WO ₃ dist of TBE lights	wt % of DIM lights	assay % WO ₃	WO ₃ dist of DIM lights	wt % of DIM heavies	assay % WO ₃	WO ₃ dist of DIM heavies	wt % of non-mags treated	assay % WO ₃	WO ₃ dist of non-mags treated	wt % of magnetics treated	assay % WO ₃	WO ₃ dist of magnetics treated
-6800 +2000	43.6	1.44	58.4	47.9	0.04	50.6	14.4	3.2	72.4	44.1	56.0	59.4	NS	-	-	NS	-	-
-2000 +1000	17.5	1.03	16.7	19.1	0.03	15.1	9.9	0.71	11.0	15.7	46.0	17.1	NS	-	-	NS	-	-
-1000 +850	4.1	1.00*	11.3	4.5	0.02	2.4	3.1	0.28	1.4	14.0	35.3*	11.7	20.5	37.1	32.7	10.5	50.6	19.6
-850 +500	8.1			8.7	0.03	6.9	7.7	0.20	2.4				28.6	30.7	37.9			
-500 +250	8.4	0.63*	4.9	8.9	0.03	7.0	14.5	0.09	2.1	8.3	25.3*	5.0	21.0	18.9	17.1	19.6	37.0	26.6
-250 +106	7.5	0.57*	3.9	7.4	0.03	5.9	27.0	0.07	3.0	9.4	17.7*	3.9	18.4	9.3	7.4	32.0	25.8	30.4
-106 +slimes	3.9	0.89*	3.2	3.5	0.13	12.1	23.4	0.21	7.7	8.5	14.5*	2.9	11.5	10.0	4.9	37.9	16.8	23.4
slimes	6.9	0.25	1.6	NS	-	-	NS	-	-	NS	-	-	NS	-	-	NS	-	-
TOTALS OF FRACTION				100		100	100		100	100		100	100		100	100		100
TOTAL FEED	100	1.08*	100		0.04*			0.63*			42.21*			23.2*			27.2*	
% OF FEED		(0.74)		88.0		3.1	2.7		1.6	2.4		93.7	0.6		13.0	0.36		9.0

* indicates calculated assays

NS indicates "not separated"

Note: tungsten and weight distributions are percentages
of each specific gravity or magnetic fraction

245030

TABLE 4

Alternative presentation of Table 3 showing the weight and tungsten distribution of the specific gravity fractions within each size fraction

size fraction µm	FEED			TBE LIGHTS			DIM LIGHTS TBE HEAVIES			DIM HEAVIES			NON-MAGNETICS			MAGNETICS		
	weight %	assay % WO ₃	WO ₃ dist %	weight % of size	assay % WO ₃	WO ₃ dist % of size	weight % of size	assay % WO ₃	WO ₃ dist % of size	weight % of size	assay % WO ₃	WO ₃ dist % of size	weight % of size	assay % WO ₃	WO ₃ dist % of size	weight % of size	assay % WO ₃	WO ₃ dist % of size
-6800 +2000	43.6	1.44*	58.4	96.7	0.04	2.7	0.9	3.2	1.9	2.4	56.0	95.4	NS	-	-	NS	-	-
-2000 +1000	17.5	1.03*	16.7	96.4	0.03	2.8	1.5	0.71	1.0	2.1	46.0	96.2	NS	-	-	NS	-	-
-1000 +500	12.2	1.00*	11.3	94.9	0.03*	2.6	2.3	0.22*	0.5	2.7	35.3*	96.9	0.3	33.4	81.3	1.4	50.6	15.6
-500 +250	8.4	0.63*	4.9	93.1	0.03	4.5	4.6	0.09	0.6	2.3	25.3*	94.9	0.8	18.9	45.7	1.5	37.0	49.2
-250 +106	7.5	0.57*	3.9	87.4	0.03	4.6	9.6	0.07	1.2	3.0	17.7*	94.2	1.5	9.3	24.4	1.5	25.8	69.8
-106+slimes	3.9	0.89*	3.2	79.1	0.13	11.6	15.7	0.21	3.7	5.2	14.5*	84.7	3.4	10.0	19.8	1.8	16.8	64.9
slimes	6.9	0.25	1.6	NS	-	-	NS	-	-	NS	-	-	NS	-	-	NS	-	-
TOTAL FEED	100	1.08*	100	88.0	0.04*	3.1	2.7	0.63*	1.6	2.4	42.21*	93.7	0.6	23.2*	13.0	0.36	27.2*	9.0

* indicates calculated assays

NS indicates "not separated"

Note: tungsten and weight distributions are percentages of each size fraction

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TABLE 5

Tin assays of specific gravity fractions

Product	size fraction assayed	tin assay % SnO ₂
slimes		0.01
TBE lights	(+2 mm)	<0.01
DIM lights	(+2 mm)	<0.01
DIM lights	(-106 μm)	<0.01
DIM heavies	(+2 mm)	<0.01
head		<0.01

