TASMANIA COPPER PROJECT

PHASE III - FINAL REPORT

MACQUARIE HARBOR
TASMANIA, AUSTRALIA

AMG REFERENCE POINTS ADDED

CITCO INTERNATIONAL MINERALS COMPANY
TULSA, OKLAHOMA

October 11, 1976

TASMANIA COPPER PROJECT

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MACQUARIE HARBOR

TASMANIA, AUSTRALIA

Douglas D. Jinks
October 11, 1976

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INTRODUCTION, CONCLUSIONS AND RECOMMENDATIONS

This is the final report on Phase III activities of the Tasmania Copper Project. It covers the full scale gravity process investigations and the pro forma economic evaluations based on these investigations. It also expands on the March 8 1976 Status Report.

Although a short review of the Project is included, for a fuller explanation, reference is made to the November 24, 1974 report, "Summary Report Metalliferous Muds Project, Macquarie Harbor, Tasmania, Australia", and to subsequent status reports.

Phase III studies lead to the conclusion that copper production from Macquarie Harbor is technically feasible but only marginally economic. A project P.I. of 13.7% is indicated based on a set of relatively conservative assumptions of capital and operating costs and commodity prices. A second conclusion, of no less importance, is that economic utilization of the pyrite content of the deposit is critical to project success.

Because of the marginal economics, the fact that sulfur rather than copper is the critical commodity, the new technology involved, the remoteness of the area, and the relatively small size of the project, we do not recommend proceeding further on project development. However, we do recommend that we dispose of our 94% interest in the project to a potentially interested Australian company or group on the best terms available. If a buyer cannot be found at the present time, we recommend maintaining the leases, which can be done at a very low cost, pending future improvement in the demand-supply relationship for sulfur (pyrites) and/or copper.

BACKGROUND

Sediments accumulating in a "Black Shale" environment, which is characterized by the accumulation of fine grained sediments under reducing conditions, are known to concentrate metals. Utilizing this concept, Mr. S.R.M. Harvey interested a group of New York/New Jersey businessmen in financing an early sampling program in Macquarie Harbor, Tasmania, Australia (Figure 1) where Mr. Harvey had concluded that a "Black Shale" environment was operative.

A partnership (HOLB) later organized into a corporation (Aberdare) was formed and a limited sampling program, identified as Phase I, was carried out. This confirmed that metal values, primarily copper, were present in the Macquarie Harbor sediments in amounts great enough to justify further investigation. CITCO International Minerals (formerly Cities Service International) after evaluating these data, entered into an agreement with Aberdare (see Appendices) to finance additional sampling and a limited laboratory scale process investigation, Phase II, under which CITCO would receive a 94% interest in the mineral leases and operations and would assume management of the Project while Aberdare would retain a 6% interest.

Phase II field work consisted of sampling the sediments on 6000 foot spacing. A probable reserve of approximately 125 million tons of sediment with 0.16% copper content and a possible reserve of an additional 167 million tons with 0.1% copper were indicated. Cobalt, molybdenum, zinc, and silver content would raise the average grade to about 0.25% copper equivalent in the probable reserve area.

Phase II laboratory tests showed that standard gravity techniques were unsuitable and conventional flotation too expensive. However, additional investigation indicated that bulk flotation combined with gravity separation techniques used in the mineral sands industry might recover sufficient metal at a cost low enough to make the project economically attractive. It was recognized at this early stage that the success of the project was dependent on inexpensively producing a bulk sulfide concentrate suitable for roasting and hydrometallurgical extraction of metal values.

The process scheme which evolved can be divided into four (4) major sections; 1) mining (suction dredging), 2) preconcentration (gravity separation), 3) final concentration (flotation), and 4) hydrometallurgy. A flowsheet is included as Figure 4. The marketable products were to be either metals, probably cathode copper and/or copper chemicals.

Pro forma economic analyses based on Phase II work indicated that a 25,000 ton per day operation with a recoverable grade of 0.15% copper could be economic. The analyses indicated that an operation of this size could return 2-3 million dollars annually after taxes to Cities Service assuming an equity interest of 49% to conform to Australian mining regulations, a capital investment of 7 million dollars, and a 70:30 debt:equity ratio.

Phase II work was encouraging enough for CITCO to exercise its option to begin Phase III and assume full project management.

The objectives of Phase III were 1) to prove the reserves by grid drilling, and 2) to develop and prove the concentration process using full scale equipment. If the results of this work still indicated technical and economic feasibility, the last part of Phase III would have been the erection and operation of a pilot plant on site.

In an effort to circumvent problems in a preliminary feasibility study, an engineering contractor, Arthur G. McKee, was contacted early in the project. McKee provided engineering personnel at the Macquarie Harbor site to observe the drilling and sampling and to evaluate the area for plant location, water resources and tailing disposal sites. The process studies were frequently discussed with the McKee personnel to keep them informed of all phases of activites.

A preliminary feasibility study was not commissioned due to the failure of the preconcentration tests to concentrate the metals into the sulfide fraction.

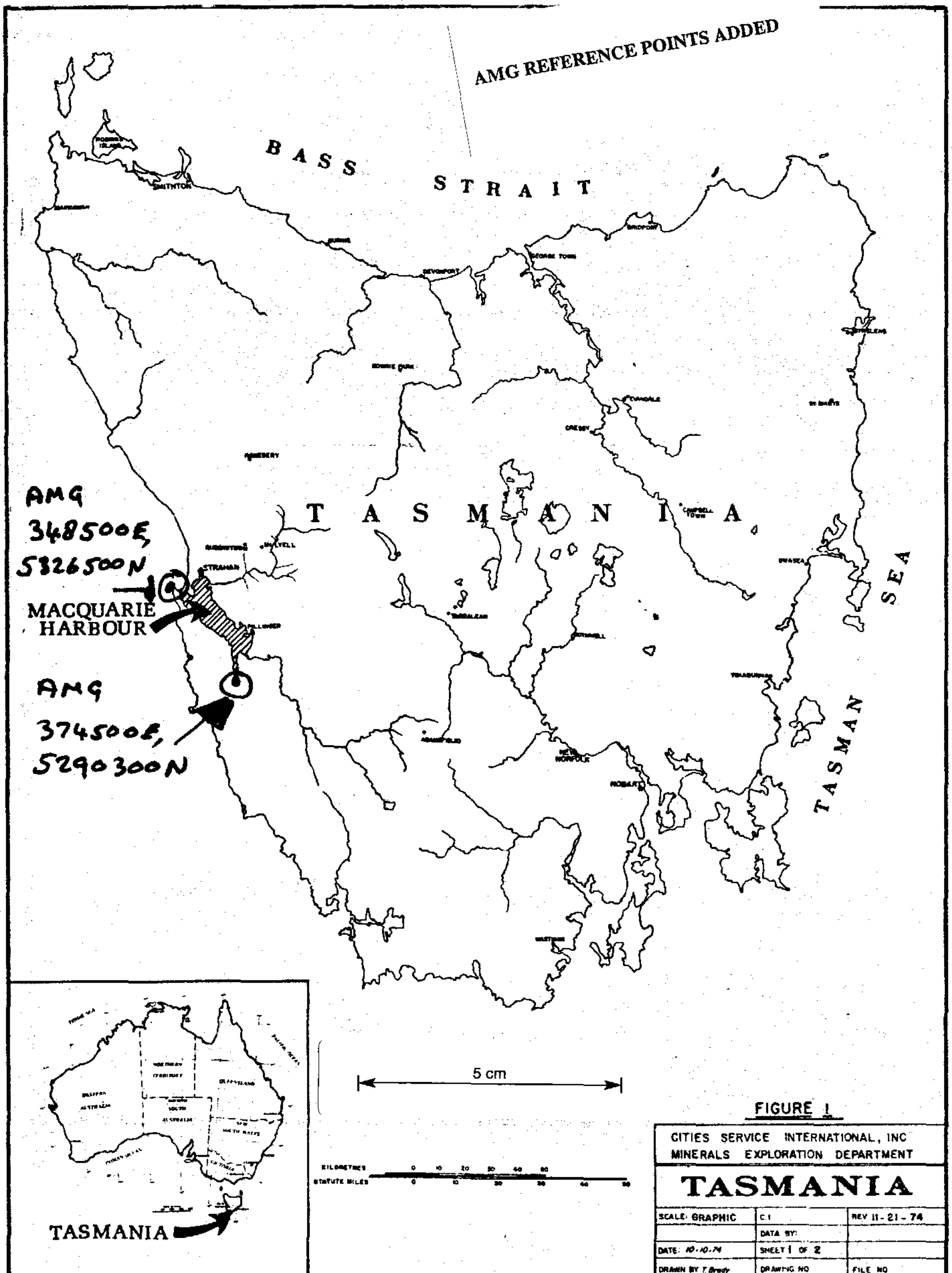


FIGURE I

CITIES SERVICE INTERNATIONAL, INC MINERALS EXPLORATION DEPARTMENT		
TASMANIA		
SCALE: GRAPHIC	C.I	REV 11-21-74
	DATA BY:	
DATE: 10-10-74	SHEET 1 OF 2	
DRAWN BY T. Brady	DRAWING NO	FILE NO

ECONOMIC GEOLOGY

The Tasmania copper project is based on the premise that present day depositional environments exist in which conditions approximate those in the paleo-environment under which "Black Shale" sediments formed. "Black Shales" commonly contain relatively high concentrations of metals, such as copper, zinc, lead, molybdenum, and silver. In specific formations these concentrations have been high enough to exploit. The Kupfershiefer deposit of central Europe is a well known example. It is therefore logical to assume that an investigation of present day "Black Shale" environments might yield sediments with metal values that although sub-economic if considered in light of conventional hardrock mining and processing costs, might prove economic if the costs of mining and processing were greatly reduced.

The "Black Shale" environment is characterized by anerobic or reducing conditions. Under these conditions organic matter accumulates, only partially decomposing and releasing H_2S which precipitates metals as sulfides. These conditions are operative today in the Black Sea where bottom muds contain unusually high concentrates of metals as sulfides. They also exist in selected coastal embayments around the world. The coastal embayments must also have restricted circulation with the open ocean and a mineralized area whose drainage empties directly into the bay; the former promotes reducing conditions, the latter provides the metals source.

The above conditions are well developed at Macquarie Harbor, an area of 148 square miles located at 145^o 25' and 42^o 15' on the west coast of the state of Tasmania, Australia (See Figure 1) Given sufficient time and favorable conditions of burial and diagenesis, sediments presently accumulating in Macquarie Harbor could develop into a "Black Shale". The harbor area is covered by Exploration Licenses 16/73 and 2/74 granted to Aberdare and assigned to

Elisna Pty. Ltd., the operating company formed to develop the project. See Appendices for copies of licenses and Cities Service/Aberdare contracts and ammendments.

The deposit consists of detrital sediments mainly resulting from mining by Mt. Lyell Mining and Railway Company at Queenston. Mt. Lyell, after mining a quartz-sericite-chlorite schist (an altered potash rich rhyolite) and producing a copper concentrate consisting mainly of chalcopyrite and bornite, disposes of tailings in the Queen River. The tailings, mainly pyrite, quartz, and micaceous material (chlorite and sericite), are transported by the Queen to the King River and eventually deposited in typical deltaic fashion in Macquarie Harbor. As the sediment is moved downstream it is progressively sorted with the coarsest and heaviest portions of the sediment deposited as bars in the river and on the delta and the finer portions deposited out in the bay.

The King River Delta has grown dramatically within recent time. The mineralized portion of the delta which accumulated in the last fifty years is underlain by an old delta composed of barren sands in the near shore area and by barren marine clays in the seaward areas.

The principal ore minerals are derived in two ways; 1) primarily from Mt. Lyell's tails and 2) in minor amounts from the direct precipitation of metal sulfides by hydrogen sulfide. Pyrite constitutes the major portion of the sulfides. Chalcopyrite is the principle copper mineral. Chalcocite and bornite are present in minor amounts. Sphalerite was identified. Cobalt is associated with the pyrite. Gangue minerals which account for 90%+ of the ore material are principally quartz, mica and detrital rock fragments containing the above plus a suite of heavy minerals. In certain areas small amounts of copper bearing slag are present.

The infrastructure in the area is good. Power is readily available from a hydroelectric complex being developed in the central part of the island. Transportation of equipment and materials and shipment of products can be made by low cost ocean freight. Skilled labor is available from the large Mt. Lyell mining district 30 miles to the northeast. Housing, shopping, schools, and etc. are readily available in Strahan and Queenston. Fresh process water is available from the King River drainage system. Land suitable for a plant site is available adjacent to the mining operation.

As in most of the developed world, environmental considerations will play a major role in the development of any mining operation in Australia. Early in the project contact was made with appropriate environmental officials on both the state and federal level. Although concern was expressed, no serious objections were raised as long as the project plan insured maintenance of air and water quality standards and protection of wildlife from major damage.

It was recognized by the environmental officials that mining of the mineralized sediments in Macquarie Harbor would in reality enhance the environment in the harbor and the King River by removing an objectional acid producing waste originating from the Mt. Lyell mining operation at Queenston. The removal of the sediments from the floor of the bay and the disposal of the clean tails on the bottom of the bay would not adversely effect the ecological balance as the bottom of the bay is euxenic and therefore devoid of life. Although a small fishing fleet operates out of Strahan, no commercial fishing is possible in the Harbor.

In compliance with the regulations, it was agreed that a preliminary environmental impact study would be made by an outside contractor. Proposals were requested and received from Robertson Research (Australia) and Davy Ashmore.

To avoid possible problems with local experts, it was stipulated that wherever possible local expertise would be utilized in the study. A study was not commissioned due to the failure of the full scale preconcentration tests to make an acceptable product and the resulting marginal economics of the project.

The government welcomes industry to the west coast of Tasmania as the area is considered depressed due to restriction on the use of shark (flake) for human consumption by many of the Australian states. The area is covered by a nearly impenetrable humid temperate rain forest and is generally unsuitable for major agricultural enterprise.

PHASE III TONNAGE AND GRADE DETERMINATION

Development drilling to establish tonnage and grade was completed during the first quarter of 1976. Drilling was accomplished in two stages, deep water drilling on contract by W. L. Sides and Sons drilling contractors, Melbourne, Australia and delta and river drilling by Australia-Cities Service with a company owned rig operated by Kitching Drilling Consultants.

Due to the unconsolidated nature of the sediments a reverse circulation drilling technique was utilized. This technique was developed for and proved by the Australian mineral sands industry. To insure representative samples for assay a rotating sample splitter was attached directly to the output stream of the drill. Assay for copper, zinc, cobalt, molybdenum and silver were done on 3 meter intervals by Geomin, a Perth, Australia based analytical laboratory. see plan in pocket

The drilling pattern is shown in Figure 2. A total of 252 sites were drilled, divided into 43 sites in deep water, 111 sites on the delta, and 98 sites in the King River. Total footage drilled amounted to 4480 feet (1358 meters).

Computer reserve calculations indicate a proven reserve of 100 million tons plus an indicated reserve of an additional 25 million tons. The delta portion of the deposit accounts for 50 million tons of the proven reserve. The weighted average grade of the entire deposit is 0.19% copper equivalent. The weighted average of the deltaic sediments is 0.25 copper equivalent and of the offshore sediments is 0.15 copper equivalent. Copper accounts for approximately 60% of the contained metal values with cobalt, zinc and molybdenum accounting for the remainder. A small amount of potentially recoverable silver is present. The deposit also contains 3.5% sulfur primarily in the form of pyrite.