< : less than

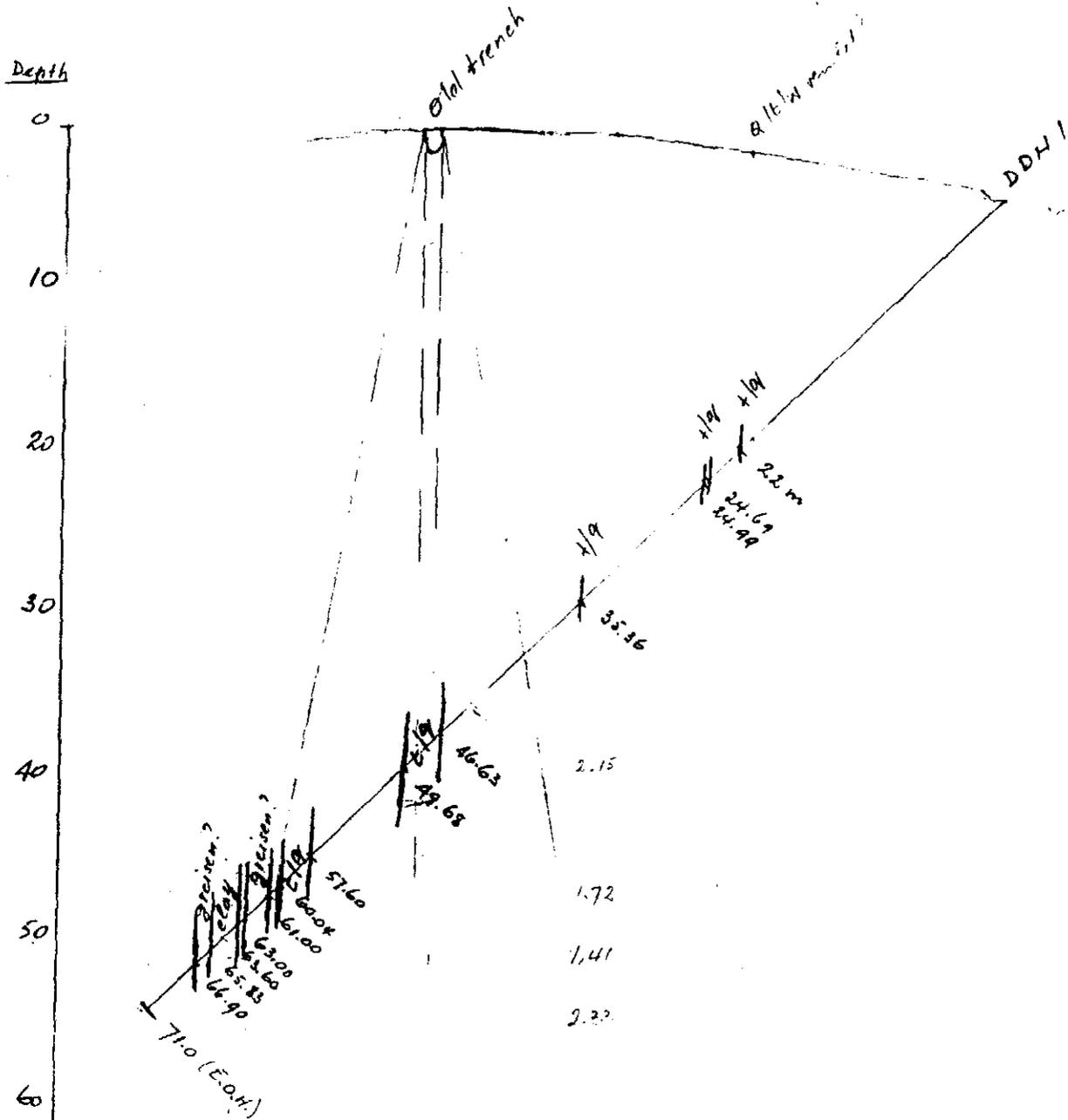
APPENDIX 3

DRILL HOLE LOGS AND PRELIMINARY CROSS-SECTIONS

Interview River DDH-1

Section through hole - Driller's log (9/12/84)

Scale 1:400, inclination 45°



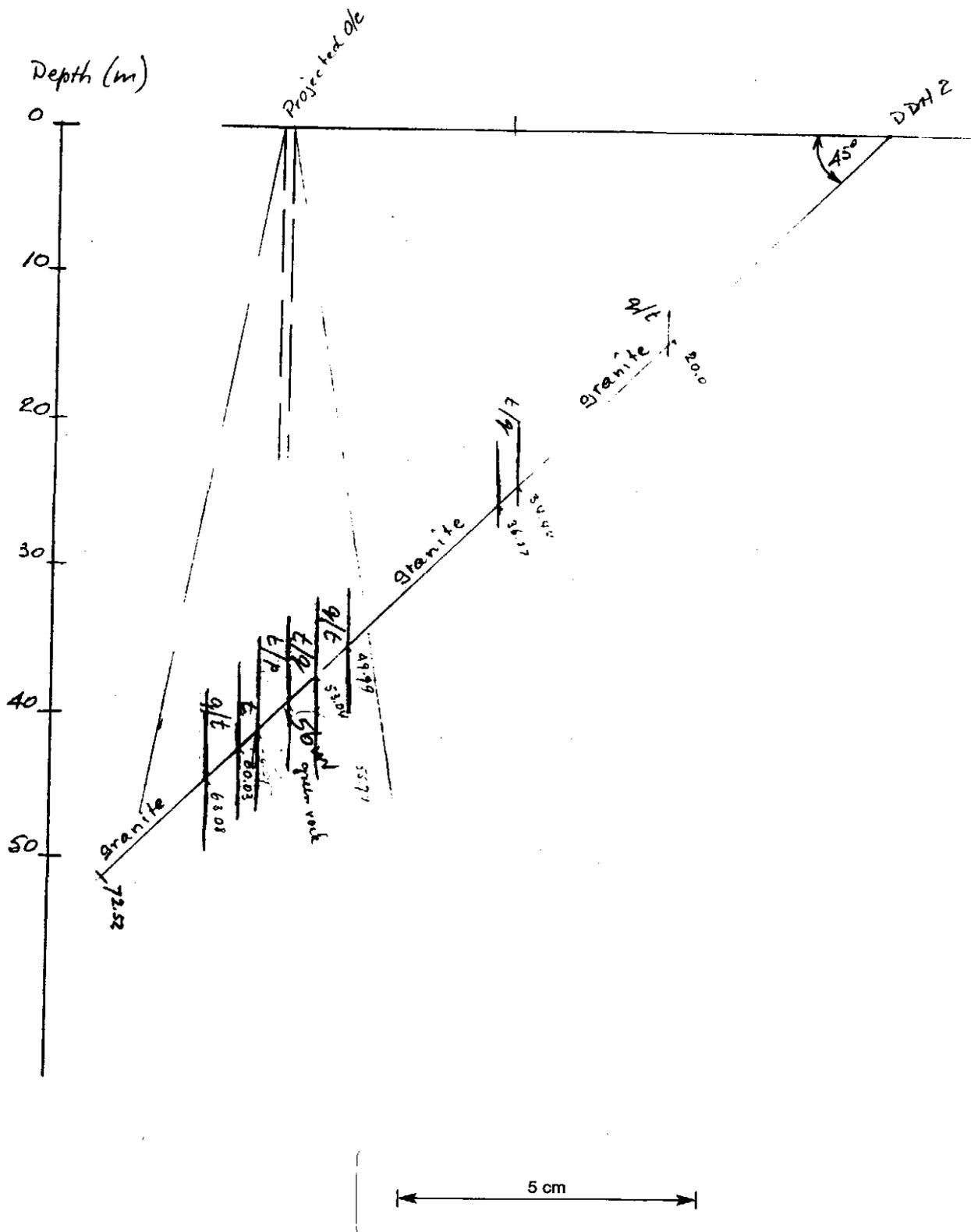
INTERVIEW RIVER - DDH 2

Driller's logs

SECTION. INCLINATION OF HOLE 45° WEST

Scale 1:400 (HOR=VER)

Hole finished 18/12'80



ABIGNANO LIMITED

MINING DIVISION

FIELD LOG OF DRILL HOLE NO. DDH 1 - 45° inclination

DRILLING METHOD: N.Q. Diamond Core
 DRILLER: Peter McCormick / Gary Howell
 LOGGER: B. Porink
 DATE COMMENCED: 28/11/84

PROJECT: INTERVIEW RIVER
 COLLAR LEVEL:
 TOTAL DEPTH: 71.00 metres.
 DATE COMPLETED: 5/12/84

INTERVAL(m)	SAMPLE	DESCRIPTION	
43.0 to 44.15	1. 43.0-44.15	Granite - altered, mainly qz. and muscovite, some biotite and hornblende altering to chlorite/epidote	
to	1. 43.0-44.15		
to			
44.15 to 48.63	2. 44.15-46.63	Granite - grey, med/coarse grained, slightly altered. A coarser, porous leached qz. rich band between 44.50 and 44.80 m. At 46.70 a small patch of mica rich material with a rosette shaped white mineral (scheelite?)	Box 8
to	3. 46.63-48.63		
to			
48.70 to 54.70		Granite - grey with black specks (biotite?) and dark diffuse patches (tourmaline?) Altered generally with green "clay" coated joint planes, which show evidence of movement. No evidence of sulphides	Box 9
to			
54.70 to 59.0		Granite - light grey, altered, med/coarse, greisenised, some chlorite coated joints. Distinct striation (schlieren?) in places (eg. 55.3 m) Note: core mixed up in transit. Some sulphides (chalcocite) in patches.	
to			