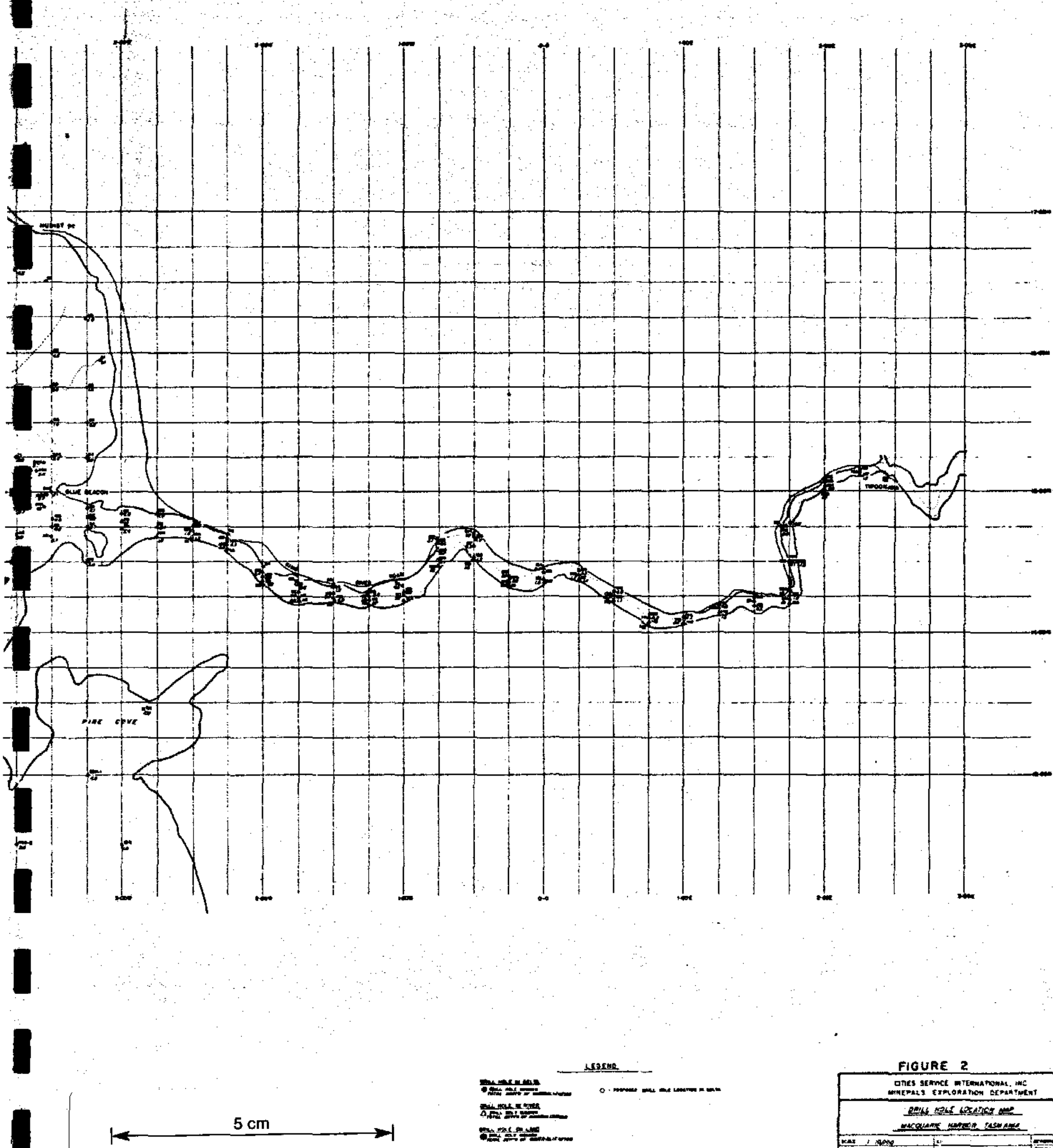


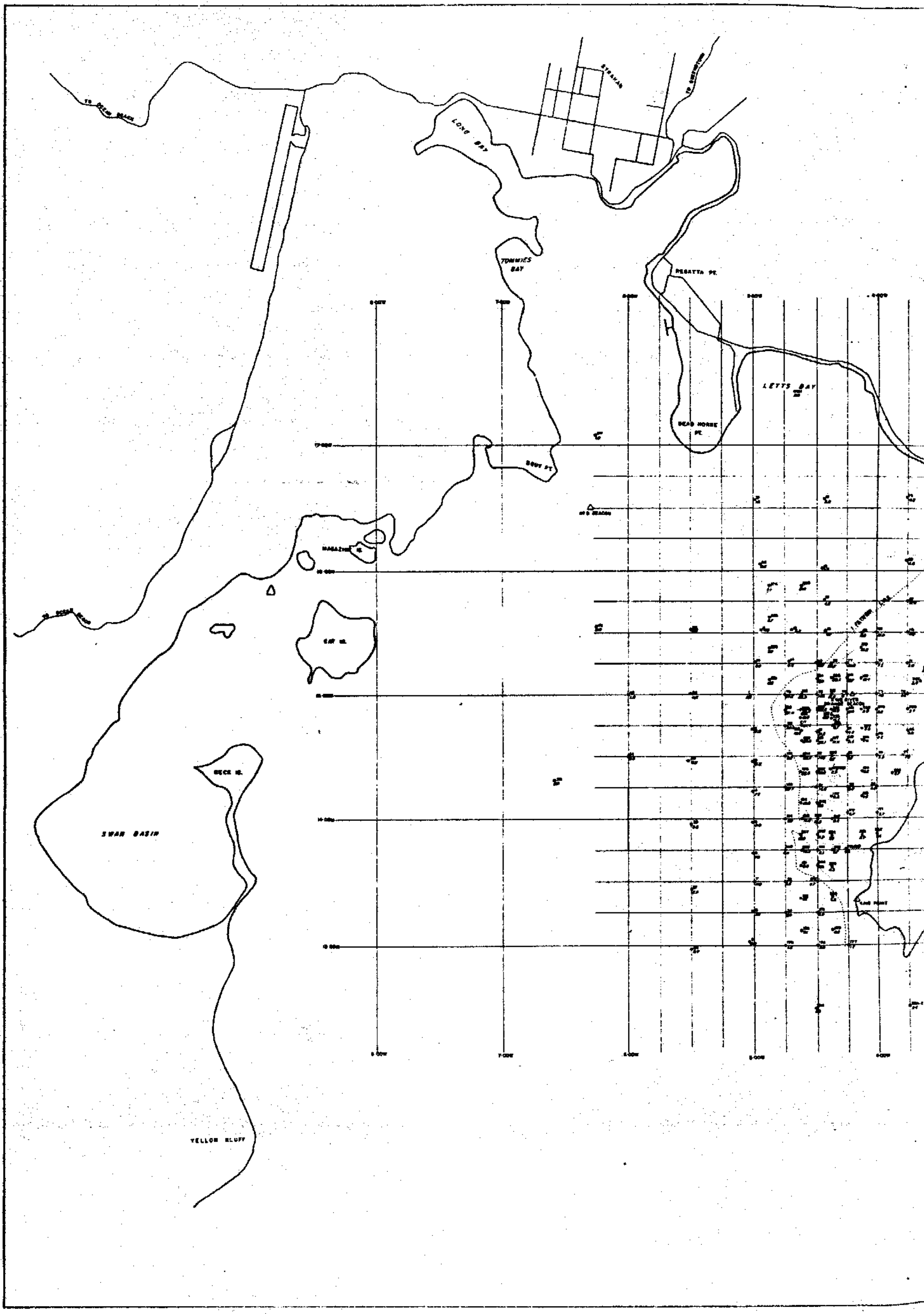
FIGURE 2

CITIES SERVICE INTERNATIONAL, INC
MINERALS EXPLORATION DEPARTMENT

DRILL HOLE LOCATION MAP

MACQUARIE HARBOR, TASMANIA

SCALE 1:10,000



TO OCEAN BEACH
STRAHAN
TO QUEENSTOWN
LONG BAY
TOMMIES BAY
REGATTA PT.
LETTS BAY
DEAD HORSE PT.
BOAT PT.
MAGAZINE IS.
CAT IS.
TO OCEAN BEACH
SWAN BASIN
YELLOW BLUFF

The drilling confirmed that the King river does not contain significant quantities of mineralized sediment. A maximum of 2 million tons is all that is present in the river channel at any point in time. The lack of significant reserve in the river is explained by the periodic flooding which results in the cleaning of accumulated sediments out of the river and depositing them in the delta.

The distribution of the metal in the sediments can be seen in Figure 3 expressed as kilograms of copper equivalent per ton. Highest metal values exist in the coarser exposed delta portion of the deposit with copper values in specific areas exceeding 0.2% and copper equivalent values exceeding 0.25%. Metal values decrease as the sediments get progressively finer away from the front of the delta. Values for copper range in the neighborhood of 0.1% with copper equivalent values ranging up to 0.15% . As would be expected from the origin of the mineralization, there appears to be a relationship between particle size and total metal values in the sediment. The pyrite/chalcopyrite and other compound grains being heavier tend to settle out in the sedimenting system more readily than the lighter micaeous fraction which contains only minor amounts of metal.

A comparison of the particle size distribution is shown in Table 1.

Table 1 Comparison of Particle Size distribution of Deltaic and Offshore Sediments in Macquarie Harbor.

Size Fraction	Deltaic		Offshore	
Microns	% Fraction	Cum. %	% Fraction	Cum %
+300	7.5	7.5	-	-
+210	21.2	28.7	0.2	0.2
+150	29.3	58.0	3.3	3.5
+105	23.3	81.3	22.1	25.6
+ 75	10.0	91.3	29.7	55.3
- 75	8.7	100.0	44.7	100.0
	D50 ≈ 165μ		D50 ≈ 80μ	


KING RIVER
CALCULATED RESERVE BOUNDARY
0 250 500 750
METERS
CITCO INTERNATIONAL
MINERALS COMPANY
COPPER EQUIVALENT
(KILOGRAMS PER METRIC TON)
MACQUARIE DELTA, TASMANIA
SCALE METERS
C.I.
REVISIONS
CONTOUR INTERVAL
DATA BY D D J
DATE 9-27-76
SHEET OF
DRAWN BY J Brody
DRAWING NO.
FILE NO.
5 cm

FIGURE 3

PHASE III PROCESS INVESTIGATIONS

Preliminary process studies carried out by Hazen Research, Golden, Colorado, the Colorado School of Mines Research Institute, Golden, Colorado, the Australian Mineral Development Laboratories (AMDEL), Adelaide, Australia, and the Tasmanian Department of Mines Metallurgical Laboratory, Launceston, Australia, although marginally successful, ruled out conventional concentration techniques as being uneconomical. Bench scale work by Mineral Development International, Jacksonville, Florida, however indicated that gravity techniques based on the principle of laminar flow did appear to have applicability. These techniques are not normally utilized in base metal operations. The pieces of equipment that operate on the laminar flow principle are the Reichert Cone and the Lamflo Sluice, both of which require full scale testing to be definitive. Additional flotation work by Lakefield Research, Lakefield, Ontario, Canada demonstrated that an acceptable sulfide concentrate could be economically produced utilizing tank type flotation techniques.

Based on these results, testing was contracted with Mineral Deposits, Ltd, Southport, Australia, AMDEL, and Lakefield Research. The results of these tests will be discussed in the sections below. All testing was accomplished on material collected during the drilling program. The sample was considered representative of the coarser portion of the deposit. It was assumed that if the coarse high grade material could not be economically processed the project would not be viable.

The process flowsheet, as devised from the preliminary bench work can be divided into four major sections: 1) dredging, 2) preconcentration, 3) final concentration, and 4) hydrometallurgy. The preconcentration was to be made with the Reichert Cone and/or Lamflo Sluice. The final concentration was to be made with tank type flotation techniques. The hydrometallurgy, consisting of roasting, leaching, extraction, and electrowinning was envisioned as a turnkey purchase of Davy-Powergas, Arthur G. McKee or a similar technology/ construction package. Figure 4 outlines the proposed flowsheet.

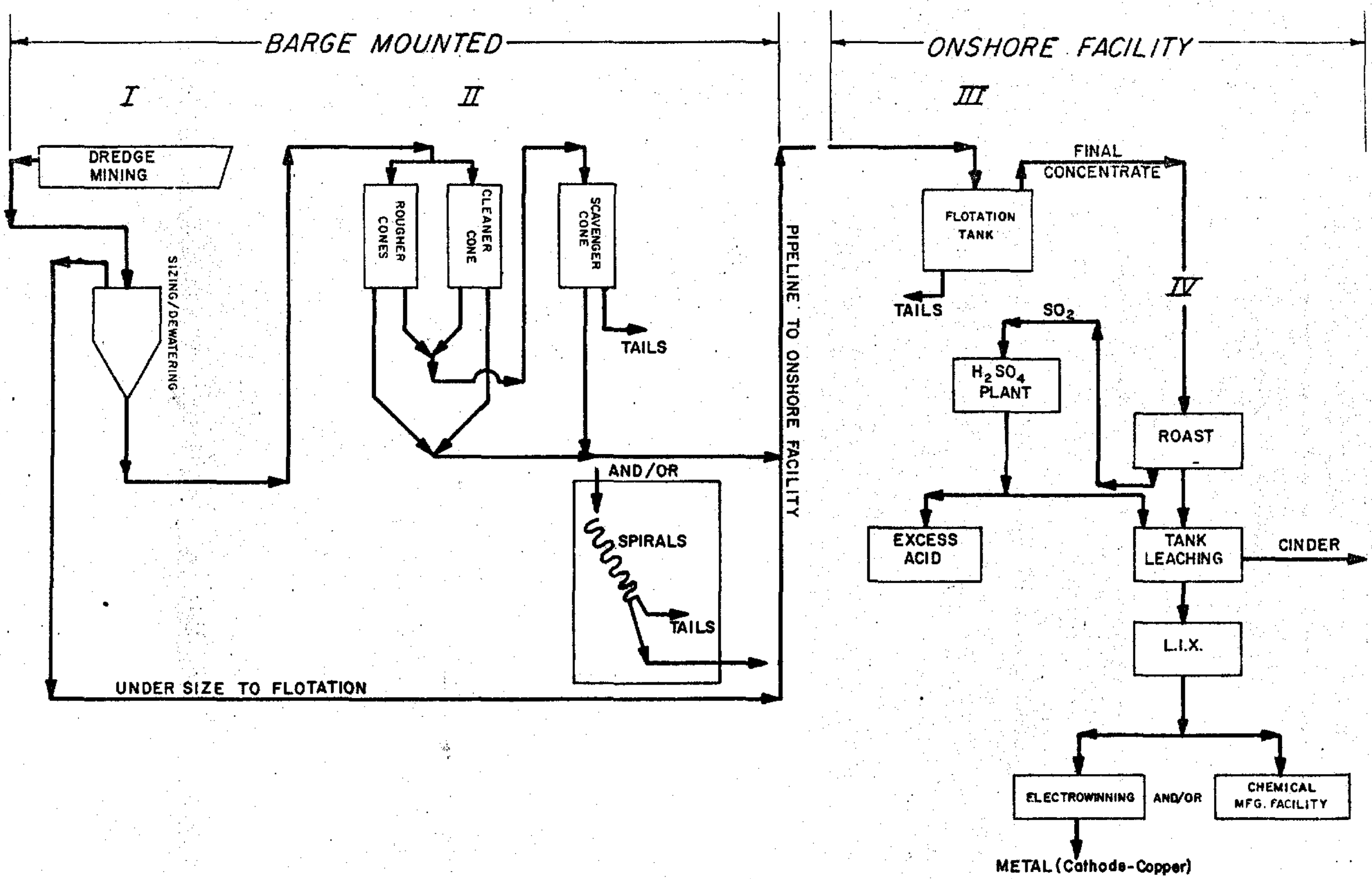


FIGURE 4 Proposed flowsheet for processing the Tasmania Sediments.

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Section I - Mining

Section I involves the mining of the metalliferous sediments using suction dredging techniques, the de-watering and sizing of the sediment with the minus 44 micron (-325 mesh) fraction being returned to the floor of bay and the plus 44 micron (+325 mesh) fraction either pumped to an onshore processing facility or to a trailing barge for gravity concentration prior to being pumped to the onshore facility.

The Macquarie Harbor sediments lend themselves to a low cost cutter-head suction dredging operation. Discussions held with Ellicott Machine Corp., Baltimore, Maryland; American Marine Co., Nashville, Tennessee; and Australian Dredging, Pty., Ltd., Melbourne, Australia can be summarized as follows: Under the mining conditions as presently exist in Macquarie Harbor, there are no serious problems forseen in raising up to 40,000 tons per day of material at an operating cost of 4-6¢ per ton raised. The equipment includes a single automated dredge and sufficient pipeline with pumping capacity to move the material to the processing facility located onshore. The use of 2 or 3 small dredges was considered and discounted as the reliability of a single large unit is such that down time would be at a very minimum and the use of more than one dredge would add to capital and operating expenses.

Working conditions in Macquarie Harbor can be described as good. Tides average approximately 1 foot with a 2.5 foot maximum in very bad weather. Swell or chop of 4 feet can be considered as maximum and rarely occurs. There are no major currents in the bay. Maximum dredging depth to sediment is 180 feet, well within present dredging capabilities. The sediment is unconsolidated, weighing 1 ton plus per cubic yard and does not contain any components that would interfere with a suction dredging operation.

Construction of the dredge haul would be on site in Macquarie Harbor. The other components would be manufactured in self-contained sections and shipped to the Macquarie Harbor site by sea freight from either Europe or the United States. At this time the Ellicott Machine Corporation appears to have the dredging system best suited to our needs. Ellicott is the leading manufacturer of suction type dredges. Mineral Deposits, a leading Australian user and manufacturer of suction dredges, after analyzing the dredging capabilities necessary to mine the Macquarie Harbor sediments, also recommended cutterhead suction dredging and referred the project to Ellicott. No problems are foreseen.

Also included in Stage I is the de-watering and sizing step. This operation will be accomplished by cyclones either on board the dredge or on the trailing barge. Conventional equipment and technology are utilized and no serious problems are anticipated. An example of a large dredging and concentration operation can be seen in Figure 5.

The de-watered/classified material or the gravity concentrate will be transported to the shore based processing facility by pipeline. Standard hydraulic pumping and slurry pipeline technology is applicable and the only problem that may arise involves the thixotropic properties of the sediment. The clay mineralogy of the sediment indicates that no problems will be encountered, however.

Section II - Preconcentration

The removal by inexpensive means of a significant portion of the non-metal bearing minerals, gangue, is essential to the economic viability of the project. Gravity techniques, standard to the metal mining industry

FIGURE 5

were eliminated during early bench and laboratory investigations. Preliminary investigations by Mineral Development International and by Mineral Deposits, Ltd. that simulated the laminar flow or sluicing principle of mineral separation demonstrated possible applicability of equipment based on this principle.

Commercial pieces of equipment utilizing the laminar flow principle are restricted to two units, the Lamflo Sluice, manufactured by Carpco, Inc., Jacksonville, Florida and the Reichert Cones, manufactured by Mineral Deposits, Ltd., Southport, Queensland, Australia.

An explanation of the laminar flow principle is as follows: Under controlled flow in an acqueous media, minerals will segregate into layers depending on the shape, size and density of the various minerals and on the density, viscosity, velocity and flow pattern of the fluid in which the minerals are suspended. If sufficient differences in the properties of the minerals exist, the characteristics of the suspending media may be adjusted so that a separation can be affected. The heavier (i.e. more dense) minerals will gradually segregate to the bottom of the stream and can be removed in a number of ways. The top of the stream containing the light minerals or gangue goes to tailings and the center of the stream containing a mixture of light and heavy minerals, called middlings, is passed through scavenger units where additional heavy minerals are recovered.

The Lamflo Sluice is a sophisticated version of a pinched sluice. The heavy fraction is cut out with adjustable edges along the bottom of the sluice rather than just at the end of the sluice. These edges also help to control the flow and the turbulence of the suspending media. The unit is designed and manufactured of materials that maximize the laminar flow principle.

The Reichert Cone is simply a circular sluice in the shape of a cone. Material is evenly fed onto the apex of the cone and flows down the cone surface, the slope of which was predetermined by extensive testing. Mineral layers are separated by edges at the bottom of the cone. Until recently the Reichert Cone had the disadvantage of a fixed edge cutting system. This has been rectified with an adjustable system making the cones a more versatile piece of equipment.

Discussions of the sediment characteristics with Mineral Deposits, the Australian Mineral Development Laboratories, Robertson Research, and Mr. R. Porter, a metallurgical consultant, all arrived at the same conclusion that either the Reichert Cone or the Lamflo Sluice could possibly do the necessary preconcentration. This supported by the Mineral Development work noted above indicated that full scale tests using ton quantities of the sediment had to be run. These tests were run at Mineral Deposits and the Australian Mineral Development Laboratories. A Metallurgist from Robertson Research was retained to supervise these tests.

The tests, although successful in concentrating the sulfides, were not successful in concentrating the metals into the sulfide fraction. A three stage cone plant was able to recover 86.5% of the sulfides but only 22% of the copper at a grade of 0.26% copper. Preliminary pro forma economics indicated that a copper recovery of 65% was necessary in a final concentrate representing 10% of the weight. In an effort to upgrade the material, the cone product was further concentrated on spirals. Additional copper losses were incurred and the tests were abandoned.

Lamflo Sluice tests using spirals as cleaners were also unsuccessful in concentrating the metal values into a sulfide concentrate. Maximum recovery of the copper was only 23.7% in 19.1% of the weight. For details of the cone and sluice tests see the Appendices.

It can be concluded from the results of these investigations that the metals present in the sediment are not amenable to concentration by inexpensive gravity techniques. A high grade pyrite concentrate can be produced at minimal cost, however.

Section III - Flotation

In order to achieve the necessary grade of 1.5% copper in the final concentrate, flotation of the gravity concentrate was necessary. Bench scale tests had shown that 55%+ of the total metal in the sample could be recovered in a dominantly sulfide concentrate accounting for 5-10% of the total weight. These were rough tests made on the whole sample without any attempt at conditioning of the surfaces. Standard flotation cell techniques were used.

Capital required for a standard flotation facility to process 25,000 tons per day would be too large for the project to carry. However, new bulk flotation methods have been developed by a number of companies in recent years. Due to their lower capital and operating cost requirements, these systems are particularly adaptable to flotation of low grade materials with a fine particle size distribution. It was estimated that capital costs and operating costs could be reduced by as much as 50% and 33% respectively from those for standard flotation.

Lakefield Research are recognized leaders in this type of flotation. Initial reaction to a preliminary sample was that the system might prove successful. This was substantiated by subsequent bench scale testing which produced a sulfide concentrate containing 1.2% copper and was, after testing, considered suitable for roasting and hydrometallurgical processing. A 65% recovery of copper and 90% recovery of the sulfur was achieved while 92% of the total weight was rejected.

To further substantiate the above, a representative sample of the material utilized for the preconcentration investigations was submitted to Lakefield. An acceptable sulfide concentrate was produced after grinding containing 80.9% of the copper in 10.3% of the original weight at a grade of 1.4% copper and 35.5% sulfur. To obtain the above a fairly complex flowsheet is required and as will be seen in the section on economics yields only a marginally attractive project.

An additional series of flotation tests were also performed by Robertson Research - Australia. Results were very similar to those described above. A concentrate containing approximately 55% of the copper in about 10% of the original weight at a grade of 0.82% copper was obtained without resorting to grinding.

For details on the above process studies, see the Appendices.

Section IV - Hydrometallurgy

Hydrometallurgical treatment of ores with lower than normal grades offers a number of distinct advantages over and above those advantages derived from environmental considerations. A standard hydrometallurgical extraction competes very favorably with a pyrometallurgical extraction both in capital requirements and in operating costs.

Bench scale process investigations and mineral characterization studies have shown that the Macquarie Harbor sediments are such that a concentrate with a copper content suitable for pyrometallurgical treatment cannot be produced without excessive loss of copper and other metals. However, it is possible to produce a sulfide concentrate, consisting principally of pyrite, containing up to 2% copper equivalent without sustaining unreasonable losses. Leaching and extraction technology developed in recent years permits consideration of material with this concentration of metal to be seriously considered as a potential source of copper. Nchanga Copper, Bagdad Copper and Ranchers Exploration at Bluebird are examples of operations that effectively leach and recover copper from tailings and ores with as little as 0.5% copper. Hecla Mining is constructing a major hydrometallurgical installation at present.

The proposed hydrometallurgical scheme will have to include a roasting step. A roast is required because of the necessity of converting the iron to an insoluble oxide and the other metals to soluble oxides. The SO_2 resulting from the roast will be collected for feed to a sulfuric acid plant. A portion of the acid produced will be captively utilized in the leaching process. The proceeds from the sale of the remainder should offset a significant portion of the cost of the roasting and acid manufacturing installation. Recently developed spray or flash roasting technology will be investigated as possible alternatives to a standard roasting facility.

Lakefield Research successfully completed a series of roasting tests and was able to show that the bulk sulfide concentrate produced by flotation could effectively be roasted and leached with sulfuric acid with more than acceptable recoveries of the contained copper.

The concentrate produced by the roast will be suitable for leaching with one of several leaching agents. The most common agents are sulfuric acid (H_2SO_4), certain cyanide salts, such as NaCN, and certain ammonia salts such as NH_4SO_4. The leaching agent used will depend on balancing cost with efficiency and availability. A vat leaching system is proposed.

Dissolved metals will be recovered from the pregnant leach solution by solvent extraction techniques known as liquid ion exchange. The LIX series of chemicals produced by General Mills and the Kelex series of chemicals produced by Ashland Chemical Company are successfully used at a number of operations. Contracting companies offering turnkey hydrometallurgical packages are Parsons Jurden, Davy Powergas, and Bechtel. Discussions have been held with Davy Powergas, Lakeland, Florida, the outcome of which was very encouraging. A schematic of a typical hydrometallurgical system is presented as Figure 6.

Solvent extraction involves the selective extraction of metal ions dissolved in an acqueous solution by ion exchange into a liquid organic compound. An active hydroxyl group acts as the exchange site. The metal ions are then stripped from the loaded organic, again by an ion exchange reaction, into an acidified acqueous solution, usually spent leach liquor (H_2SO_4). During the two exchanges, the metal in solution is concentrated to the level required for electrowinning or chemical manufacture.

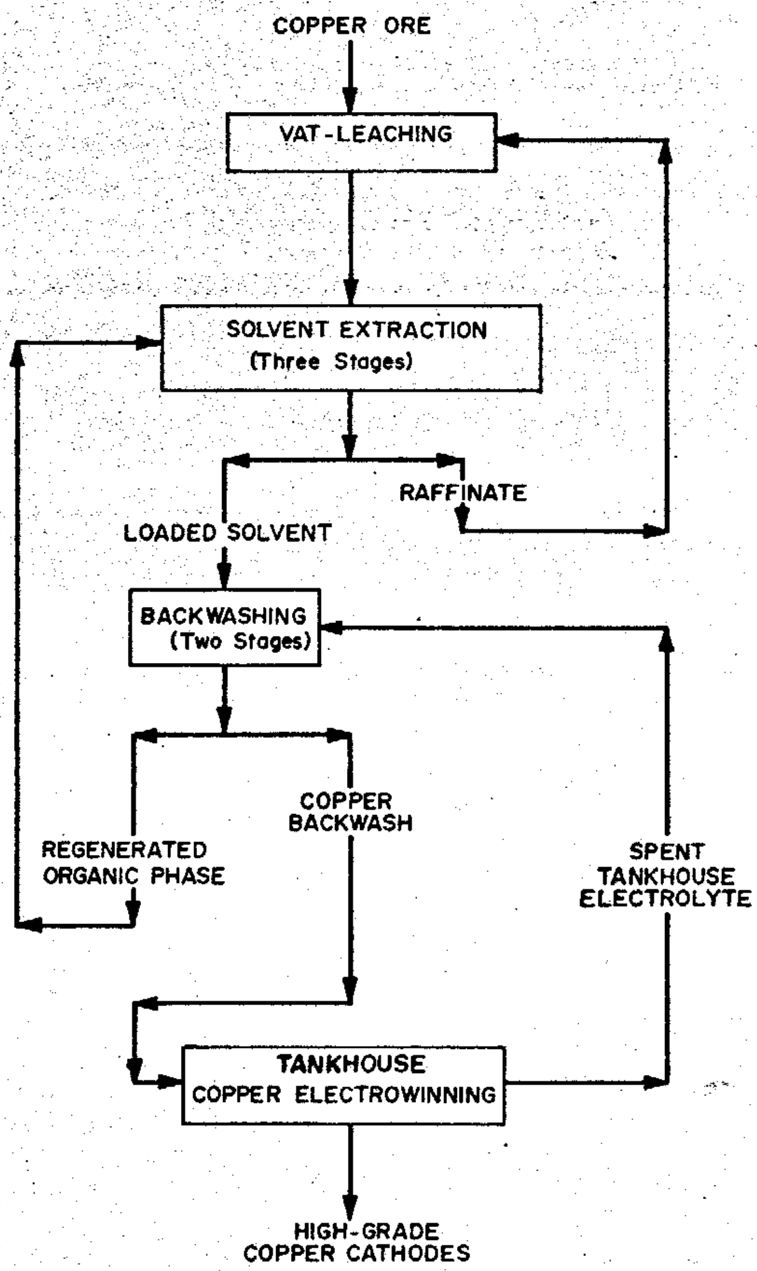

Figure 6 Copper production by leaching, solvent extraction and electrowinning.

The concentrated metal solution will go to an electrowinning unit and metal (i.e. cathode copper) will be produced directly. A hydrometallurgical system offers an alternative, which is not available from a pyrometallurgical system. As the metals are in solution and concentrated, the option is open to go vertical to various chemicals without first making the metal and then re-dissolving it; basically two steps in the chemical manufacturing process are eliminated. Therefore, if a market exists for a specific chemical, the stream can be tapped and the chemical manufactured. Unit value for metal contained in a chemical is always greater than for that of the primary metal. A brief discussion on small volume chemical manufacture was held with the chemical group at Copperhill and tentative plans made to continue these discussions if the project proved viable and additional data became available.

An alternative to leaching that should be considered is chloride volatilization of the metals. Several commercial processes, including the Lurgi and Kowa, might have applicability especially in light of the possibility of interest in the pyrite concentrate. These processes also produce a pelletized iron cinder suitable for blast furnace feed.

PRODUCTS AND MARKETING

A multi-product operation as can be seen in Figure 7, was proposed at Macquarie Harbor. Copper will be the principle metal produced. Significant co-products will be cobalt and zinc. The metals silver and molybdenum will also be produced in small quantities. Options are open to produce either primary metals or to establish a small manufacturing facility to produce chemicals. It has been determined that Australia imports essentially all of its requirements for copper, cobalt, and zinc chemicals. A detailed marketing effort will be necessary to assess the business potential for local chemical production.

Sulfuric acid is an equally important product. At this juncture it appears that unless the acid can be disposed of for cost, the project is not economically viable. A brief study was made of the sulfuric acid market in Australia. It was concluded that sufficient manufacturing capacity was available to satisfy domestic requirements until 1980[1]. This being the case, a brief look was taken at the fertilizer industry, the single largest user of bulk sulfuric acid. It was discovered that a major Australian mining company might possibly be in the market by 1980 for 1500 tons per day of sulfuric acid to be used with phosphate rock from their northern Queensland properties for the production of superphosphates. Contact was made through a third party and a definite interest in holding discussions was confirmed. It is anticipated that beneficiated rock would be shipped to Tasmania for processing at Macquarie Harbor. It should also be noted that this company is totally Australian owned and would make a very acceptable partner.

[1] See section on Economics for information and explanation that may make the sale of large quantities of sulfuric acid viable.

035

261036

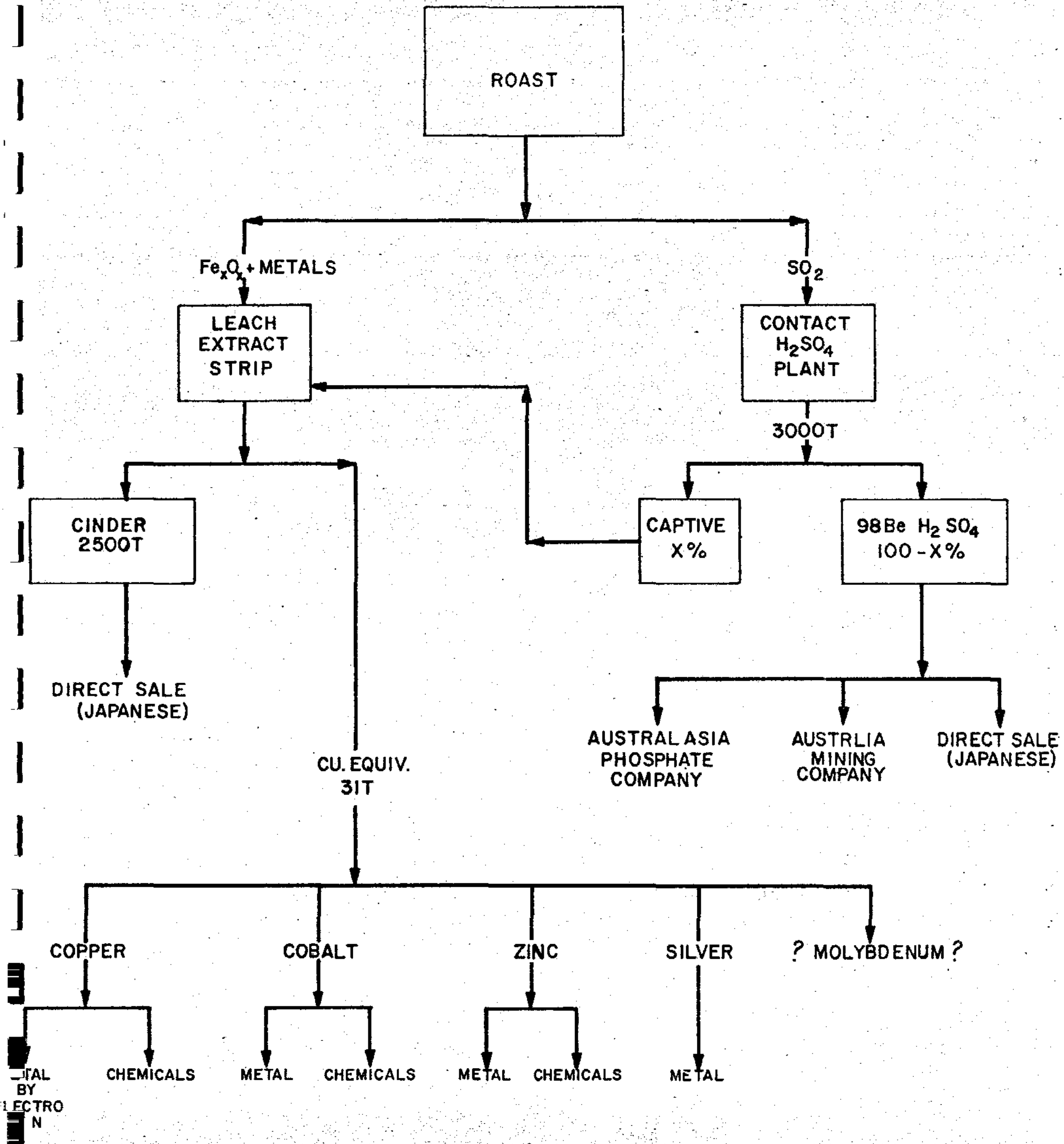

FIGURE 7 Products derived from the Tasmania Copper Operation and possible markets.

A third party contact was also made with another major Australasia phosphate producer. This company is planning to install a phosphoric acid manufacturing facility if a supply of sulfuric acid can be obtained at competitive prices. Their needs would be in the neighborhood of 2000 tons per day. In addition, it was determined that the Japanese, due to severe air pollution problems, are interested in purchasing sulfuric acid to ship to Japan.

It is recognized that there are problems in ocean transport of bulk sulfuric acid. However, it is presently being done successfully from Northwest Acid located at Burnie, Tasmania, Australia by Imperial Chemical Industries. A detailed marketing study will be required to determine the market aspects of the sulfuric acid problem.

ECONOMIC EVALUATION

The project considered herein differs significantly in grade, capital and operating parameters from a conventional mining operation of equivalent size. The following comparison, Table 2, was made initially to demonstrate that it was possible to consider a very low grade deposit if mining and processing could be effected at very low operating costs and with minimum capital expenditures.

Table 2 - Comparison of a Conventional Porphyry Mining/Milling/Smelting Operation with the Proposed Tasmania Copper Operation

	Conventional Porphyry	Tasmania Copper
Orebody	Hard Rock	Unconsolidated Sediment
Grade	0.5% Cu equivalent	0.25 Cu equivalent
Daily ore production	40,000 tons	40,000 tons
Type mining	Conventional open pit	Dredge
Processing		
a. Crushing & grinding	Yes	No
b. Preconcentration	No	Yes
c. Flotation	Yes - standard	Yes - tank
d. Metallurgy	Pyro	Hydro
Capital required	$200 million (not including smelting & refining)	Estimated $40-60 million (including refining)
Operating costs/ton ore mined	$ 3.31	$ 1.04
Mining	$ .63	$ .04 - .06
Crushing and grinding	.80	.00
Preconcentration	.00	.10
Flotation	.60	.40
Smelting, refining & transportation	1.28	.48
Cost/pound copper produced	.37	.34

It can be seen from Table 2 that the potential for developing an economically viable operation from ore with a grade as low as 0.25 copper equivalent exists due to significant reductions in operating and capital costs. The low cost of dredge mining and the elimination of the single most costly step in the process operation, crushing and grinding, greatly reduce per ton operating costs. A further reduction is accomplished by the use of bulk flotation techniques utilizing large volume cells that reduce flotation operating costs by as much as 33%.

Capital requirements are also greatly reduced (Table 2). This is accomplished by utilization of dredging, by elimination of grinding and crushing, by utilization of tank type flotation which reduces the capital for a flotation circuit by as much as 50%, and by preconcentrating the raw ore prior to flotation thereby reducing the amount of flotation capacity necessary. It should be noted that the direct cost of producing a pound of copper is estimated to be slightly less for the Tasmania copper project than for a conventional low grade mining project. Both figures are well within the latest published range for free world per pound operating costs.

Pro forma economic analyses based on the above assumptions demonstrated that an operation of 25,000 tons per day might be economic under the following conditions: 1) operating costs - $1.40 per ton, 2) capital - $40 million, 3) copper price - $0.85, 4) recovery of contained metals - 70%, 5) operating life - 16 years, and 6) the sulfuric acid sold at cost ($10/ton). Under these conditions the project level profitability index was 18.5%. The profitability index at the corporate level was approximately 27%. Cities Service would own 49% and retain management. A 70:30 debt:equity ratio was assumed. Yearly net income to Cities Service was estimated to be in the neighborhood of $2-3 million.

Final process studies utilizing representative material have conclusively demonstrated that the initial assumptions based on bench scale testing were not valid and as a consequence the economics of the project are not as attractive as originally thought. Pro forma economic analyses now demonstrate that the project might only be marginally profitable. The following conditions are most likely to prevail: 1) Operating costs: $1.55, 2) Capital: $49 MM,

3) Copper Price: $0.80, 4) Recovery of contained metals: 75%, 5) operating life: 16 yrs, and 6) The sulfuric acid sold at cost ($10/ton). Under these conditions the project level profitability index was 13.7%. The profitability index at the corporate level was approximately 16.2%. Cities Service would own 49% and retain management. A 70:30 debt:equity ratio was assumed. Sensitivity analyses are presented in Table 3.

The change in economic attractiveness is due primarily to the failure of the preconcentration step to significantly concentrate the contained metals into the sulfide fraction. This leads to the need to concentrate the entire volume of material mined by flotation which must be proceeded by a grinding step. In essence when a sample representative of the deposit was processed it was found that significant parts of the economic advantage anticipated from the initial bench work were eliminated, leaving only the cost of dredging as a significant economic advantage. This was not sufficient to balance the low grade of the ore.

When it became apparent that inexpensive preconcentration of the metals into a sulfide concentrate was not technically possible by gravity means, it was considered essential to see if an operation could be developed that might be attractive to an organization with a specific need for either pyrite or sulfuric acid.

Pro forma economics of producing a sulfur concentrate by gravity means were run using the following parameters: 1) operating costs: $0.25/ton, 2) capital: $11 MM, 3) sulfur recovery: 80%, 4) operating life: 16 years. Under these conditions the project level profitability index was 26.0%. On this basis, the project could be economically viable for an organization with a specific requirement for the pyrite concentrate. As it was assumed that Cities Service would not retain a significant interest, corporate level analyses were not performed.