ABIGNANO LIMITEDMINING DIVISIONFIELD LOG OF DRILL HOLE NO. DDH 1

DRILLING METHOD:

PROJECT: INTERVIEW RIVER

DRILLER:

COLLAR LEVEL:

LOGGER:

TOTAL DEPTH:

DATE COMMENCED:

DATE COMPLETED:

INTERVAL	SAMPLE	DESCRIPTION	
59.0 to 59.80 to to to 59.80 to 60.00 to	4. 59.0-59.80	Granite - altered (preisen) with Fe, Cu, As pyrites. Abundant black mineral in central part with grey/white mineral (torromaline + schelite?) Granite - grey, altered	Box 10
60.0 to 61.40 to to to to to to	5. 59.80-61.40	Granite - grey, altered with brownish porous patches (oxidised sulphides?) Semi-digested "pebble" at 60.8m. Core becomes increasingly pitted and rough (eg) with thin veins of green chlorite.	
61.40 to 63.66 to to to	6. 61.40-63.66	Granite - grey with black specks, soft and friable. Many green coated joint planes, some brownish bands. black minerals → chlorite	Box 11
63.66 to 64.80 to to	7. 63.66-64.80	Clay - gritty, grey/brown, probably extremely altered granite.	
64.80 to 65.83 to to to		Granite - grey, very friable, extremely altered with abundant brown specks. White mineral schelite?	

ABIGNANO LIMITED

MINING DIVISION

FIELD LOG OF DRILL HOLE NO. DDH 1

DRILLING METHOD:
 DRILLER:
 LOGGER:
 DATE COMMENCED:

PROJECT: INTERVIEW RIVER.
 COLLAR LEVEL:
 TOTAL DEPTH:
 DATE COMPLETED:

INTERVAL	SAMPLE	DESCRIPTION	
65.83 to 67.34 to to to to	8. 64.80-67.34	<u>Granite</u> - friable and broken, black specks, some pink colouration Green coated joints and altered biotite/hornblende/tourmaline? Very fine speck of gold?	Box 12
67.34 to 69.17 to to	9. 67.34-69.17	<u>Granite</u> - altered, fairly friable and broken. Some pink and green colouration.	
69.17 to 71.00 to		<u>Granite</u> - altered, as above, 2 cm thick band? of hard, dark green vitreous mineral (hornblende?) Some evidence of very fine sulphides towards base.	
		END OF HOLE	

ABIGNANO LIMITED
MINING DIVISION

FIELD LOG OF DRILL HOLE NO. DDH 2

DRILLING METHOD: *NQ coring*
DRILLER: *P. McCormick*
LOGGER: *B. Porink*
DATE COMMENCED: *7/12/84*

PROJECT: *INTERVIEW RIVER*
COLLAR LEVEL:
TOTAL DEPTH: *72.72 m.*
DATE COMPLETED: *18/12/84*

INTERVAL	SAMPLE	DESCRIPTION	
47.78 to 51.31 to to to to		<i>Granite, slightly altered, core broken in places. Medium to coarse grained granite, (35, feldspar, muscovite, biotite, hornblende observed) feldspar slightly kaolinised. No sulphides observed.</i>	Box 8
51.31 to 52.10 to to	<i>2/1</i>	<i>Granite, altered with pegmatite bands containing sulphides and wolframite? Scattered sulphides throughout interval.</i>	
52.10 to 53.00 to to		<i>Granite, near fresh, some sulphides on joint/slip planes.</i>	
53.00 to 53.30 to to to to to to to	<i>2/4</i> <i>2/5</i>	<i>Granite, slightly altered.</i> <i>Granite/pegmatite, mineralised, specks of chalcopyrite and arsenopyrite at top, increasing with depth decreasing towards base. Patches of black wolframite with schreibite? rims and coarse patches of yellow chalcopyrite, coarse muscovite and quartz, some tourmaline. A light to dark purple crystalline mineral inclusions in black patches of wolframite (fluorite?)</i>	Box 9
53.80 to 54.83 to	<i>2/6</i>	<i>Granite, slightly altered, thin bands of sulphides, also scattered sulphides</i>	

ABIGNANO LIMITEDMINING DIVISIONFIELD LOG OF DRILL HOLE NO.

DRILLING METHOD:

PROJECT:

DRILLER:

COLLAR LEVEL:

LOGGER:

TOTAL DEPTH:

DATE COMMENCED:

DATE COMPLETED:

INTERVAL	SAMPLE	DESCRIPTION	
to 54.83 to 55.77	2/7	some dull tourmaline, slickensided faces, some scheelite? <u>Granite</u> , as above. some wolframite?	Box 9
to 55.77 to 56.71	2/8	scheelite?, speck of gold? <u>Granite</u> , altered (greisenised) patches	
to		and veins of sulphides, some	
to		tourmaline, wolframite? speck	
56.71 to 58.60	2/9	of gold? Core ven. broken. <u>Granite</u> , slightly altered, rare sulphides,	
to		but yellow oxidised pits (after sulphides?) some brown acicular	
to		tourmaline. Thin band of	
to		tourmaline at 57.70 and	
to		faint bands of mica rich	
to		greisen? Massive core	
58.60 to 59.33	2/2	<u>Granite</u> , slightly altered with scattered	Box 10
to		yellow sulphides, conspicuous	
to		longitudinal vein rich in	
to		sulphides with tourmaline	
to		penetrated into adjacent granite	
to		Some wolframite? and some	
to		broken yellow sulphide (gold?)	
to		Massive core	
59.33 to 61.25	2/3	<u>Granite</u> , slightly altered, with thin	
to		veins of 93/tourm / sulphides	
to		at 60.23 m and 61.23 m.	
to		Pits in sulphides have tarnished	
to		flourishings.	
to		Massive core	

ABIGNANO LIMITEDMINING DIVISIONFIELD LOG OF DRILL HOLE NO.

DRILLING METHOD:

PROJECT:

DRILLER:

COLLAR LEVEL:

LOGGER:

TOTAL DEPTH:

DATE COMMENCED:

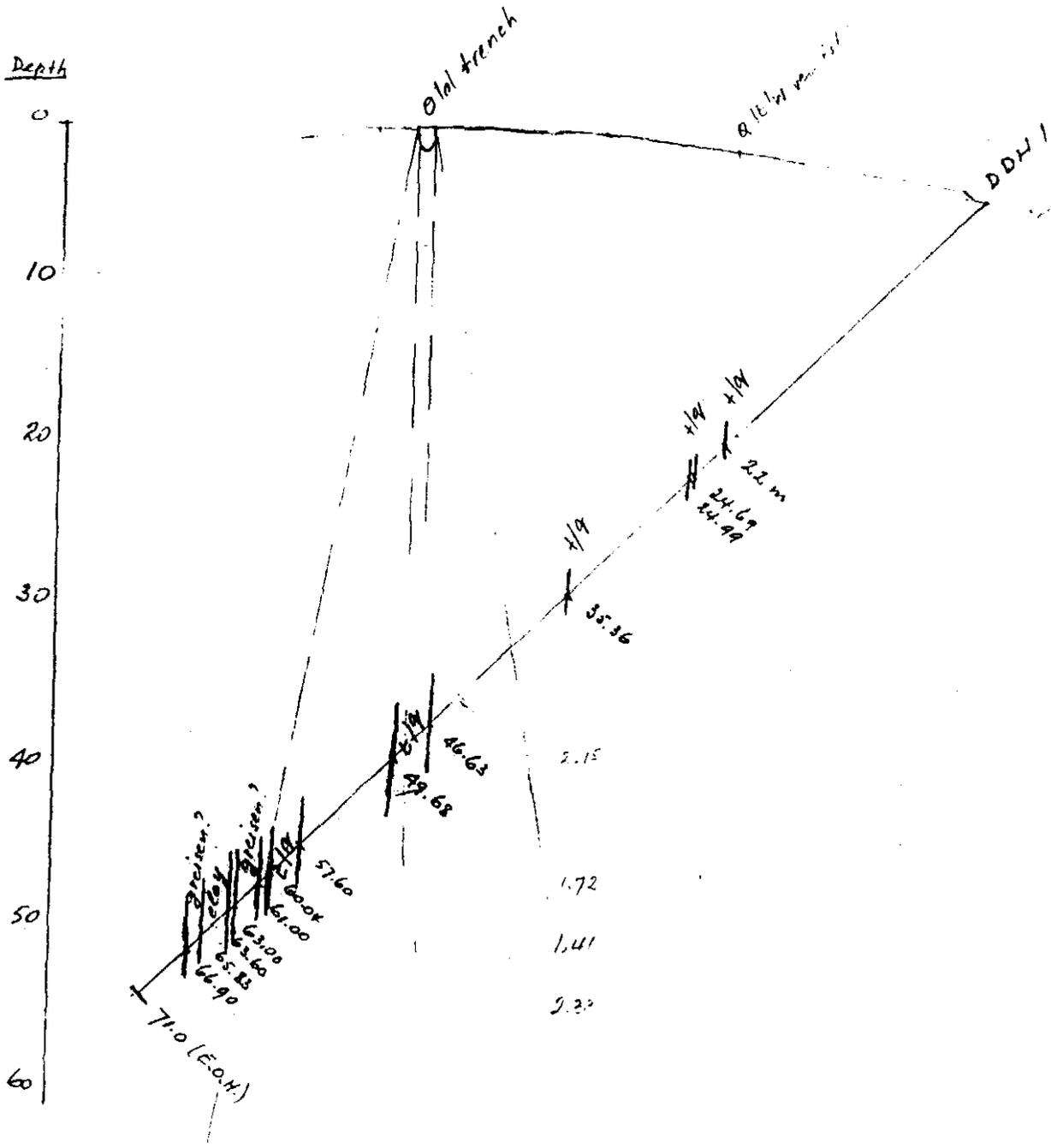
DATE COMPLETED:

INTERVAL	SAMPLE	DESCRIPTION	
61.25 to 64.43 to to to		<u>Granite</u> , slightly weathered to near-fresh. Massive core, but lowest 0.2 m. broken and oxidised with yellow mylonitic clay and on planes.	Box 10
64.43 to 70.03 to to to to to to to to		<u>Granite</u> , slightly altered, rare sulphides (yellow and white) yellow pitting, brown and white small clusters of tangled needles (tourmaline). Monotonous, fairly massive rock with minor signs of movement on some joints and thin veins of black tourmaline enrichment	Box 11
70.03 to 72.52 to to to to to to to to		<u>Granite</u> , slightly altered to almost fresh. No sulphides observed.	Box 12
		END OF HOLE	

Interview River DDH 1

Section through hole - Driller's log (9/12/84)

Scale 1:400, inclination 45°



q = quartz
 t = tourmaline
 w = wolframite

5 cm

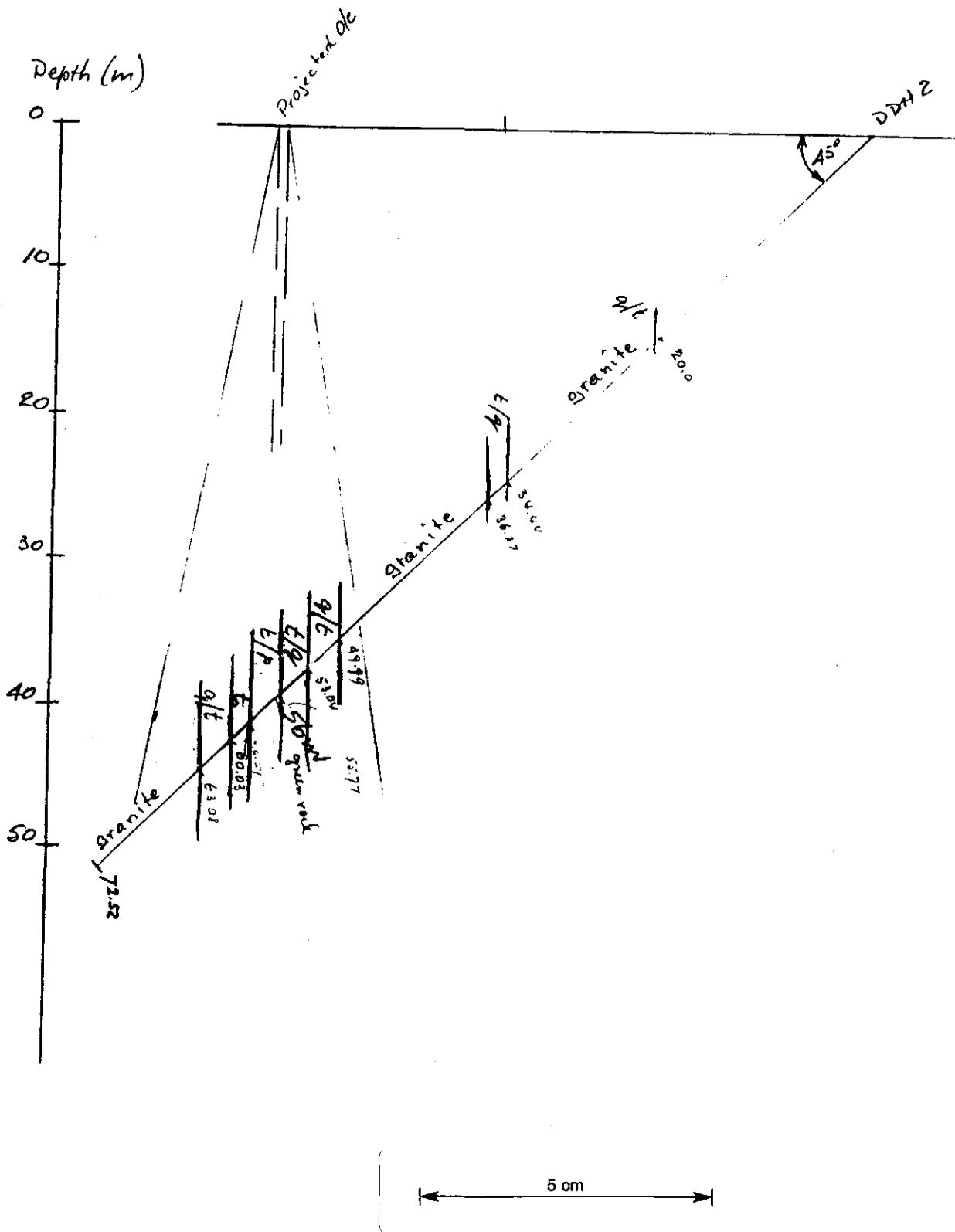
INTERVIEW RIVER - DDH 2

Driller's log

SECTION - INCLINATION OF HOLE 45° WEST

Scale 1:400 (HOR=VER)