TABLE 3

SENSITIVITY ANALYSES

TASMANIAN COPPER PROJECT

Recovery at 75%

Price		Project		Corporate	
Copper ¢/lb	H_2SO_4 $/ST	P.I.	Payout Years*	P.I.	Payout Years*
70	10	11.3	5.7	12.6	6.0
80**	10**	13.7**	4.9**	16.2**	6.0**
90	10	15.9	4.3	19.6	5.3
70	20	16.7	4.2	20.9	5.2
80	20	18.6	3.8	24.0	4.2
90	20				
70	30	21.0	3.4	27.8	6.2
80	30	22.6	3.1	30.5	2.7
90	30	24.1	2.9	33.0	2.4

Recovery at 65%

Price		Project		Corporate	
Copper ¢/lb	H_2SO_4 $/ST	P.I.	Payout Years*	P.I.	Payout Years*
70	10	8.8	6.7	9.0	8.9
80	10	11.2	5.7	12.3	7.1*
90	10	13.3	5.0	15.5	6.2
70	20	14.8	4.7	17.9	5.8
80	20	16.6	4.3	20.7	5.2
90	20				
70	30				
80	30	20.9	3.4	27.6	3.3
90	30	22.3	3.2	30.0	2.8

Recovery at 55%

Price		Project		Corporate	
Copper ¢/lb	H_2SO_4 $/ST	P.I.	Payout years*	P.I.	Payout Years*
70	10	6.0	8.5	4.9	15.5
80	10	8.3	6.9	8.2	9.8
90	10				
70	20	12.8	5.4	14.7	6.6
80	20	14.4	4.8	17.2	5.9
90	20				
70	30				
80	30	19.1	3.8	24.7	4.1
90	30				

* From Startup

**Base Case

APPENDIX III

Agreement with HOLB

APPENDIX V

Exploration Licenses

No. 10/ 12 (Regulation 6A) 043

261044

THE MINING ACT 1929

EXPLORATION LICENCE UNDER SECTION 15B

ISSUED to S.R.M. HARVEY of Box 2211, Princeton, New Jersey, 08540 U.S.A.

in respect of One hundred and forty eight (148) square miles of land in the Land District of MONTAGU

vicinity of MACQUARIE HARBOUR as described in the schedule hereto.

This licence shall remain in force until the Twenty eighth day of December, 1973.

This licence is subject to the following conditions:—

That the licensee shall immediately on the issue of this licence take steps to commence preliminary works necessary for the investigation of the area.

That the licensee shall carry out investigations as may be necessary to determine the potential of the area and in particular will commence a reconnaissance sampling of the area for analysis combined with a survey of the underwater topography. If results warrant this will be followed by sampling at much higher density and measurement of shallow sediments.

That the licensee shall employ such technical and other staff and equipment as may be necessary effectively to carry out such investigations.

That the licensee shall furnish the Director of Mines, Hobart, with complete records including plans of drilling and other work within the compass of the programme of exploration. Such records and plans shall be held for official purposes during such time as the areas involved are lawfully held by the licensee or as otherwise agreed to.

That the licensee shall observe the provisions of Section 35 of the Mines Inspection Act, 1968, with regard to notification of boreholes preservation of cores and disposal thereof.

If required by the Director of Mines, the licence holder will forward duplicate samples of rock and mineral samples obtained in the licence area to a place approved by the Director of Mines.

That a Statement of Expenditure verified by statutory declaration shall be lodged with the Director of Mines, Hobart, at the end of each calendar month from the date of this licence.

That such Statement shall be accompanied by a progress report of operations.

This licence shall apply to all minerals.

When no longer required, all large or deep excavations, particularly those made by bulldozer or other earth moving equipment, shall be filled in, or otherwise made safe, in accordance with the Mines Inspection Act, 1968, and reasonable rehabilitation measures taken to the satisfaction of the Director of Mines.

The licensee shall conduct operations so as not to disturb the environment except in so far as this may be necessary to undertake the programme of exploration required by this licence: and in particular shall -

(a) not clear any natural vegetation or make excavations which may be visible from populated areas;

(b) discuss with the local Municipal Council any proposal to clear areas of natural vegetation and shall comply with the reasonable requirements of such Council.

12. The licensee shall observe any instructions which may be given by the Director of Mines with a view of minimising or preventing damage to public or private property.

13. If it is found, that the operations hereby authorised, are causing any undue damage to, or erosion of, the subject land or other land in the vicinity thereof or are unnecessarily disturbing the environment, the Minister may cancel the licence without compensation to the licensee by giving seven days' notice in writing of his intention so to do.

14. The licensee shall not light any fires without the approval of the Rural Fires Board or other relevant authority.

15. Where any aboriginal relic or objects of historic interest are discovered, operations shall be conducted so as not to damage or interfere with such site or object and details of such discovery must be reported to the Director of Mines.

16. The licensee shall notify the local representative of the Forestry Commission before entering on a State Forest and shall comply with the reasonable requirements of such officer in operations on any such State Forest.

17. The licensee shall not interfere in any way with native fauna or bird life.

18. The licensee shall notify the owner and occupier of private land before entering such land.

19. That the security provided by Section 15B (5) of the Mining Act, 1929, (see below) shall be lodged with the Director of Mines before entering private land.

20. Where investigations are undertaken near the coastline or in sand dune areas or where erosion is likely the following conditions shall apply in addition to the foregoing conditions.

(a) Earth moving equipment comprising bulldozer, back hoes, or similar mechanical equipment is not to be used and operations are to be conducted by means of drilling or by the use of hand tools or equipment.

(b) The licensee shall conduct operations as not to cause any damage to shrubs, trees, or other native flora growing on the area demised.

(c) Where it is necessary to remove native or other grasses to enable prospecting operations by boring or otherwise the licensee shall keep to a minimum the area of surface to be disturbed.

(d) All surface soil and grasses shall be stacked separately for replacement.

(e) All excavations shall be filled in as soon as practicable and surface soils and grasses replaced.

21. The licensee shall deposit an amount of $500 as security that the conditions contained herein shall be observed. Upon expiry or sooner determination of the licence, if the licensee satisfies the Director of Mines that such conditions have been complied with, the Director shall refund such deposit, or such portion thereof, as he may determine.

SCHEDULE

Commencing at the posted notice situate at a north east corner of the area whose grid co-ordinates are 820,000 yards N. 334,000 yards E. thence grid east to 340,000 yards E. grid south to 800,000 yards N. again grid east to low water mark on the shoreline of Macquarie Harbour thence by that shoreline of Macquarie Harbour aforesaid and Birch Inlet in a general south easterly, south westerly and north westerly direction to 803,000 yards N. again grid east to low water mark on Ocean Beach thence by that low water mark in a general northerly direction to 830,000 yards N. again grid east to 334,000 yards E. aforesaid thence again grid south to the point of commencement.

EXCLUSIONS

The area embraced by this licence shall be exclusive of:-

(a) All municipal and public reserves and roadways.

(b) All forms of mining tenements and water licences including leases, water licences, easement licences, special and exploration licences, prospectors licences, miners rights, permits to enter, owners consents and owners rights which were in lawful possession or marked out prior to the date of marking out of this licence.

(c) Land exempt from the provisions of the Mining Act, 1929.

(d) Land under the National Parks and Wildlife Act, 1970, not subject to the Mining Act, 1929.

(e) Any Crown reservations or other land set apart or dedicated for any public purposes.

MINISTER FOR MINES.

3 July, 1973.

EXTENSION

This licence is extended under the provisions of Section 15C (6) of the Mining Act, 1929, until the 28th June, 1974.

MINISTER FOR MINES.

17 January, 1974.

EXTENSION

This licence is extended under the provisions of Section 15C (6) of the Mining Act, 1929, until the 28th December, 1974.

MINISTER FOR MINES

1st July, 1974.

TASMANIA

No. 2/74 (Regulation 66)

THE MINING ACT 1929

EXPLORATION LICENCE UNDER SECTION 15B

ISSUED to L.R. WASTELL of 27 Paterson Street, LAUNCESTON, Tas.

in respect of Sixteen (16) square kilometres of land in the Land District of MONTAGU

vicinity of MACQUARIE HARBOUR as described in the schedule hereto.

This licence commences on [illegible] Twenty Eight day of [illegible] 1974.

This licence is subject to the following conditions —

1. That the licensee shall immediate on the issue of this licence take steps to commence preliminary works necessary for the investigation of the area.
2. That the licensee shall carry out investigations as may be necessary to determine the potential of the area and in particular will commence a reconnaissance sampling of the area for analysis combined with a survey of the underwater topography. If results warrant this will be followed by sampling at much higher density and measurement of shallow sediments.
3. That the licensee shall employ such technical and other staff and equipment as may be necessary effectively to carry out such investigations.
4. That the licensee shall furnish the Director of Mines, Hobart, with complete records including plans of drilling and other work within the compass of the programme of exploration. Such records and plans shall be held for official purposes during such time as the areas involved are lawfully held by the licensee or as otherwise agreed to.
5. That the licensee shall observe the provisions of Section 35 of the Mines Inspection Act, 1968, with regard to notification of boreholes preservation of cores and disposal thereof.
6. If required by the Director of Mines, the licence holder will forward duplicate samples of rock and mineral samples obtained in the licence area to a place approved by the Director of Mines.
7. That a Statement of Expenditure verified by statutory declaration shall be lodged with the Director of Mines, Hobart, at the end of each calendar month from the date of this licence.
8. That such Statement shall be accompanied by a progress report of operations.
9. This licence shall apply to all minerals.
10. When no longer required, all large or deep excavations, particularly those made by bulldozer or other earth moving equipment, shall be filled in, or otherwise made safe, in accordance with the Mines Inspection Act, 1968, and reasonable rehabilitation measures taken to the satisfaction of the Director of Mines.
11. The licensee shall conduct operations so as not to disturb the environment except in so far as this may be necessary to undertake the programme of exploration required by this licence: and in particular shall -

    (a) not clear any natural vegetation or make excavations which may be visible from populated areas;

(b) discuss with the local Municipal Council any proposal to clear areas of natural vegetation and shall comply with the reasonable requirements of such Council.

12. The licensee shall observe any instructions which may be given by the Director of Mines with a view of minimising or preventing damage to public or private property.

13. If it is found, that the operations hereby authorised, are causing any undue damage to, or erosion of, the subject land or other land in the vicinity thereof or are unnecessarily disturbing the environment, the Minister may cancel the licence without compensation to the licensee by giving seven days' notice in writing of his intention so to do.

14. The licensee shall not light any fires without the approval of the Rural Fires Board or other relevant authority.

15. Where any aboriginal relic or objects of historic interest are discovered operations shall be conducted so as not to damage or interfere with such site or object and details of such discovery must be reported to the Director of Mines.

16. The licensee shall notify the local representative of the Forestry Commission before entering on a State Forest and shall comply with the reasonable requirements of such officer in operations on any such State Forest.

17. ~~The licensee shall not interfere in any way with ... [illegible]~~

18. The licensee shall notify the owner and occupier of private land before entering such land.

19. That the security provided by Section 153 (5) of the Mining Act, 1929 (see below) shall be lodged with the Director of Mines before entering private land.

20. Where investigations are undertaken near the coastline or in sand dune areas or where erosion is likely the following conditions shall apply in addition to the foregoing conditions.

(a) Earth moving equipment comprising bulldozer, back hoes, or similar mechanical equipment is not to be used and operations are to be conducted by means of drilling or by the use of hand tools or equipment.

(b) The licensee shall conduct operations as not to cause any damage to shrubs, trees, or other native flora growing on the area demised.

(c) Where it is necessary to remove native or other grasses to enable prospecting operations by boring or otherwise the licensee shall keep to a minimum the area of surface to be disturbed.

(d) All surface soil and grasses shall be stacked separately for replacement.

(e) All excavations shall be filled in as soon as practicable and surface soils and grasses replaced.

21. That the amount of $[illegible] held as security on Exploration Licence 16/73 be applied to this licence also that the conditions contained herein shall be observed. Upon expiry or sooner determination of the licence, if the licensee satisfies the Director of Mines that such conditions have been complied with, the Director shall refund such deposit, or such portion thereof, as he may determine.

SCHEDULE

Commencing at the posted notice situate at the north-west corner of the area at a point whose grid coordinates are 362 493 metres E. 5320 [illegible] metres N. and being a point at low water mark on Macquarie Harbour thence southerly by part of an eastern boundary of E.L. 16/73 to grid coordinate 362 [illegible] metres E. 5322 702 metres N. thence easterly by a northern boundary of E.L. 16/73 aforesaid to grid coordinates 364 623 metres E. 5 322 726 metres N. and being at low water mark on Macquarie Harbour and thence by that low water mark in a general north westerly direction to the point of commencement.

DEPARTMENT OF MINES - TASMANIA

SCHEDULE 'A'

Conditions of Special Prospectors' Licences and Exploration Licences under the Mining Act, 1929.

Operational:

1. That the licensee shall observe the provisions of Section 35 of the Mines Inspection Act 1968 with regard to notification of bore holes preservation of cores and disposal thereof.

2. At the termination of the licence or at any time at the option of the licensee all drill core and samples required by the Director of Mines shall be delivered in approved containers into the Department of Mines Drill Store at Hobart at the cost of the licensee unless the Director of Mines notifies the licensee in writing that such core or samples are not required.

3. When no longer required all large or deep excavations, particularly [illegible] shall be [illegible] with the Mines Inspection Act, 1968 and reasonable rehabilitation measures taken to the satisfaction of the Director of Mines.

4. The licensee shall conduct operations so as not to disturb the environment except in so far as this may be necessary to undertake the programme of exploration required by this licence; and in particular shall -

   (a) not clear any natural vegetation or make excavations beyond that necessary for actual sampling;

   (b) discuss with the local Municipal Council any proposal to clear areas of natural vegetation and shall comply with the reasonable requirements of such Council.

5. The licensee shall observe any instructions which may be given by the Director of Mines with a view of minimising or preventing damage to public or private property.

6. The licensee shall not light any fires without the approval of the Rural Fires Board or other relevant authority. In the case of Crown land the approval of the Director of Lands is required.

7. The licensee shall notify the local representative of the Forestry Commission before entering on a State Forest and shall comply with the reasonable requirements of such officer in operations on any such State Forest.

8. Where any aboriginal relic or objects of historic interest are discovered, operations shall be conducted so as not to damage or interfere with such site or object and details of such discovery must be reported to the Director of Mines.

9. The licensee shall not interfere in any way with native fauna or bird life.

10. Where investigations are undertaken near the coastline or in sand dune areas or where erosion is likely the following conditions shall apply in addition to the foregoing conditions.

    (a) Earth moving equipment comprising bulldozer, back-hoes, or similar mechanical equipment is not to be used and operations are to be conducted by means of drilling or by the use of hand tools or equipment.

    (b) The licensee shall conduct operations as not to cause any damage to shrubs, trees, or other native flora growing on the area demised.

DEPARTMENT OF MINES - TASMANIA

SCHEDULE 'A'

10. (c) Where it is necessary to remove native or other grasses to enable prospecting operations by boring or otherwise the licensee shall keep to a minimum the area of surface to be disturbed.
    (d) All surface soil and grasses shall be stacked separately for replacement.
    (e) All excavations shall be filled in as soon as practicable and surface soils and grasses replaced.

11. Before commencing the construction of access routes or other works [illegible] on the environment the [illegible] the Director of Environmental Control at Hobart and shall forward a copy to the Director of Mines.

Reporting:

1. A Statement of Expenditure verified by statutory declaration shall be lodged with the Director of Mines, Hobart at the end of each calendar month from the date of this licence.

2. This Statement shall be accompanied by a progress report of operations.

3. The licensee shall furnish the Director of Mines, Hobart, with complete records of all investigations undertaken during the term of the licence. These shall include detailed reports, plans, sections, analyses, metallurgical investigations and feasibility and other studies. All plans must include transparencies unless the Director of Mines advises in writing that such are not required.

4. All information furnished to the Director of Mines under this licence will be held for official purposes:
   (a) during a period of five years from the date on which such information was furnished to the Director of Mines; or
   (b) until the areas to which the reports relate are no longer lawfully held under the Mining Act, 1929

   whichever shall occur first.

5. Upon relinquishment of any part of the area described in the schedule hereto the licensee shall furnish a report containing all information relating to such area unless the Director of Mines advises in writing that such report is not required.

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SCHEDULE 'A'

DEPARTMENT OF MINES - TASMANIA

Other:

1. That the licensee shall carry out investigations as may be necessary to determine the mineral potential of the area.

2. That the licensee shall employ such technical and other staff and equipment as may be necessary effectively to carry out such investigations.

3 ~~This licence shall apply to~~ ALL MINERALS EXCLUDING COAL.......... AND OIL

4 The licensee shall notify the owner and occupier of private land before entering such land

5. That the security provided by Section 15B (5) of the Mining Act, 1929 (see below) shall be lodged with the Director of Mines before entering such land.

6. The licensee shall observe, perform and fulfill the conditions as set forth in Schedule 'A' attached hereto.

7. The lessee shall be liable to pay the cost of any work carried out to remedy any damage arising from any breach of the conditions of this licence.

8. The licensee shall deposit an amount of $500 as security that the conditions contained herein shall be observed. Upon expiry or sooner determination of the licence, if the licensee satisfies the Director of Mines that such conditions have been complied with, the Director of Mines shall refund such deposit, or such portion thereof, as he may determine.

9. If it is found that the operations hereby authorised, are causing any undue damage to, or erosion of, the subject land or other land in the vicinity thereof or are unnecessarily disturbing the environment, the Minister may cancel the licence without compensation to the licensee by giving seven days' notice in writing of his intention so to do.

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ENDORSEMENTS

EXCLUSIONS

The area [illegible] by this licence shall be exclusive of:-

(a) All municipal and public reserves and roadways.

(b) A[illegible] for[illegible] of mining tenements and water licences including leases, water licences, easement licences, special and exploration licences, [illegible] spectors licences, miners rights, permits to enter, owners consents and owners rights which were in lawful possession or marked out [illegible] to the date of marking out of this licence.

(c) [illegible] from the provisions of the Mining Act, 1929.

(d) [illegible] the [illegible] Parks and Wildlife Act, 1970, not subject to the Mining Act, 1929.

(e) Any Crown reservations or other land set apart or dedicated for any pu[illegible] purposes.

MINISTER FOR MINES.
4 February, 1974.

EXTENSION

This licence is extended under the provisions of Section 15C (6) of the Mining Act, 1929 until the 28th December 1974.

MINISTER FOR MINES
26 June 1974.

EXTENSION & ALTERATION OF CONDITIONS

This licence is extended under the provisions of Section 15C (6) of the Mining Act, 1929, until the 28th June, 1975. The current conditions of the licence are rescinded and replaced by those on Schedule 'A' attached.

MINISTER FOR MINES
[illegible] December, 1974

ADDITION TO AREA

In accordance with Section 15c (7) of the Mining Act, 1929, this licence shall now apply to a total area of 16 square kilometres (more or less) as described hereunder:

SCHEDULE

Commencing at the north-west corner of the area being the intersection of grid 36[illegible] metres E. and the high water mark on Macquarie Harbour and being a point on a eastern boundary of E.L. [illegible]6/73 thence southerly by that eastern boundary aforesaid to grid co-ordinates 36[illegible] metres E. 5 322 702 metres N. thence easterly by a northern boundary of E.L. [illegible]6/73 aforesaid to the high water mark on Macquarie Harbour aforesaid thence in a general westerly direction by that high water mark to the point of commencement.

EXCLUSIONS

The area embraced by this licence shall be exclusive of:

(a) All municipal and public reserves and roadways.

(b) All forms of mining tenements and water licences including leases, water licences, easement licences, special and exploration licences, prospectors licences, miners rights, permits to enter, owners consents and owners rights which were in lawful possession or marked out prior to the date of marking out of this licence.

(c) Land exempt from the provisions of the Mining Act, 1929.

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EXEMPTIONS (Cont.)

(d) Land under the National Parks and Wildlife Act, 1970, not subject to the Mining Act, 1929.

(e) All Crown reservations or other land set apart or dedicated for any public purposes.

[signature]

MINISTER FOR MINES

18 March, 1975

EXTRACTS FROM THE PROVISIONS OF THE MINING ACT 1929

Exploration Licences

Section 15a (1) An exploration licence—

(i) Has effect in relation to such area, and for such period as the Minister may determine:

(ii) Shall be granted upon and subject to such terms and conditions as are prescribed in this section and such other terms and conditions (including conditions as to the fees and rent to be paid by the holder thereof) as the Minister may determine:

and

(iii) While in force, has effect to authorize the holder thereof, subject to the observance of the terms and conditions thereof and to the provisions of this Act, to enter upon and pass over or across Crown Lands and, subject to subsection (5) of this section, private lands, within the area to which it relates, and to prospect and search therein for such mining products as may be specified in the licence and to do all such other acts and things as may be reasonably necessary to enable the holder thereof to engage in large scale exploration work.

(4) The holder of an exploration licence shall, subject to this Act—

(i) Engage in the [illegible] such aerial geological or geophysical [illegible] exploration as the Minister [illegible]

(ii) Furnish the Director [illegible]

and

(iii) Keep an adequate record of all operations conducted under the authority of the licence, and at all reasonable times permit the Director, or any officer authorized by the Director so to do, to examine those records and inspect any specimens or materials obtained in the course of those operations.

(5) The holder of an exploration licence shall not enter on private land thereunder unless he has given security as provided in subsection (2) of section seventy, and, upon entering on private land, is subject to sections seventy-one and seventy-two as if his exploration licence were a permit under section seventy.

(6) Upon application made in that behalf by the holder of a special prospector's licence or of an exploration [illegible] the expiration of the period for which it was granted to have effect the Minister may [illegible] the licence for such further period or periods and upon such conditions as he thinks fit but so that [illegible] special prospector's licence, the aggregate period for which the licence and all extensions thereof are granted does not exceed twelve months.

(7) Where a licence is extended pursuant to subsection (6) of this section, the Minister may—

(i) On the recommendation of the Director, add to; or

(ii) Reduce the area of land comprised in the licence.

(8) If the holder of a special prospector's licence or of an exploration licence contravenes or fails to comply with any of the provisions of this Act or any of the terms and conditions to which the licence is subject, the Minister may, by notice in writing to the holder, revoke the licence.

(9) With the consent of the holder of a special prospector's licence or an exploration licence—

(i) A prospector's licence, mining lease, water licence, or easement licence may be granted in respect of land comprised therein as if the special prospector's licence or exploration licence did not exist; and

(ii) A Miner's right may be exercised as if the land comprised in the special prospector's licence or exploration licence were unoccupied land.

(10) Special prospector's licences and exploration licences may, with the consent of the Minister, be transferred on prescribed payment of the prescribed fee.

Regulation 4 (5) Where application is made for a special prospector's licence or an exploration licence, and a licence is granted [illegible] in respect of an area less than that comprised in the application, the holder, within seven days after the [illegible] such licence, shall affix to his datum-post an amended notice, showing the area which he is authorized to prospect.

261054 053

TASMANIA

No. E.L. 16/75

(Regulation 6A)

THE MINING ACT 1929

EXPLORATION LICENCE UNDER SECTION 15B

Issued to Australia-Cities Service Inc. of 5th Floor, Metropolitan Freeholds Building, 151 Macquarie Street, SYDNEY, N.S.W. 2000

in respect of Twenty five (25) ~~square miles~~ square kilometres (more or less) of land in the Land District of Montagu & Franklin

vicinity of Macquarie Harbour as described in the schedule hereto.

This licence shall remain in force until the First day of March, 1976

This licence is subject to the following conditions:—

1. That the licensee shall immediately on the issue of this licence take steps to commence preliminary works necessary for the investigation of the area.
2. That the licensee shall carry out investigations as may be necessary to determine the mineral potential of the area, and in particular will commence an extensive drilling programme at one half kilometre centres to test the sediments in the King River channel.
3. That the licensee shall employ such technical and other staff and equipment as may be necessary effectively to carry out such investigations.
4. This licence shall apply to all minerals excluding coal and oil.
5. The licensee shall notify the owner and occupier of private land before entering such land.
6. That the security provided by Section 15B (5) of the Mining Act, 1929 (see below) shall be lodged with the Director of Mines before entering private land.
7. The licensee shall observe, perform and fulfill the conditions as set forth in Schedule 'A' attached hereto.
8. The lessee shall be liable to pay the cost of any work carried out to remedy any damage arising from any breach of the conditions of this licence.
9. The licensee shall deposit an amount of $250.00 as security that the conditions contained herein shall be observed. Upon expiry or sooner determination of the licence, if the licensee satisfies the Director of Mines that such conditions have been complied with, the Director of Mines shall refund such deposit, or such portion thereof, as he may determine.
10. If it is found, that the operations hereby authorised, are causing any undue damage to, or erosion of, the subject land or other land in the vicinity thereof or are unnecessarily disturbing the environment, the Minister may cancel the licence without compensation to the licensee by giving seven days' notice in writing of his intention so to do.

261055

SCHEDULE

Commencing at the north-west angle of the area and being a point on the eastern boundary of E.L. 2/74 at high-water-mark on Macquarie Harbour whose grid co-ordinates are 364400 metres E. 5329005 metres N. thence grid east to 371650 metres E. grid south to 5326005 metres N. grid west to the high-water-mark on Macquarie Harbour aforesaid thence in a general northerly direction by that high-water-mark to the point of commencement.

EXCLUSIONS

The area embraced by this licence shall be exclusive of:-

(a) All municipal and public reserves and roadways.

(b) All forms of mining tenements and water licences including leases, water licences, easement licences, special and exploration licences, prospector's licences, miners rights permits to enter, owners consents and owners rights which were in lawful possession or marked out prior to the date of marking out of this licence.

(c) Land exempt from the provisions of the Mining Act, 1929.

(d) Land under the National Parks and Wildlife Act, 1970, not subject to the Mining Act, 1929.

(e) All Crown reservations or other land set apart or dedicated for any public purposes.

S. Frost

MINISTER FOR MINES

1st September, 1975.

APPENDIX VI

MINERAL DEPOSITS LIMITED

"Cone Concentrator Studies and Circuit Designs

On Alluvial Sulphide Ore"

AUSTRALIA CITIES SERVICE

REPORT 15.213.1/1

CONE CONCENTRATOR STUDIES AND CIRCUIT DESIGNS ON ALLUVIAL SULPHIDE ORE

M.S. CROSS B.Met.E., A.M.A.I.M.M.

29.4.76

MINERAL DEPOSITS LIMITED
RESEARCH AND ENGINEERING DEPARTMENT
SOUTHPORT QUEENSLAND
AUSTRALIA

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1. INTRODUCTION

1.1 MATERIAL

The material supplied to Mineral Deposits Limited was a bulk sample of sedimentary material from Macquarie Harbour in the south west corner of Tasmania. The material is a deposition of tailings from the Mount Lyell Mining Company's copper operation. A complete sizing copper and sulphur analysis is found in Table 1.1. The mineralogy of the material is found in reports of Robertson Research.

The sample as supplied came in 212, 44 gallon drums. The drums were half or three-quarters full, the material packed in plastic bags. The numbering on the drums indicated three distinct samples:-

Group 1 - Drum No. only.
Group 2 - Drum No. plus a Drill Hole No.
Group 3 - Drum No. plus a number between 211 and 221.

The Group 3 series of drums was used as the bulk sample for testwork. The other two groups of drums are held in storage at Southport.

The head analysis in Table 1.1 was carried out on a composite sample from the Group 3 series of drums.

1.2 OBJECTIVES

The object of the testwork was to define a gravity separation flowsheet for the concentration and recovery of the sulphides as a bulk concentrate. The emphasis was to be placed on recovery with final grade of product being of secondary importance.

1.3 TERMS OF REFERENCE

1.3.1 The terms of reference are laid down in correspondence and discussions with Australia Cities Service. Metallurgical testwork was to be sufficient to provide adequate information for a preliminary capital estimate for plant and equipment.

1.3.2 Particular reference to these objectives can be found in the following correspondence:-

(a) Letter to D. H. Buchholz from R.A. Graves dated 18.7.75.

(b) Letter to Mineral Deposits Limited from Dr. D.D. Jinks dated 12.11.75.

Both these letters are found in the appendix.

1.4 SUMMARY

1.4.1 Sulphur Recovery

The test results and circuit studies indicate that a three stage cone concentrator plant would provide a 16% sulphur concentrate at a recovery of 86.5%S with a weight recovery of 16%.

The sulphur losses would be in the very coarse fraction as composite grains and as fine sulphides in the minus 63 micron material.

The fine sulphides could be recovered by cycloning plant tailings followed by a flotation stage.

1.4.2 Copper Recovery

The test results indicate a copper recovery of approximately 22% in the final concentrate. Final grade of copper in the concentrate would be 0.26%.

The copper minerals have been difficult to find and identify. An extensive program is being carried out by Robertson Research to identify the form in which Cu is present.

1.4.3 Process Design

A three stage cone plant has been designed that will provide the recovery of sulphur and copper outlined in the previous sections. This flowsheet has been designed for 1000 tonnes per hour and is drawn in Figure 4.1.

1.4.4 Spiral Testwork

The cleaner cone concentration results indicated a very high recovery of sulphur. The grade of final concentrate could only be upgraded to 24%S by cones. Additional tests were then carried out on cleaner concentrates on Reichert Spiral Concentrators. The results of this testwork are found in a second report No.15.213.1/2.

M. S. Cross

M.S. CROSS B.Met.E., A.M.A.I.M.M.

SENIOR PROJECT METALLURGIST

29th April, 1976.

MSC:MM

2. TEST EQUIPMENT

2.1 MACHINES

2.1.1 Single Cone Concentrator Test Rig

A sketch of the single cone test rig is Fig. 2.1. The sketch is self-explanatory; feed rate and pulp density can be controlled and all products from the test rig can be sampled via a semi automatic sampler.

2.1.2 Multi-Stage Cone Concentrator Test Rig

A sketch of the multi-stage cone test rig is Fig. 2.2. The sketch again is self-explanatory; feed rate, pulp density and wash water additions can be controlled within limits. All final cone products can be sampled.

2.1.3 For the multi-stage testwork the cone configuration used was a DSV.DSV.DSV.DSV. This configuration is shown in Fig. 2.3. The coding is D for double cone, S for single cone, and V for variable insert on the single cone. The variable inserts are used for testwork for simplicity.

2.2 ANALYTICAL SYSTEMS

2.2.1 Sampling

Both test rigs have semi-automatic sampling systems incorporated with the rig for the taking of timed samples.

2.2.2 Cone Concentrator products were examined and analysed as outlined in Fig. 2.4.

2.2.3 Product Sizing

Cone products were dry screened to 63 microns. Procedure is outlined in Fig. 2.5.

2.2.4 Product Sizing including minus 63 micron material was analysed as shown in Fig. 2.6.

2.2.5 (i) Analyses for sulphur and copper were carried out by XRF. For sulphur assays were also carried out by gravimetric methods as a check on the XRF method.

(ii) Heavy mineral separations were carried out using bromoform to produce a heavy mineral sink fraction and a float gangue fraction.

2.2.6 Sizing of minus 63 micron material was carried out by a Warman Cyclosizer.

3. TESTWORK AND PROCEDURES

3.1 INTRODUCTION

The test program was carried out in two major sections:-

Stage 1 - Single Cone Concentrator Testwork.

Stage 2 - Multi-Stage Cone Concentrator Testwork.

Stage 2, being the major part of the program, was then divided into a number of sections:-

A - Rougher Cone Concentrator Tests.

B - Rougher Cone Concentrator production to supply concentrates for Parts C and D.

C - Rougher Retreat Cone Concentrator Testwork.

D - Cleaner Cone Concentrator Testwork.

3.2 STAGE 1 : SINGLE CONE CONCENTRATOR TESTWORK

3.2.1 Feed for this stage was from the Group 3 series of drums; 8 drums from the hole number 216 were used.

The material was washed into the single cone test rig through a 1/8 inch screen to prevent oversize interfering with the cones operation.

3.2.2 Test Program

A series of tests was then run to determine some of the separation characteristics of the sulphides and heavy mineral. The following table outlines the test program that was carried out.

Test No.	Insert Setting	Remarks
1	1	Original head feed from Hole No.216
2	5	
3	9	
4	1	20 gallons of concentrates were bled out.
5	5	
6	9	
7	1	Additional 20 gallons of concentrates were bled out.
8	5	
9	9	
10	1	Original feed was wet tabled to produce concentrates for test 10-12
11	5	
12	9	
13	1	Repeat of tests 1-3 at lower tonnages and higher pulp densities
14	5	
15	9	

Test No.	Insert Setting	Remarks
16	1	)
17	5	)
18	9	) Repeat of tests 4-9
19	1	) at lower tonnages and
20	5	) higher pulp densities
21	9	)

3.2.3 Results

Tests 1 to 12 were analysed for both heavy mineral and sulphur. Tests 13-21 were analysed for heavy mineral only. The results for both groups of tests are shown in Tables 3.1 and 3.2.

3.2.4 Discussion

The main feature of the results is the consistency of performance. Recovery and upgrade ratio are not very high for heavy mineral and are slightly higher for sulphur.

The major reason for the heavy mineral result is that the mineral excluding sulphides would be iron silicates of low S.G. The sulphur results are better due to the higher S.G. of the sulphides. The consistent performance can be seen by examination of the curves drawn in Fig. 3.3.

The top straight lines are a plot of concentrate TPH versus feed TPH for different insert settings. The curves at the bottom are a plot of upgrade ratio versus feed TPH for different insert settings.

3.3 STAGE 2 : MULTI-STAGE CONE CONCENTRATOR TESTWORK

3.3.1 Rougher Cone Concentrator Tests

This section of the testwork was divided into three sections:-

(a) Preliminary Investigation.

(b) Rougher Cone Concentrator testwork for modelling investigations.

(c) Optimising Rougher Stage testwork confirming previous testwork.

(a) Preliminary

Material used for this group of tests was a mixture of the drums from Group 3; 1 drum from the even numbered holes and 2 drums from the odd numbered holes.

An initial series of three tests were run as a preliminary investigation. Insert settings used were based on information gained from the single cone testwork.

<u>Results</u> The insert settings used are recorded on Fig. 3.4. The results for the tests are shown on Table 3.5. Test 3 was assayed for sulphur and these results are shown in Table 3.5.

<u>Discussion</u> The major conclusion is gained from Test 3. Heavy mineral recovery shows 77% of the mineral recovered into 66% of the weight and sulphur recovery is 93% into the same weight. This result was very encouraging for sulphur recovery and weight rejection. The reason for the difference between the two recoveries is the lower S.G. of the heavy mineral fraction which is in the form of iron silicates.

(b) Rougher Cone Concentrator Tests for modelling investigations

Material used for this testwork was the same material as for (a) with additional drums added to maintain pulp density in the test rig.

<u>Results</u> The insert settings used for the two series of tests are shown in Fig. 3.6. The cone coding is also shown on this figure. The results obtained from the tests are shown in Tables 3.7 A, B, C, D.

The results from these tests are used to establish a mathematical model that will describe the ebhaviour of the alluvial sulphide material on the cone concentrator.

(c) Optimising Rougher Tests

Material used for this testwork was a mixture of the remaining group 3 drums. All the remaining group 3 drums were sorted into their hole numbers as shown below with the drums used for the testwork shown.

Hole No.	No. Left	No. Used
211	9	2
212	9	2
213	8	2
214	9	2
215	8	1
216	2	1
217	6	1
218	8	2
219	1	1
220	5	1
221	-	-

<u>Results</u> Insert settings used for this testwork are shown in Fig. 3.8. The cone products were assayed for heavy mineral, sulphur and copper. The cone results are shown in Table 3.9A.

The cone products for Test 13 were sized and analysed for sulphur and copper. Results are shown in Table 3.9B and 3.9C.

Discussion : Sulphur Recovery Sulphur recovery is maintained at 80% to the concentrates. The basic reason is that the sulphur is present in the size ranges where the cone concentrator is most effective. This can be seen from the sulphur sections of Table 3.9C. Weight rejection to tails is also high due to the sizing of feed being coarse in relation to the sulphur distribution.

Discussion : Copper Recovery The copper recovery and distribution is the reverse to the sulphur. The recovery of copper is similar to the sulphur when comparing size distribution, but the copper distribution in the heads follows the weight distribution, so the overall recovery of copper is poor. Reports by Robertson-Research indicate that the copper is present in a number of different forms that are difficult to identify.

Flowsheet Design On completion of the evaluation of the rougher results, an initial flowsheet was drawn, Fig. 3.10. In order to test the various stages of this flowsheet, Rougher concentrates 1 and 2, plus Rougher Concentrates 3 and 4 would have to be produced. As mentioned in the initial pages of the report, recovery of sulphur and gangue rejections was more important than grade of final product and recovery. The flowsheet Fig. 3.10 is a basic circuit designed for recovery and not optimum product grade.

3.3.2 Production of Rougher Concentrates

The purpose of this section of the testwork was the production of Rougher Concentrates to provide feed material to a cleaner stage and a rougher retreat stage of cone concentration.

The multi-stage cone test rig as previously described in Fig. 2.2 was modified so that final products would be bled out of the circuit and new feed added to circuit at a controlled rate.