Hole finished 18/12/84



ABIGNANO LIMITED

MINING DIVISION

FIELD LOG OF DRILL HOLE NO. DDH 1 - 45° inclination

DRILLING METHOD: N.Q. Diamond Core
 DRILLER: Peter McCormick / Gary Farrell
 LOGGER: B. Borink
 DATE COMMENCED: 28/11/84

PROJECT: INTERVIEW RIVER
 COLLAR LEVEL:
 TOTAL DEPTH: 71.00 metres.
 DATE COMPLETED: 5/12/84

INTERVAL(m)	SAMPLE	DESCRIPTION	
43.0 to 44.15	1. 43.0-44.15	Granite - altered, mainly qz. and muscovite, some biotite and hornblende altering to chlorite/epidote	Box 8
to	1. 43.0-44.15		
to			
44.15 to 48.63	2. 44.15-46.63	Granite - grey, med/coarse grained, slightly altered. A coarser, porous leached qz. rich band between 44.50 and 44.80 m. At 46.70 a small patch of mica rich material with a rosette shaped white mineral (scheelite?)	Box 8
to	3. 46.63-48.63		
to			
48.70 to 54.70		Granite - grey with black specks (biotite?) and dark diffuse patches (tourmaline?). Altered generally with green "clay" coated joint planes, which show evidence of movement. No evidence of sulphides	Box 9
to			
54.70 to 59.0		Granite - light grey, altered, med/coarse, greisenised, some chlorite coated joints. Distinct striation (schlieren?) in places (eg. 55.3 m) <u>Note:</u> core mixed up in transit. Some sulphides (chalc) in patches.	
to			

ABIGNANO LIMITED

MINING DIVISION

FIELD LOG OF DRILL HOLE NO. DDH 1

DRILLING METHOD:

PROJECT: INTERVIEW RIVER

DRILLER:

COLLAR LEVEL:

LOGGER:

TOTAL DEPTH:

DATE COMMENCED:

DATE COMPLETED:

INTERVAL	SAMPLE	DESCRIPTION	
59.0 to 59.80 to to to 59.80 to 60.00 to	4. 59.0-59.80	<u>Granite</u> - altered (greisen) with Fe, Cu, As pyrites. Abundant black mineral in central part with grey/white mineral (tourmaline + scheelite?) <u>Granite</u> - grey, altered	Box 10
60.0 to 61.40 to to to to to	5. 59.80-61.40	<u>Granite</u> - grey, altered with brownish porous patches (oxidised sulphides?) Semi-digested "pebble" at 60.8m. Core becomes increasingly pitted and rough (2%) with thin veins of green chlorite.	Box 11
61.40 to 63.66 to to to	6. 61.40-63.66	<u>Granite</u> - grey with black specks, soft and friable. Many green coated joint planes, some brownish bands. black minerals → chlorite	
63.66 to 64.80 to to	7. 63.66-64.80	Clay - gritty, grey/brown, probably extremely altered granite.	
64.80 to 65.83 to to to		<u>Granite</u> - grey, very friable, extremely altered with abundant brown specks. White mineral scheelite?	

ABIGNANO LIMITEDMINING DIVISIONFIELD LOG OF DRILL HOLE NO. DDH 2

DRILLING METHOD: *NQ coring*
 DRILLER: *P. McCormick*
 LOGGER: *B. Porink*
 DATE COMMENCED: *7/12/84*

PROJECT: *INTERVIEW RIVER*
 COLLAR LEVEL:
 TOTAL DEPTH: *72.72 m.*
 DATE COMPLETED: *18/12/84*

INTERVAL	SAMPLE	DESCRIPTION	
47.78 to 51.31		<u>C</u> granite, slightly altered, core broken in places. Medium to coarse grained granite, (qs, fclsp, muscovite, biotite, hornblende observed) fclsp slightly kaolinised. No sulphides observed.	Box 8
51.31 to 52.10	2/1	<u>C</u> granite, altered with pegmatite bands containing sulphides and wolframite? Scattered sulphides throughout interval.	
52.10 to 53.00		<u>G</u> ranite, near fresh, some sulphides on joint/slip planes.	
53.00 to 53.30	2/4	<u>C</u> granite, slightly altered.	Box 9
53.30 to 53.80	2/5	<u>C</u> granite/pegmatite, mineralised, specks of chalcocopyrite and arsenopyrite at top increasing with depth decreasing towards base. Patches of black wolframite with scheelite? veins and coarse patches of yellow chalcocopyrite, coarse muscovite and quartz, some tourmaline a light to dark purple crystalline mineral rimbed in black patch of wolframite (fluorite?)	
53.80 to 54.83	2/6	<u>C</u> granite, slightly altered, thin bands of sulphides, also scattered sulphides	

ABIGNANO LIMITEDMINING DIVISIONFIELD LOG OF DRILL HOLE NO.

DRILLING METHOD:

PROJECT:

DRILLER:

COLLAR LEVEL:

LOGGER:

TOTAL DEPTH:

DATE COMMENCED:

DATE COMPLETED:

INTERVAL	SAMPLE	DESCRIPTION	
to 54.83 to 55.77	2/7	some dull tourmaline, slicken-sided faces, some schelite? <u>Granite</u> , as above, some wolframite?	Box 9
to 55.77 to 56.71	2/8	schelite?, speck of gold? <u>Granite</u> , altered (greisenised) patches and veins of sulphides, some tourmaline, wolframite? speck of gold? Core ven. broken.	
to 56.71 to 58.60	2/9	<u>Granite</u> , slightly altered, rare sulphides, but yellow oxidised pits (after sulphides?) some brown acicular tourmaline. Thin band of tourmaline at 57.70 and faint bands of mica rich greisen! Massive core.	
58.60 to 59.33	2/2	<u>Granite</u> , slightly altered with scattered yellow sulphides, conspicuous longitudinal veins rich in sulphides with tourmaline penetrated into adjacent granite. Some wolframite? and some butter yellow sulphide (gold?) Massive core	Box 10
to			
to			
to			
59.33 to 61.25	2/3	<u>Granite</u> , slightly altered, with thin veins of 93/tourm / sulphides at 60.23 m and 61.23 m. Pits in sulphides have tarnished globular fillings. Massive core	
to			
to			
to			

ABIGNANO LIMITEDMINING DIVISIONFIELD LOG OF DRILL HOLE NO. _____

DRILLING METHOD:

PROJECT:

DRILLER:

COLLAR LEVEL:

LOGGER:

TOTAL DEPTH:

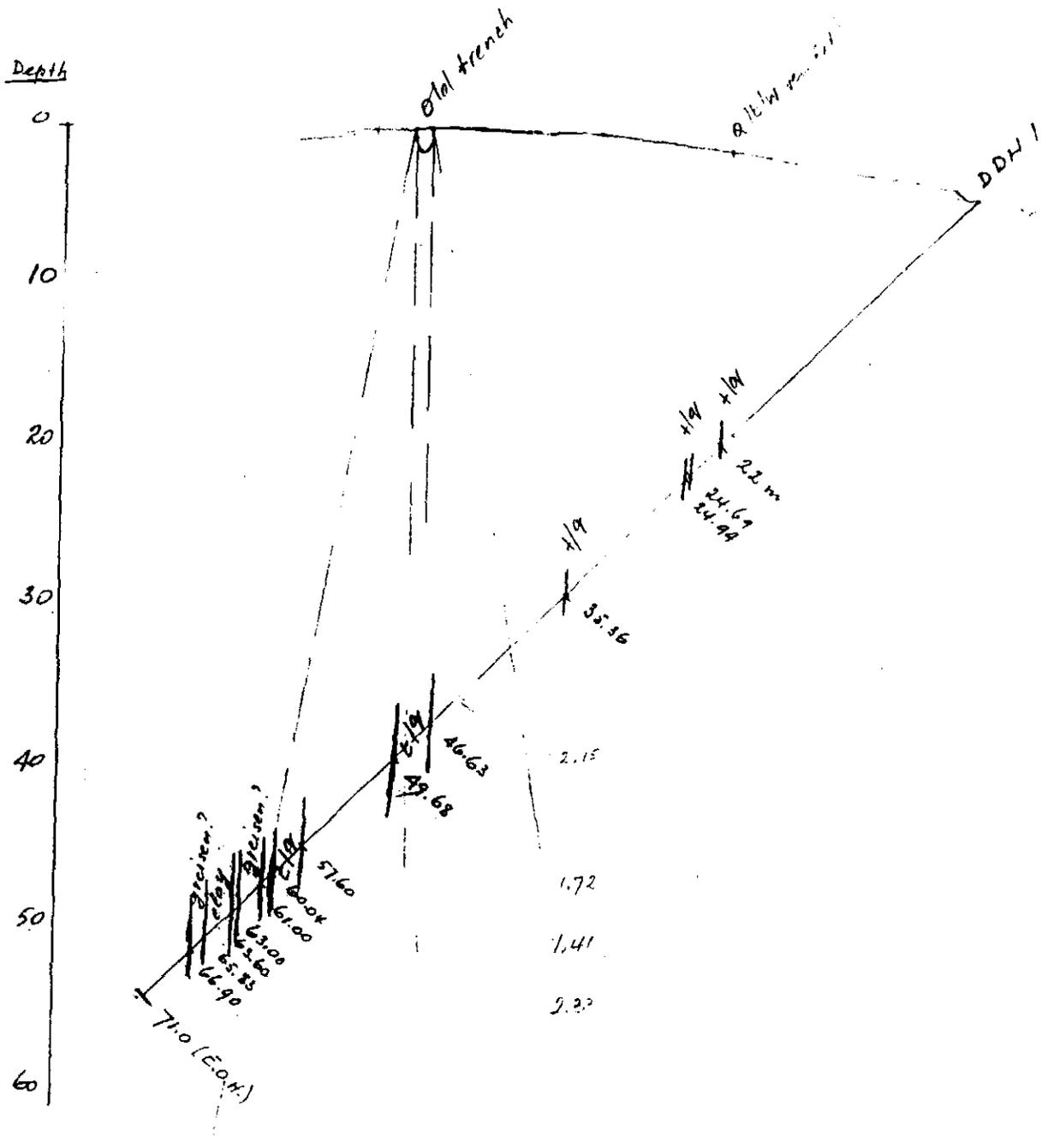
DATE COMMENCED:

DATE COMPLETED:

INTERVAL	SAMPLE	DESCRIPTION	
61.25 to 64.43 to to to		<u>Granite</u> , slightly weathered to near-fresh. Massive core, but lowest 0.2 m. broken and oxidised with yellow mylonitic clay and on planes.	Box 10
64.43 to 70.03 to to to to to to to to		<u>Granite</u> , slightly altered, rare sulphides (yellow and white) yellow pitting, brown and white small clusters of tangled needles (tourmaline.) Monotonous, fairish massive rock with minor signs of movement on some joints and thin veins of black tourmaline enrichment.	Box 11
70.03 to 72.52 to to to to to to to to		<u>Granite</u> , slightly altered to almost fresh. No sulphides observed.	Box 12
		END OF HOLE	

Section through hole - Driller's log (9/12/64)