The remaining drums of bulk sample, excluding drums with numbers between 197 and 210 were then processed to produce two concentrate products plus a final tailing.

The concentrate products were:-

Rougher Concentrates 1 and 2 - feed to Cleaner Stage.
Rougher Concentrates 3 and 4 - feed to R.Retreat Stage.
Rougher Tailings - Drummed and discarded.

3.3.3 Rougher Retreat Cone Concentrator Tests

The feed materials to this stage were Rougher concentrates 3 and 4 produced earlier in the test program (3.3.2). There was only enough feed to run the test rig and insufficient feed to produce more concentrates for a cleaner stage.

Results for the Rougher retreat stages are found in Table 3.11.

The result for both tests are similar to those found for the rougher cone tests.

3.3.4 Cleaner Cone Concentrator Testwork

The materials used for this section of the work were the Rougher Concentrates Nos. 1 and 2 produced in the Rougher cone testwork (3.3.2).

The cleaner cone testwork was divided into a number of sections:-

- Section (a) - Preliminary investigation of cleaner cone to see what actually happens.
- Section (b) - Full cone testing to produce information for modelling procedures.
- Section (c) - Optimising of cleaner cone to investigate maximum sulphur grades obtainable.
- Section (d) - Production of Cleaner Concentrate for further upgrading tests on spiral concentrators.

(a) Preliminary Testwork

A series of two tests was run with different insert settings to evaluate the performance of the sulphides in a cleaner operation.

The insert settings are found on Fig. 3.12 and the results for the two tests are in Table 3.13.

The results from the preliminary tests were very encouraging, particularly with regard to sulphur recovery. Copper recovery is only slightly better than rougher tests.

(b) Cleaner Cone Test for Modelling

The results for this series of tests are shown in Table 3.14 A & B. Insert settings were the same as for Test 19; these can be found in Fig. 3.12. Cone coding is found in Fig. 3.6.

A check test on the overall result was also run to verify results obtained in tests 20-23. This is found at the bottom of tables 3.14 A & B and was test 24.

(c) Optimising of Cleaner Stage

The initial cleaner tests 18-24 indicate excellent recoveries of sulphur into concentrates. A series of tests were then run to maximise sulphur grade of the final product. The results of these tests Nos.

25, 26 & 27 are shown in Table 3.16. The insert settings were the major alteration. The gap of the insert on both the double cones and single cones were made progressively smaller. A list of insert settings is shown on Fig. 3.15.

The results from tests 25-27 indicate that to achieve any significant change in sulphur grades above 20%S would mean a significant change in the recovery with an increase in middling load: running with a large middling load would increase losses of sulphur.

(d) Production of Cleaner Concentrates

The previous section marked the end of cone concentrator tests. Cleaner concentrates were then produced by a similar method to that described for the production of rougher concentrates. These concentrates were then used for spiral testwork to evaluate further upgrading of sulphur content of final product.

4. PROCESS DESIGN

4.1 MATHEMATICAL MODELLING

A mathematical model for the Reichert Cone Concentrator has been designed by Mineral Deposits Limited. The model utilises the variables, solids, water and mineral tonnes per hour and slot or insert setting. The model can then be used to predict performance of the material on cone concentrators. A second function is to develop circuits and evaluate the performance of the designed circuit.

4.2 TYPE OF MODEL

In the Australia Cities Service testwork a model for the sulphide ore was developed with heavy mineral being used. Additional models were attempted for pyrite and sulphur but were not successful.

4.3 FLOWSHEET

The flowsheet for a three stage cone plant is shown in Fig. 4.1. Table 4.2 lists cone settings used for each stage. The stage results obtained from these figures are shown in Table 4.3 and a comparison with the actual testwork is shown in Table 4.4. These comparisons are for both heavy mineral and sulphur recovery.

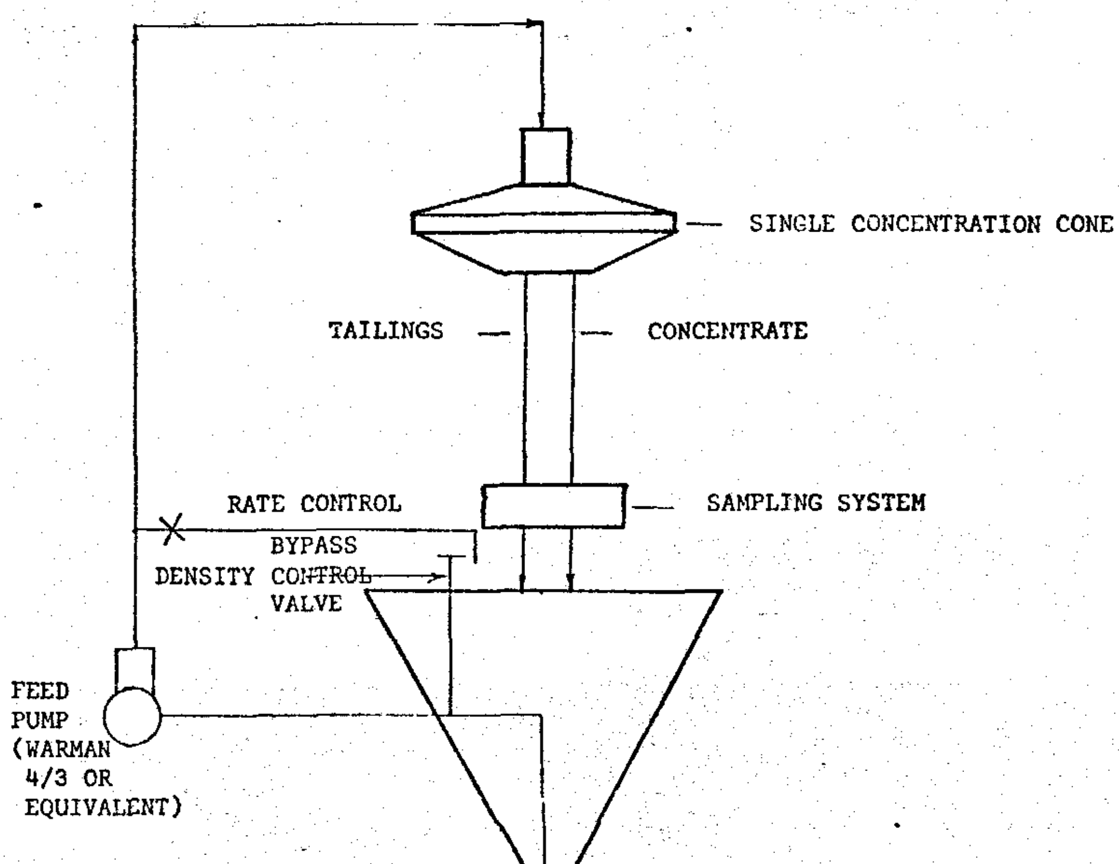
To relate heavy mineral to sulphur recovery, regression curves have been drawn. Four curves were drawn for the various stages of cone concentration testwork:-

1. Single Cone Testwork - Fig. 4.5A.
2. Rougher Cone Testwork - Fig. 4.5B.
3. Rougher Retreat Cone Testwork - Fig. 4.5C.
4. Cleaner Cone Testwork - Fig. 4.5D.

TABLE 1.1

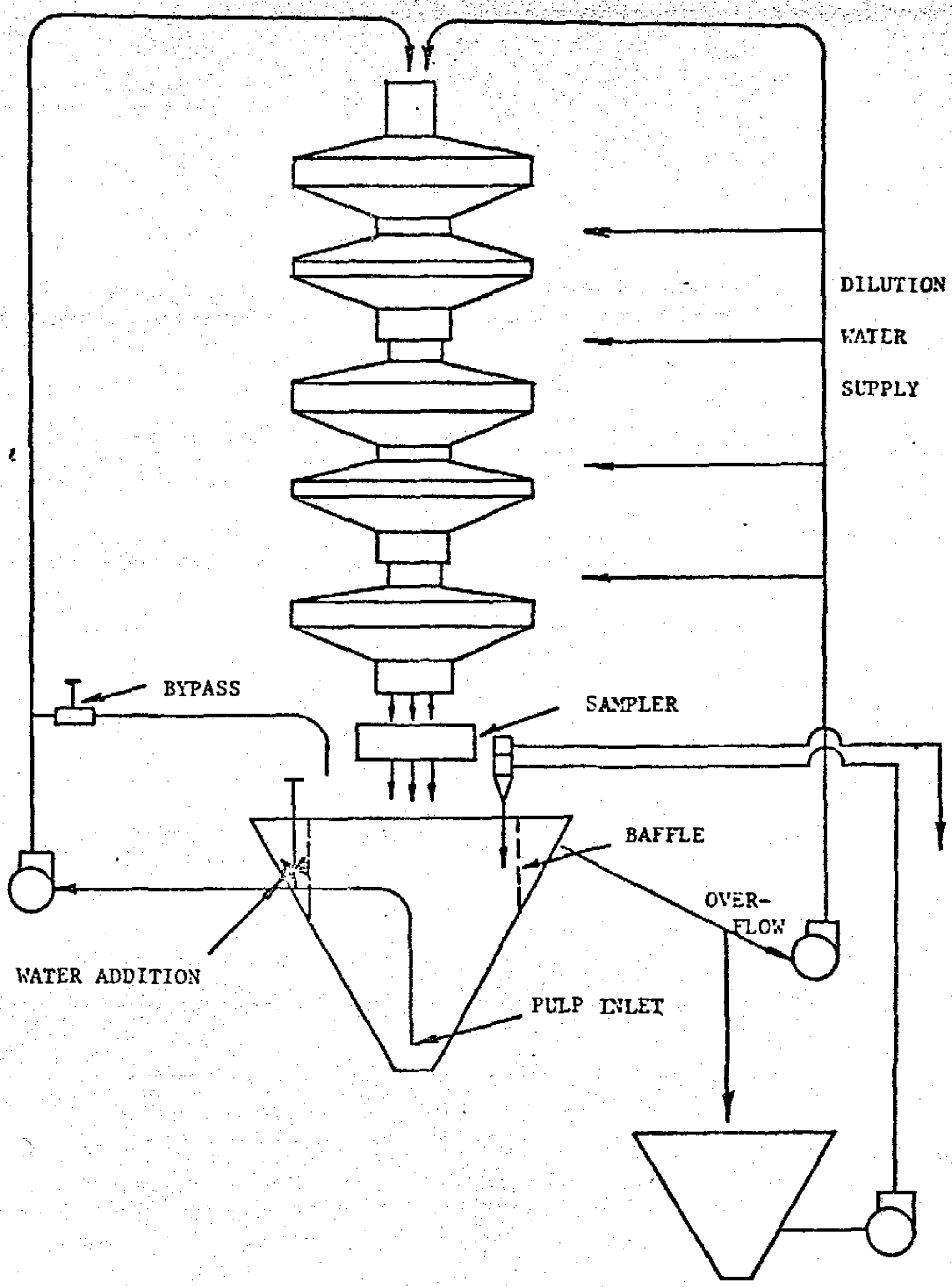
HEAD FEED SIZE - MACQUARIE HARBOUR SULPHIDE DEPOSIT - RESULTS

SIZE FRACTION	% WT.	% S	DIST.S	% Cu	DIST.Cu
300	7.45	0.90	1.63	0.21	9.57
250	6.94	0.80	1.35	0.17	7.21
210	14.27	0.80	2.77	0.17	14.83
178	20.04	0.70	3.41	0.15	19.60
150	9.29	1.40	3.16	0.15	8.52
124	14.76	2.60	9.32	0.15	13.53
105	8.51	6.80	18.01	0.16	8.32
90	6.45	10.90	17.07	0.16	6.31
75	3.52	16.40	14.01	0.16	3.44
63	2.98	17.40	12.39	0.14	2.51
-63	3.59	19.20	16.73	0.17	3.73
Cyclone 1	0.32	32.00	2.49	0.23	0.45
2	0.73	10.00	1.78	0.14	0.62
3	0.25	7.20	0.45	0.18	0.29
4	0.16	4.00	0.18	0.20	0.23
5	0.05	1.70	0.02	0.20	0.06
6	0.68	1.50	0.25	0.19	0.79
Head	100	4.12	100	0.16	100



RESEARCH AND ENGINEERING DEPARTMENT MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND	FIG. 2.1
SKETCH OF SINGLE CONE CONCENTRATOR CLOSED CIRCUIT TEST RIG	DRAWN: R.A.G. DATE: 10/9/74 SCALE: N.T.S.

261072 14



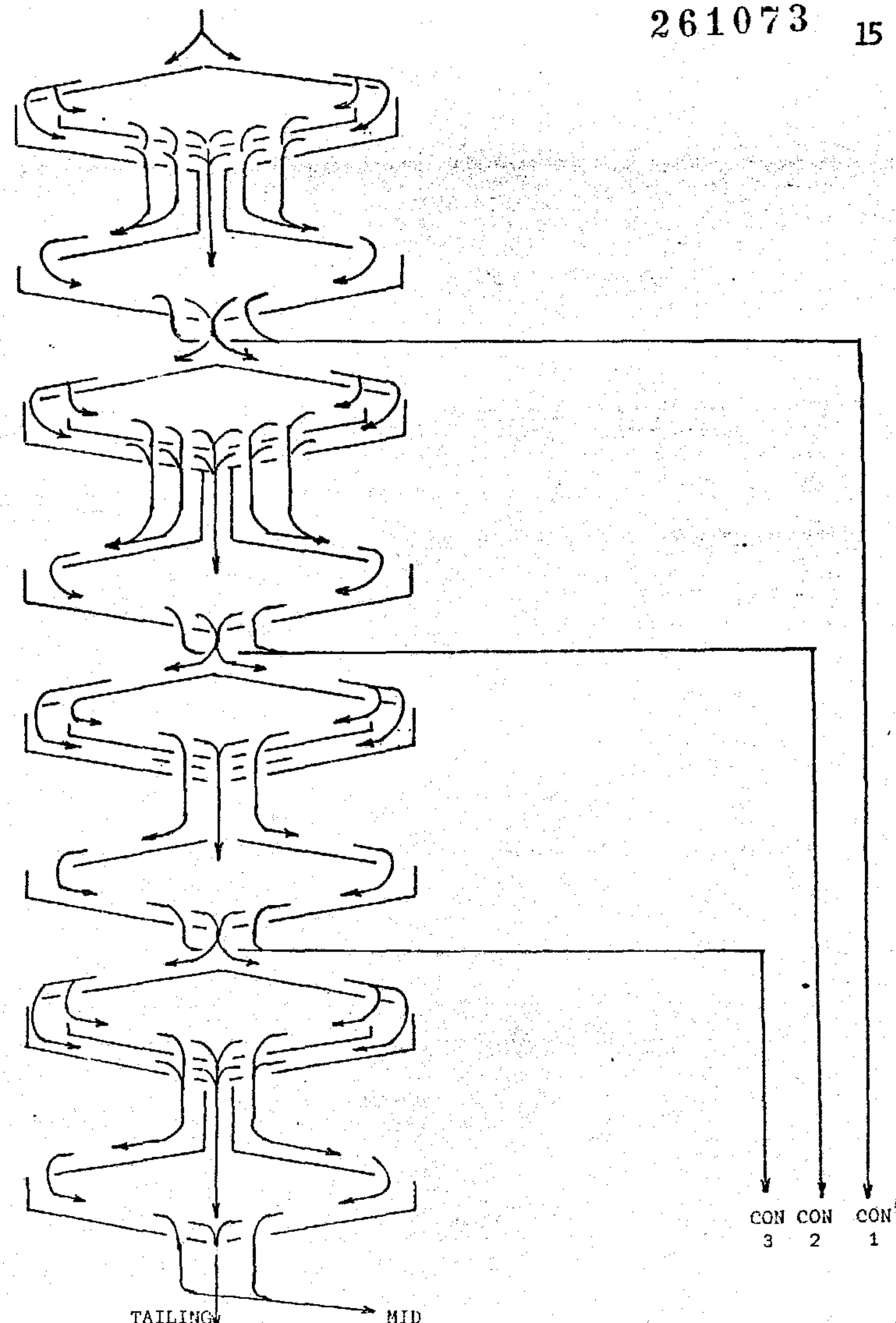
REICHERT MINING EQUIPMENT DIVISION
MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND

FIG. 2.2

CONCENTRATOR TEST RIG FEED RATE AND DENSITY CONTROL FLOW DIAGRAM

DRAWN	M.G.
DATE	16.2.73
SCALE	n.t.s.

261073 15



RESEARCH AND ENGINEERING DEPARTMENT
MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND

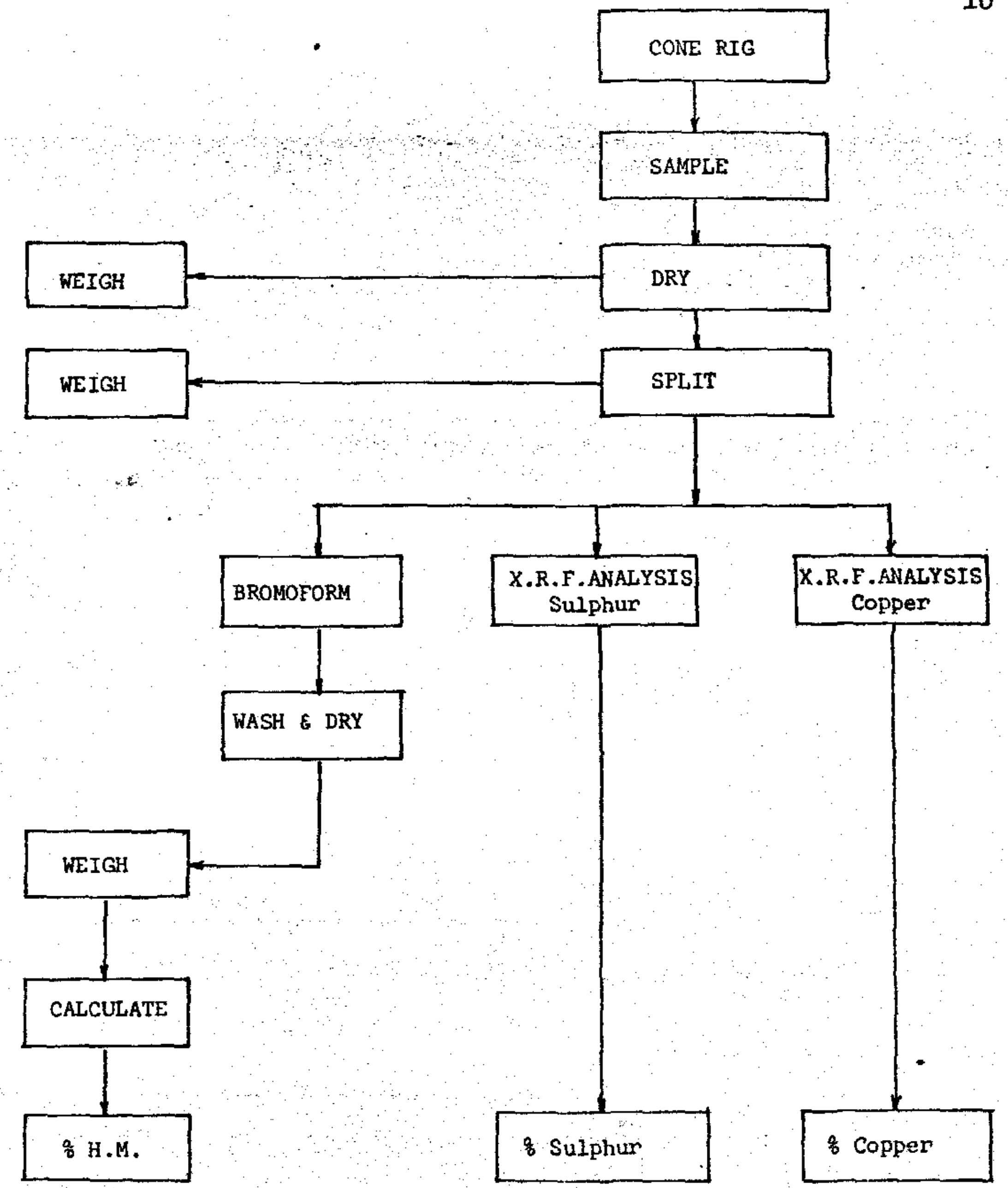
FIG. 2.3

4DS

DRAWN: M.V.A.

DATE: 16.4.75

SCALE: N.T.S.



RESEARCH AND ENGINEERING DEPARTMENT
MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND

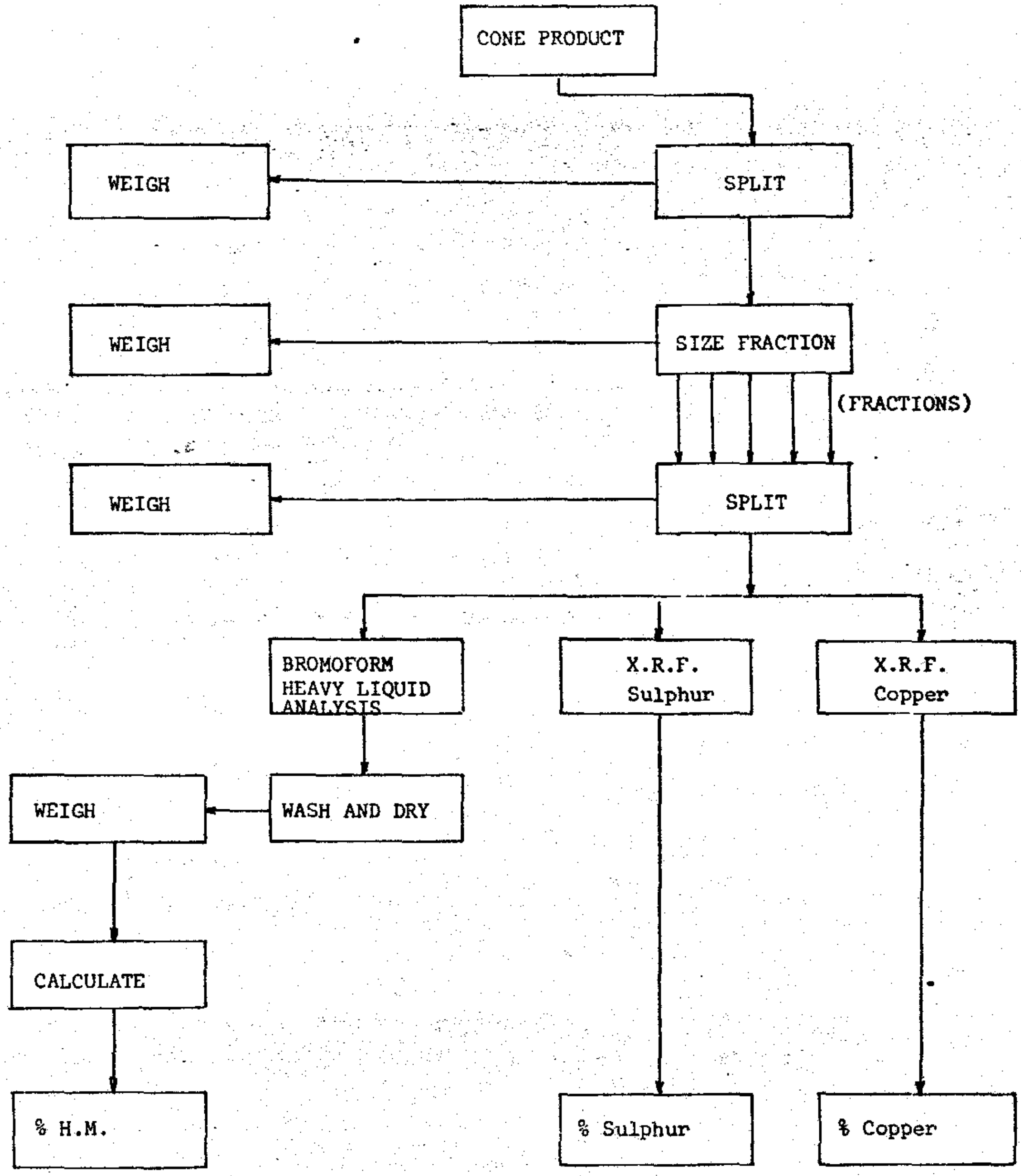
FIG. 2.4

CONE CONCENTRATOR PRODUCT ANALYSIS

DRAWN: M.S.C.

DATE: 29.4.76

SCALE: N.T.S.

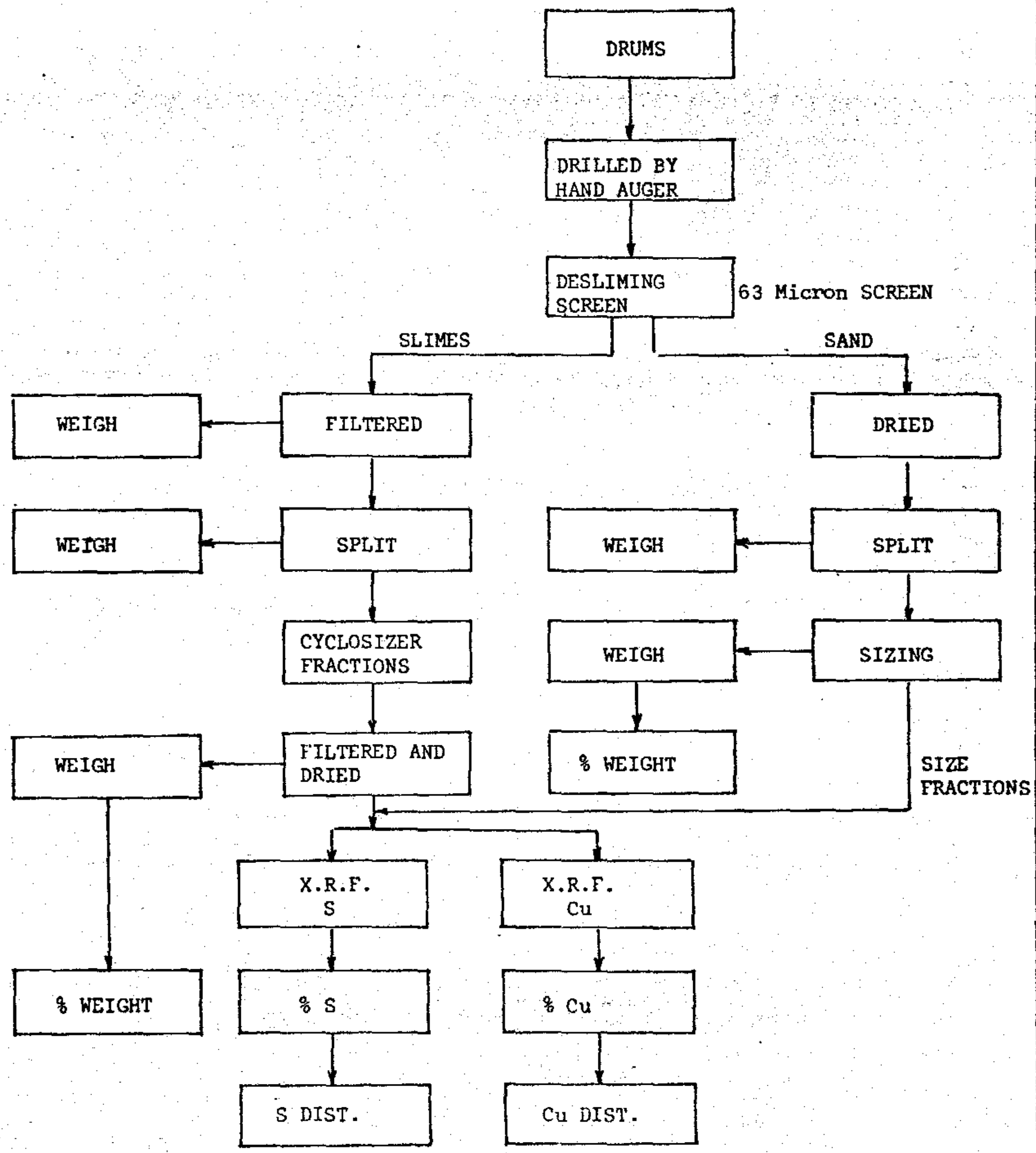


RESEARCH AND ENGINEERING DEPARTMENT
MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND

FIG. 2.5

PRODUCT SIZING AND ANALYSIS TO 63 MICRONS

DRAWN: M.S.C.
DATE: 30.4.76
SCALE: N.T.S.



RESEARCH AND ENGINEERING DEPARTMENT
MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND

FIG. 2.6

PRODUCT SIZING AND ANALYSIS INCLUDING CYCLOSIZER
(USED ON BULK SAMPLE HEAD FEED)

DRAWN: M.S.C.
DATE: 30.4.76
SCALE: N.T.S.

MINERAL DEPOSITS LIMITED, SOUTHPORT, QUEENSLAND, AUSTRALIA **TABLE 3.1** Form LRS 1 Issue 3

RESEARCH AND DEVELOPMENT LABORATORIES - TEST RESULTS AND SUMMARY SHEET

PROJECT.....A.C.S.............. TESTWORK...SINGLE CONE MK.III INSERT UNIT.......................... DATE..........

TEST NO.	HEAD FEED			TAILINGS				INSERT SETTINGS	CONCENTRATE				RECOVERY	UPGRADE RATIO
				SOLIDS		MINERAL			SOLIDS		MINERAL			
	TPH	GRADE % H.M.	DENSITY %	TPH	%DIST	GRADE % H.M.	%DIST		TPH	%DIST	GRADE % H.M.	%DIST		
1	39.1	18.13	55.8	21.60	55.2	14.55	44.3	+12	17.5	44.8	22.56	55.7	55.7	1.24
2	45.04	15.45	55.8	34.02	75.5	13.17	64.4	+ 4	11.02	24.5	22.54	35.6	35.6	1.46
3	45.03	18.08	55.5	42.57	94.5	17.13	89.6	- 4	2.46	5.5	34.41	10.4	10.4	1.90
4	43.72	16.56	53.2	24.47	56.0	13.64	46.1	+12	19.25	44.0	20.24	53.9	53.9	1.22
5	45.66	16.36	52.3	35.27	77.2	14.07	66.4	+ 4	10.39	22.8	24.11	33.6	33.6	1.47
6	43.87	14.79	51.5	41.99	95.7	13.90	90.0	- 4	1.88	4.3	34.58	10.0	10.0	2.34
7	43.48	18.77	51.1	24.60	56.6	15.72	47.4	+12	18.88	43.4	22.77	52.6	52.6	1.21
8	41.91	17.70	50.7	31.54	75.3	14.78	62.8	+ 4	10.37	24.7	26.66	37.2	37.2	1.51
9	41.35	18.62	49.7	39.90	96.5	17.90	92.7	- 4	1.45	3.5	38.71	7.3	7.3	2.08
10	41.56	33.49	55.1	23.26	57.2	27.2	46.6	+12	17.80	42.8	41.74	53.4	53.4	1.25
11	39.03	33.74	55.3	29.81	76.4	28.98	65.6	+ 4	9.22	23.6	49.14	34.4	34.4	1.46
12	38.34	33.91	54.3	36.55	95.3	32.50	91.4	- 4	1.79	4.7	62.35	8.6	8.6	1.87
13	30.61	17.8	58.5	16.20	52.9	13.67	40.6	+12	14.41	47.1	22.49	59.4	59.4	1.26
14	29.46	20.26	57.7	20.93	70.0	15.04	52.8	+ 4	8.53	30.0	33.04	47.2	47.2	1.63
15	35.36	20.79	57.0	32.36	91.5	18.78	82.8	- 4	9.00	8.5	42.11	17.2	17.2	2.03
16	28.77	17.87	56.8	14.67	51.9	11.18	31.9	+12	14.1	49.0	24.81	68.1	68.1	1.39
17	26.69	16.75	54.2	19.76	74.0	13.40	59.3	+ 4	6.93	26.0	26.23	40.7	40.7	1.57

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MINERAL DEPOSITS LIMITED, SOUTHPORT, QUEENSLAND, AUSTRALIA

TABLE 3.1 (CONT) Form LRS 1 Issue 3

RESEARCH AND DEVELOPMENT LABORATORIES - TEST RESULTS AND SUMMARY SHEET

PROJECT A.C.S. TESTWORK SINGLE CONE MK.III INSERT UNIT .... DATE ....

TEST NO.	HEAD FEED			TAILINGS				INSERT SETTINGS				CONCENTRATE				RECOVERY	UPGRADE RATIO
				SOLIDS		MINERAL						SOLIDS		MINERAL			
	TPH	GRADE % H.M.	DENSITY %	TPH	%DIST	GRADE % H.M.	%DIST					TPH	%DIST	GRADE % H.M.	%DIST		
18	27.92	16.44	53.6	26.24	94.0	14.97	85.6		- 4			1.68	6.0	39.28	14.4	14.4	2.39
19	24.06	15.09	54.1	11.60	48.3	10.83	34.7		+12			12.46	51.7	19.05	65.3	65.3	1.28
20	22.90	14.67	53.0	16.96	74.1	11.78	59.5		+ 4			5.94	25.9	22.83	40.5	40.5	1.56
21	23.61	14.61	52.0	21.75	92.1	13.44	84.6		- 4			1.86	7.9	28.40	15.4	15.4	1.94

MINERAL DEPOSITS LIMITED, SOUTHPORT, QUEENSLAND, AUSTRALIA

TABLE 3.2

Form L... 1
Issue 3

RESEARCH AND DEVELOPMENT LABORATORIES - TEST RESULTS AND SUMMARY SHEET

PROJECT......A.C.S................TESTWORK....S ANALYSIS...................UNIT....SINGLE CONE..........................DATE..........

TEST NO.	HEAD FEED			TAILINGS				INSERT SETTINGS	CONCENTRATE				RECOVERY	UPGRADE RATIO
				SOLIDS		MINERAL			SOLIDS		MINERAL			
	TPH	GRADE %.S..	DENSITY %	TPH	%DIST	GRADE %.S..	%DIST		TPH	%DIST	GRADE %.S..	%DIST		
1		2.81			55.2	2.00	39.3	+12		44.8	3.80	60.7	60.7	
2		2.57			75.5	2.13	62.5	+ 4		24.5	3.93	37.5	37.5	
3		2.71			94.5	2.42	84.4	- 4		5.5	7.67	15.6	15.6	
4		2.19			56.6	1.59	40.6	+12		44.0	2.96	59.4	59.4	
5		2.88			77.2	2.33	62.4	+ 4		22.8	4.76	37.6	37.6	
6		4.03			95.7	3.87	91.9	- 4		4.3	7.55	8.1	8.1	
7		3.27			56.6	2.79	48.3	+12		43.4	3.90	51.7	51.7	
8		3.41			75.3	2.89	63.8	+ 4		24.7	4.99	36.2	36.2	
9		3.29			96.5	3.07	90.1	- 4		3.5	9.26	9.9	9.9	
10		13.74			57.2	11.6	48.3	+12		42.8	16.6	51.7	51.7	
11		13.26			76.4	11.3	65.1	+ 4		23.6	19.6	34.9	34.9	
12		13.62			95.3	13.1	91.7	- 4		4.7	24.1	8.3	8.3	