Scale 1:400, inclination 45°



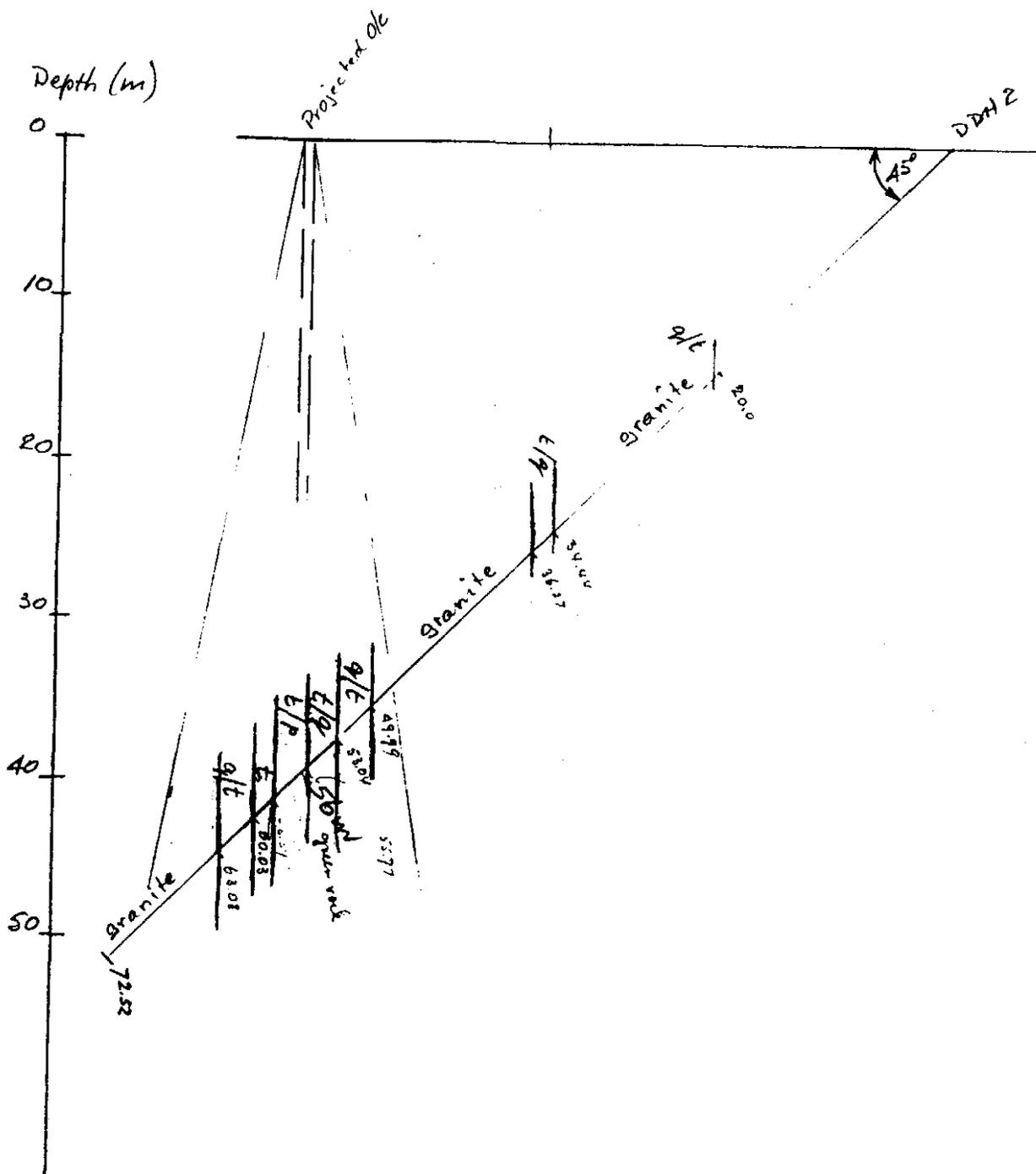
INTERVIEW RIVER - DDH 2

Driller's log

SECTION - INCLINATION OF HOLE 45° WEST

Scale 1:400 (HOR=VER)

Hole finished 18/12/84



5 cm

ABIGNANO LIMITED

MINING DIVISION

FIELD LOG OF DRILL HOLE NO. DDH 1 - 45° inclination

DRILLING METHOD: N.Q. Diamond Core
 DRILLER: Peter McCormick / Gary Farrell
 LOGGER: B. Porink
 DATE COMMENCED: 28/11/84

PROJECT: INTERVIEW RIVER
 COLLAR LEVEL:
 TOTAL DEPTH: 71.00 metres.
 DATE COMPLETED: 5/12/84

INTERVAL(m)	SAMPLE	DESCRIPTION	
43.0 to 44.15	1. 43.0-44.15	Granite - altered, mainly qz. and muscovite, some biotite and hornblende altering to chlorite/epidote	
to	1. 43.0-44.15		
to			
44.15 to 48.63	2. 44.15-46.63	Granite - grey, med/coarse grained, slightly altered. A coarser, porous leached qz. rich band between 44.50 and 44.80 m. At 46.70 a small patch of mica rich material with a rosette shaped white mineral (scheelite?)	Box 8
to	3. 46.63-48.63		
to			
48.70 to 54.70		Granite - grey with black specks (biotite?) and dark diffuse patches (tourmaline?) Altered generally with green "clay" coated joint planes, which show evidence of movement. No evidence of sulphides	Box 9
to			
54.70 to 59.0		Granite - light grey, altered, med/coarse, greisenised, some chlorite coated joints. Distinct striation (schlieren?) in places (eg. 55.3 m) <u>Note:</u> core mixed up in transit. Some sulphides (chalc) in patches.	
to			

ABIGNANO LIMITED

MINING DIVISION

FIELD LOG OF DRILL HOLE NO. DDH 1

DRILLING METHOD:

PROJECT: INTERVIEW RIVER

DRILLER:

COLLAR LEVEL:

LOGGER:

TOTAL DEPTH:

DATE COMMENCED:

DATE COMPLETED:

INTERVAL	SAMPLE	DESCRIPTION	
59.0 to 59.80 to to to 59.80 to 60.00 to	4. 59.0-59.80	Granite - altered (greisen) with Fe, Cu, As pyrites. Abundant black mineral in central part with grey/white mineral (tourmaline + schelite?) Granite - grey, altered	Box 10
60.0 to 61.40 to to to to to	5. 59.80-61.40	Granite - grey, altered with brownish porous patches (oxidised sulphides?) Semi-digested "pebble" at 60.8m. Core becomes increasingly pitted and rough (sg) with thin veins of green chlorite.	
61.40 to 63.66 to to to	6. 61.40-63.66	Granite - grey with black specks, soft and friable. Many green coated joint planes, some brownish bands. black minerals → chlorite	Box 11
63.66 to 64.80 to to	7. 63.66-64.80	Clay - gritty, grey/brown, probably extremely altered granite.	
64.80 to 65.83 to to to		Granite - grey, very friable, extremely altered with abundant brown specks. White mineral schelite?	

ABIGNANO LIMITED

MINING DIVISION

FIELD LOG OF DRILL HOLE NO. DDH 1

DRILLING METHOD:

DRILLER:

LOGGER:

DATE COMMENCED:

PROJECT: INTERVIEW RIVER.

COLLAR LEVEL:

TOTAL DEPTH:

DATE COMPLETED:

INTERVAL	SAMPLE	DESCRIPTION	
65.83 to 67.34 to to to to	8. 64.80-67.34	<u>Granite</u> - friable and broken, black specks, some pink colouration Green coated joints and altered biotite/hornblende/tourmaline? Very fine speck of gold?	Box 12
67.34 to 69.17 to to	9. 67.34-69.17	<u>Granite</u> - altered, fairly friable and broken. Some pink and green colouration.	
69.17 to 71.00 to to to to		<u>Granite</u> - altered, as above, 2 cm thick band? of hard, dark green vitreous mineral (hornblende?) Some evidence of very fine sulphides towards base.	
to to to to to to to to to to		END OF HOLE	

ABIGNANO LIMITEDMINING DIVISIONFIELD LOG OF DRILL HOLE NO. DDH 2

DRILLING METHOD: *NQ coring*
 DRILLER: *P. McCormick*
 LOGGER: *B. Porink*
 DATE COMMENCED: *7/12/84*

PROJECT: *INTERVIEW RIVER*
 COLLAR LEVEL:
 TOTAL DEPTH: *72.72 m.*
 DATE COMPLETED: *18/12/84*

INTERVAL	SAMPLE	DESCRIPTION	
47.78 to 51.31 to to to to		<u>Granite</u> , slightly altered, core broken in places. Medium to coarse grained granite, (qz, feldspar, muscovite, biotite, hornblende observed) feldspar slightly kaolinised. No sulphides observed.	Box 8
51.31 to 52.10 to to	2/11	<u>Granite</u> , altered with pegmatite bands containing sulphides and wolframite? Scattered sulphides throughout interval.	
52.10 to 53.00 to to		<u>Granite</u> , near fresh, some sulphides on joint/slip planes.	
53.00 to 53.30 to to to to to to to	2/4 2/5	<u>Granite</u> , slightly altered. <u>Granite/pegmatite</u> , mineralised, specks of chalcopyrite and arsenopyrite at top increasing with depth decreasing towards base. Patches of black wolframite with scheelite? veins and coarse patches of yellow chalcopyrite, coarse muscovite and quartz, some tourmaline A light to dark purple crystalline mineral rimbed in black patches of wolframite (fluorite?)	Box 9
53.80 to 54.83 to	2/6	<u>Granite</u> , slightly altered, thin bands of sulphides, also scattered sulphides	

ABIGNANO LIMITEDMINING DIVISIONFIELD LOG OF DRILL HOLE NO.

DRILLING METHOD:

PROJECT:

DRILLER:

COLLAR LEVEL:

LOGGER:

TOTAL DEPTH:

DATE COMMENCED:

DATE COMPLETED:

INTERVAL	SAMPLE	DESCRIPTION	
to 54.83 to 55.77	2/7	some dull tourmaline, slickensided faces, some schuchite? <u>Granite</u> , as above, some wolframite?	Box 9
to 55.77 to 56.71	2/8	schuchite?, speck of gold? <u>Granite</u> , altered (greisenised) patches	
to		and veins of sulphides, some	
to		tourmaline, wolframite? speck	
56.71 to 58.60	2/9	of gold? Core ven. broken. <u>Granite</u> , slightly altered, rare sulphides,	
to		but yellow oxidised pits (after sulphides?) some brown acicular	
to		tourmaline. Thin band of	
to		tourmaline at 57.70 and	
to		faint bands of mica rich	
to		greisen? Massive core	
58.60 to 59.33	2/2	<u>Granite</u> , slightly altered with scattered	Box 10
to		yellow sulphides, conspicuous	
to		longitudinal vein rich in	
to		sulphides with tourmaline	
to		penetrated into adjacent granite	
to		Some wolframite? and some	
to		broken yellow sulphide (gold?)	
to		Massive core	
59.33 to 61.25	2/3	<u>Granite</u> , slightly altered, with thin	
to		veins of 93/tourm / sulphides	
to		at 60.23 m and 61.23 m.	
to		Pits in sulphides have tarnished	
to		globular fillings.	
to		Massive core	

ABIGNANO LIMITED

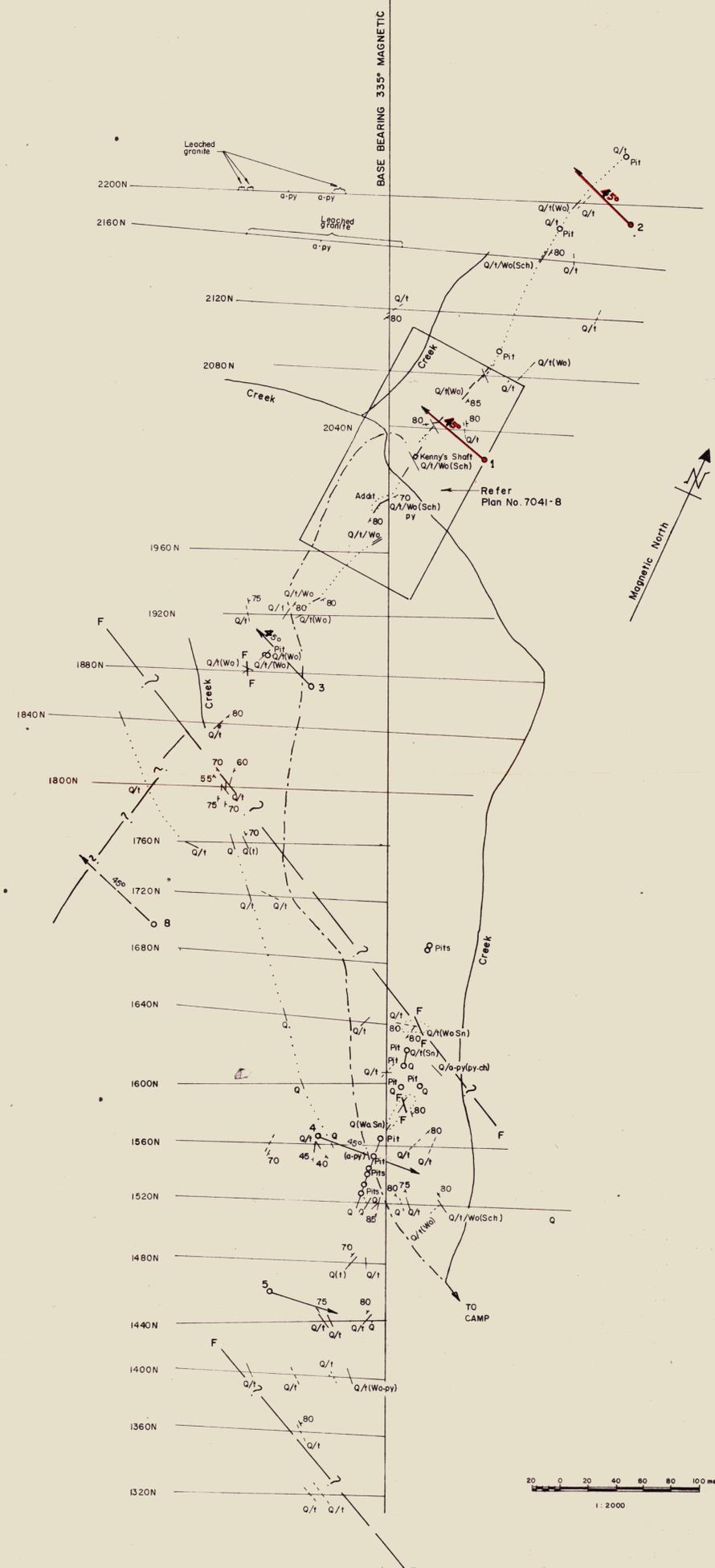
MINING DIVISION

FIELD LOG OF DRILL HOLE NO. _____

DRILLING METHOD:
 DRILLER:
 LOGGER:
 DATE COMMENCED:

PROJECT:
 COLLAR LEVEL:
 TOTAL DEPTH:
 DATE COMPLETED:

INTERVAL	SAMPLE	DESCRIPTION	
61.25 to 64.43 to to to		<u>Granite</u> , slightly weathered to near-fresh. Massive core, but lowest 0.2 m. broken and oxidised with yellow mylonitic clay and on planes.	Box 10
64.43 to 70.03 to to to to to to to to		<u>Granite</u> , slightly altered, rare sulphides (yellow and white) yellow pitting, brown and white small clusters of tangled needles (tourmaline.) Monotonous, fairly massive rock with minor signs of movement on some joints and thin veins of black tourmaline enrichment.	Box 11
70.03 to 72.52 to to to to to to to to		<u>Granite</u> , slightly altered to almost fresh. No sulphides observed.	Box 12
		END OF HOLE	



- LEGEND:**
- 1840N — New trenches
 - - - Veins less than 10 cm wide
 - Veins greater than 10 cm wide
 - ... Possible extension of vein
 - 85 Strike and dip of vein
 - Q - Quartz
 - t - tourmaline
 - py - pyrite
 - a-py - arsenopyrite
 - ch-py - chalcopyrite
 - sch - scheelite
 - Wo - wolfronite
 - Sn - tin
 - Vein system showing wolfram
 - Shaft
 - Pit
 - Old trenching, bulldozed areas
 - - - Old vehicle tracks
 - F - F Fault Interpretation
 - 20 55° Proposed Drill Hole with Inclination
 - 45° Holes Drilled (Dec.'84)
- Note: All veins without dip marked are vertical

245057

5 cm

1:2000

A Abignano Limited
 Civil Engineering Contractors
 19-21 Bridge Street
 Pymble NSW 2073
 Telephone 02-4493344
 Telex 244604

INTERVIEW RIVER E.L. 13/81 TASMANIA	SCALE 1:2000	DATE 25-2-82
KENNY'S GRID GEOLOGICAL INVESTIGATION AND DRILL SITES	DRAWN P.G.	DRAWING NUMBER 7041-2
	TRACED J.P.	
	CHECKED G.B.B.	