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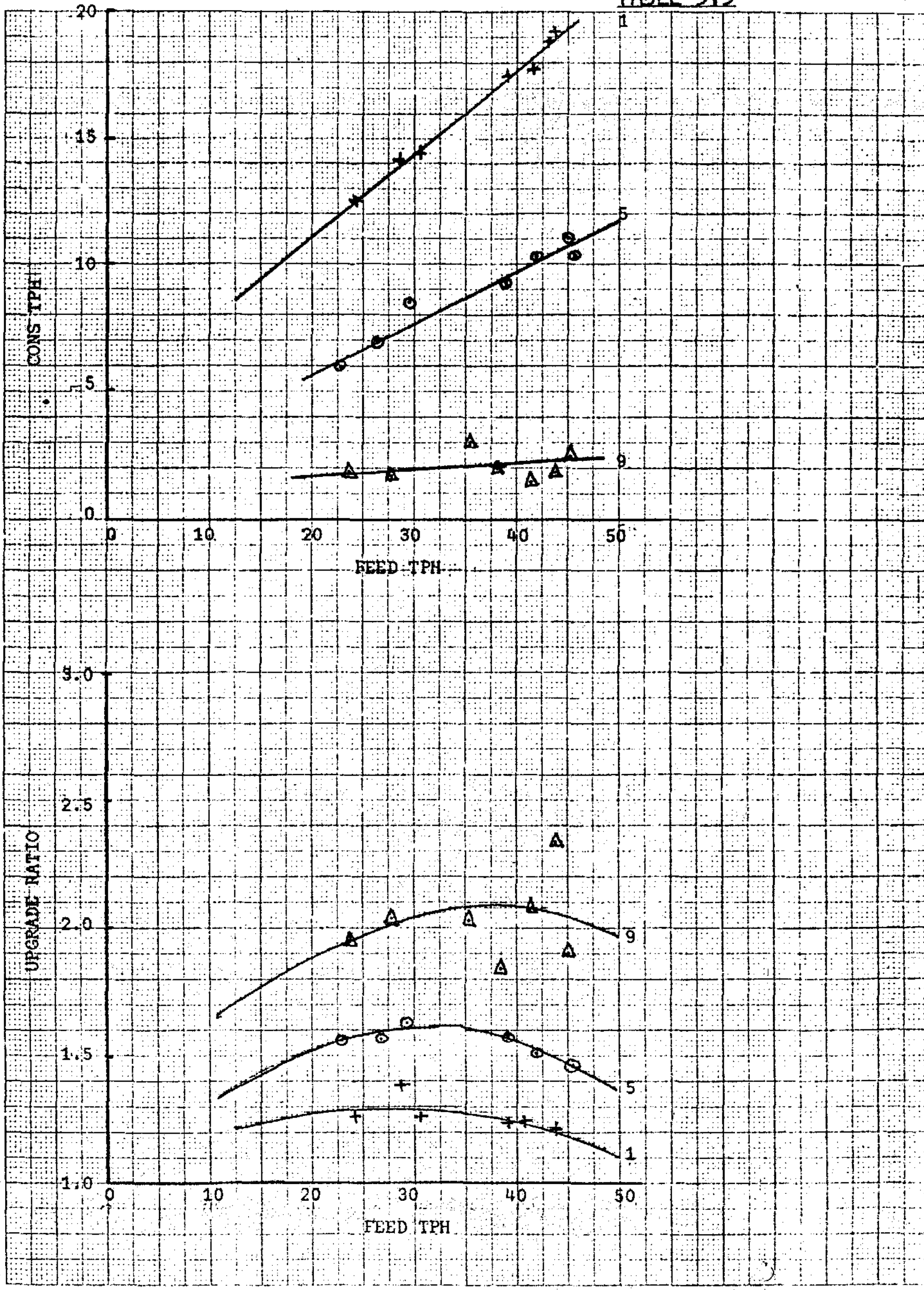
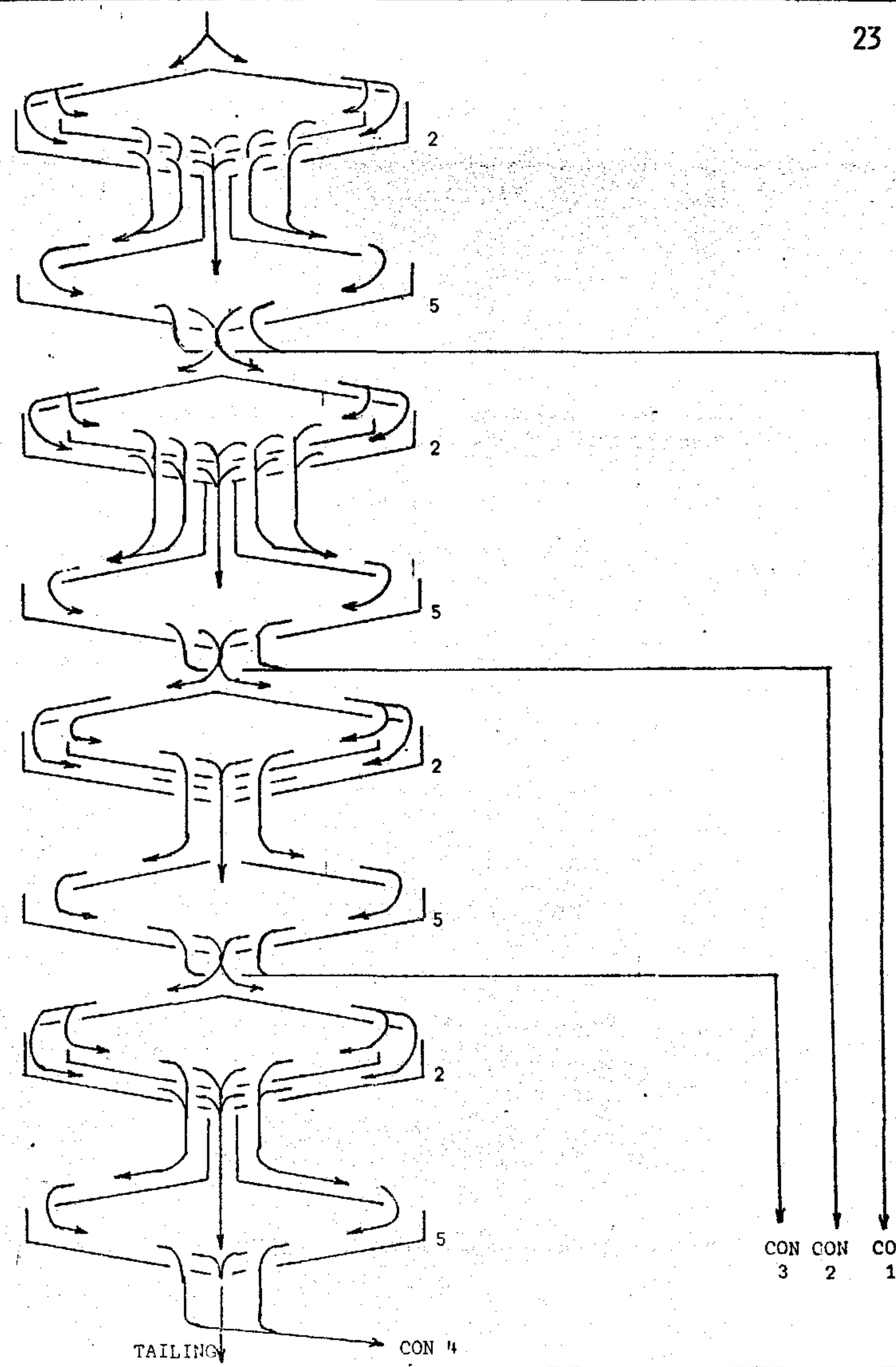
CITIES SERVICE SINGLE CONE TESTS

TABLE 3.3





RESEARCH AND ENGINEERING DEPARTMENT
MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND

FIG. 3.4

INSERT SETTINGS FOR TESTS 1-3

4DSV

DRAWN: M.V.A.

DATE: 16.4.75

SCALE: N.T.S.

MINERAL DEPOSITS LIMITED, SOUTHPORT, QUEENSLAND, AUSTRALIA

TABLE 3.5 — Form LPS Issue 3

RESEARCH AND DEVELOPMENT LABORATORIES - TEST RESULTS AND SUMMARY SHEET

PROJECT: AUSTRALIAN CITIES SERVICE ... TESTWORK: ROUGHER TESTS ... UNIT: 4DSV ... DATE: ....

TEST NO.	HEAD FEED			TAILINGS				CONCENTRATES 3 + 4				CONCENTRATES 1 + 2				RECOVERY	UPGRADE RATIO
				SOLIDS		MINERAL		SOLIDS		MINERAL		SOLIDS		MINERAL			
	TPH	GRADE % H.M.	DENSITY %	TPH	%DIST	GRADE % H.M.	%DIST	TPH	%DIST	GRADE % H.M.	%DIST	TPH	%DIST	GRADE % H.M.	%DIST		
1	44.89	21.59	58.0	15.12	33.6	9.88	15.4	13.96	31.1	17.0	24.3	15.81	35.3	36.9	60.3		
2	54.45	21.30	58.0	22.31	41.0	7.87	15.2	14.21	26.1	20.0	24.4	17.93	32.8	39.1	60.4		
3	69.15	21.29	58.0	33.7	48.7	9.83	22.5	15.68	22.7	21.3	22.7	19.77	28.6	40.8	54.8		
				TEST 3 WAS ALSO ASSAYED OF SULPHUR													
3	69.15	2.46	58.0	33.7	48.7	0.35	7.0	15.68	22.7	1.65	15.2	19.77	28.6	6.71	77.8		

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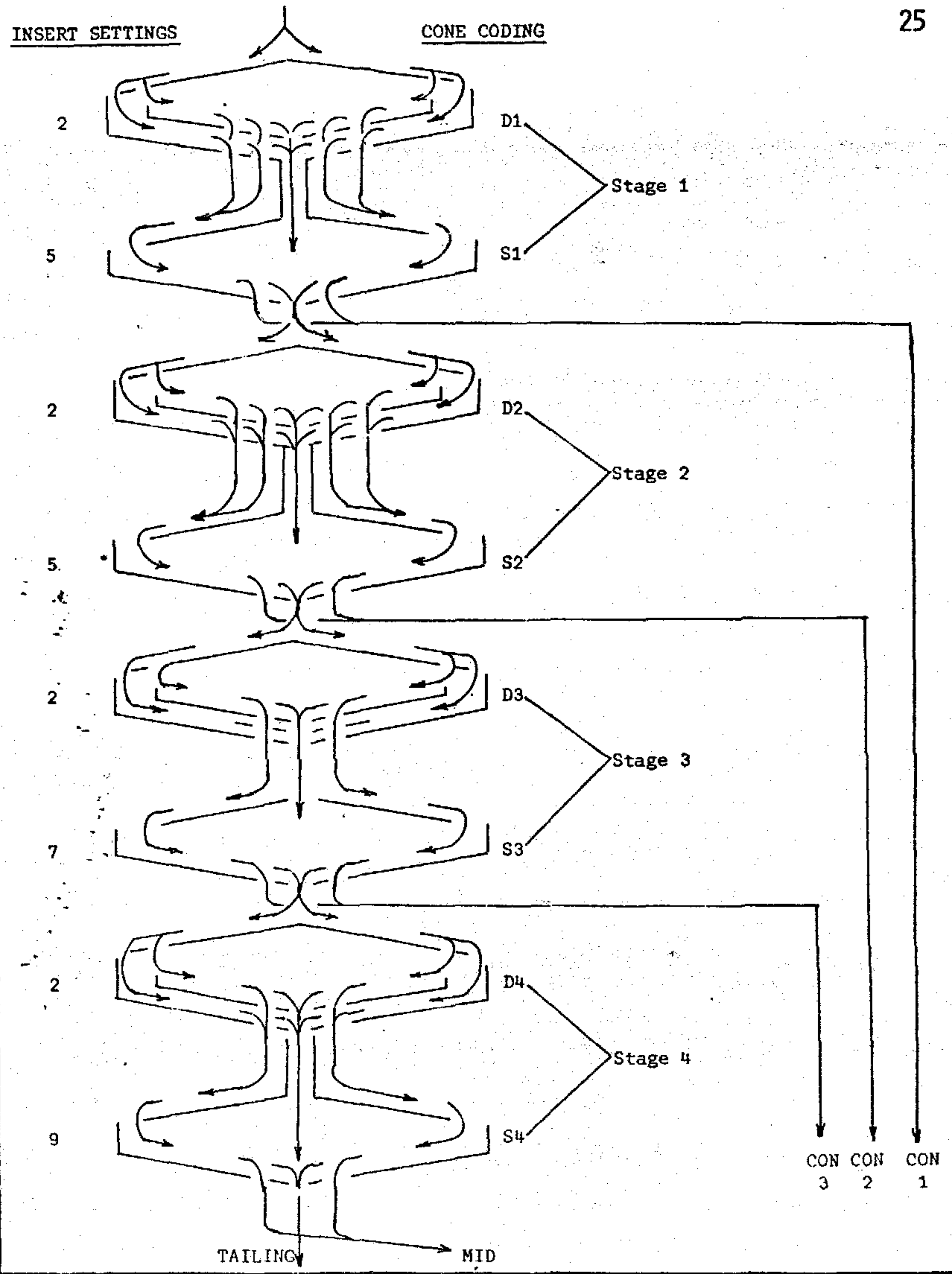

RESEARCH AND ENGINEERING DEPARTMENT
MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND

FIG. 3.6

4DS

DRAWN: M.V.A.

DATE: 16.4.75

SCALE: N.T.S.

MI AL C LI ED, H RT, EEN ND, STR A

Form LRS 1
Issue 3

RESEARCH AND DEVELOPMENT LABORATORIES - TEST RESULTS AND SUMMARY SHEET TABLE 3.7A

PROJECT. AUSTRALIAN CITIES SERVICE....TESTWORK.....ROUGHER...............UNIT.....4DSV....................DATE.........

TEST NO. 4 - 7	HEAD FEED			TAILINGS				MIDDLINGS				CONCENTRATE				RECOVERY	UPGRADE RATIO
				SOLIDS		MINERAL		SOLIDS		MINERAL		SOLIDS		MINERAL			
	TPH	GRADE %H.M.	DENSITY %	TPH	%DIST	GRADE %H.M.	%DIST	TPH	%DIST	GRADE %H.M.	%DIST	TPH	%DIST	GRADE %H.M.	%DIST		
STAGE 1																	
D1	106.70	18.76	57.5	78.38	73.5	16.45	64.8					28.32	26.5	24.89	35.2		1.33
S1	28.32	24.89	58.9	18.92	66.8	18.24	49.1					9.40	33.2	38.19	50.9		1.53
STAGE 2																	
D2	98.32	17.79	55.3	65.71	66.8	14.12	53.1					32.61	33.2	25.18	46.9		1.42
S2	32.61	25.18	59.4	20.95	64.2	18.16	46.3					11.66	35.8	37.82	53.7		1.50
STAGE 3																	
D3	84.61	15.10	53.7	57.09	67.5	12.57	56.1					27.32	32.5	20.39	43.9		1.35
S3	27.52	20.39	56.3	18.99	69.0	15.91	53.8					8.53	31.0	30.34	46.2		1.49
STAGE 4																	
D4	72.64	13.74	52.0	42.27	58.2	11.44	48.5					30.37	41.8	16.92	51.5		1.23
S4	30.37	16.92	58.2	26.11	86.0	14.90	75.7					4.26	14.0	29.33	24.3		1.73
OVERALL SUMMARY OF TESTS				4 - 7	- HEAVY MINERAL												
		FEED			TAILING				CONC. S3 & S4				CONC S1 AND 2				
4-7	102.27	20.12	57.5	68.38	66.9	12.77	42.4	12.79	12.5	30.00	18.7	21.06	20.6	37.98	37.9		

MINERAL DEPOSITS LIMITED, SOUTHPORT, QUEENSLAND,AUSTRALIA

Form LRS 1
Issue 3

RESEARCH AND DEVELOPMENT LABORATORIES - TEST RESULTS AND SUMMARY SHEET TABLE 3.7B

PROJECT..AUSTRALIAN CITIES SERVICE...TESTWORK...ROUGHER..................UNIT..4DSV.......................DATE..........

TEST NO. 4 - 7	HEAD FEED			TAILINGS				MIDDLINGS				CONCENTRATE				RECOVERY	UPGRADE RATIO
				SOLIDS		MINERAL		SOLIDS		MINERAL		SOLIDS		MINERAL			
	TPH	GRADE % S	DENSITY %	TPH	%DIST	GRADE % S	%DIST	TPH	%DIST	GRADE % S	%DIST	TPH	%DIST	GRADE % S	%DIST		
STAGE 1																	
D1	106.70	1.62	57.5	78.38	73.5	1.38	62.4					28.32	26.5	2.6	37.6		
S1	28.32	2.6	58.9	18.92	66.8	1.4	40.7					9.40	33.2	4.1	59.3		
STAGE 2																	
D2	98.32	1.56	58.3	65.71	66.8	1.04	44.6					32.61	33.2	2.6	55.4		
S2	32.61	2.60	59.4	20.95	64.2	1.2	29.6					11.66	35.8	5.1	70.4		
STAGE 3																	
D3	84.61	1.38	53.7	57.09	67.5	0.75	36.8					27.32	32.5	2.69	63.2		
S3	27.52	2.69	56.3	18.99	69.0	0.7	18.0					8.53	31.0	7.1	82.0		
STAGE 4																	
D4	72.64	0.64	52.0	42.27	58.2	0.4	36.3					30.37	41.8	0.98	63.7		
S4	30.37	0.98	58.2	26.11	86.0	0.5	43.8					4.26	14.0	3.9	56.2		
OVERALL SUMMARY OF TEST 4 - 7 - SULPHUR																	
		FEED			TAILING				CONS 3 + 4				CONC 1 + 2				
4-7	102.27	1.98	57.5	68.38	66.9	0.44	14.7	12.79	12.5	6.02	39.5	21.06	20.6	4.65	47.8		

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TABLE 3.7C Form LPS 1
Issue 3

RESEARCH AND DEVELOPMENT LABORATORIES - TEST RESULTS AND SUMMARY SHEET

PROJECT: AUSTRALIAN CITIES SERVICE TESTWORK: ROUGHER UNIT: 4DSV LOGIC DATE:

TEST NO. 8 - 11	HEAD FEED			TAILINGS				MIDDLINGS				CONCENTRATE				RECOVERY	UPGRADE RATIO
				SOLIDS		MINERAL		SOLIDS		MINERAL		SOLIDS		MINERAL			
	TPH	GRADE %.H.M.	DENSITY %	TPH	%DIST	GRADE %.H.M.	%DIST	TPH	%DIST	GRADE %.H.M.	%DIST	TPH	%DIST	GRADE %H.M.	%DIST		
D1	67.95	19.88	56.5	41.56	61.1	14.13	43.5					26.39	56.5	28.95	56.5		
S1	26.39	28.95	58.0	17.86	67.7	16.34	38.2					8.53	32.3	55.34	61.8		
D2	62.27	15.10	55.0	35.83	59.1	11.66	45.7					25.44	40.9	20.09	54.3		
S2	25.44	20.09	57.5	16.24	63.8	13.78	43.7					9.20	36.2	31.17	56.2		
D3	52.16	11.96	53.5	27.19	52.1	9.55	41.7					24.97	47.9	14.58	58.3		
S3	24.97	14.58	58.0	18.47	74.0	11.78	59.9					6.50	26.0	22.43	40.1		
D4	39.04	10.55	51.0	16.68	42.7	8.52	34.4					22.38	57.3	12.06	65.6		
S4	22.38	12.06	59.0	18.75	83.8	10.45	72.6					3.63	16.2	20.11	27.4		
			OVERALL SUMMARY OF TESTS 8-11 - HEAVY MINERAL														
		FEED			TAILING				CONS 3 + 4				CONS 1 + 2				
8-11	63.35	20.79	56.5	35.43	55.9	9.54	25.7	10.19	16.1	21.6	16.7	17.73	28.0	42.8	57.6		

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[illegible] LIMITED, [illegible], TRA[illegible]

TABLE 5./D

Form [illegible]PS
Issue 3

RESEARCH AND DEVELOPMENT LABORATORIES - TEST RESULTS AND SUMMARY SHEET

PROJECT.........A.C.S.............TESTWORK...ROUGHER.....................UNIT...........4DSV LOGIC..........DATE..........

TEST NO. 8-11	HEAD FEED			TAILINGS				MIDDLINGS				CONCENTRATE				RECOVERY	UPGRADE RATIO
				SOLIDS		MINERAL		SOLIDS		MINERAL		SOLIDS		MINERAL			
	TPH	GRADE %.S..	DENSITY %	TPH	%DIST	GRADE %.S..	%DIST	TPH	%DIST	GRADE %.S..	%DIST	TPH	%DIST	GRADE %.S..	%DIST		
D1	67.95	2.01	56.5	41.56	61.1	1.2	37.2					26.39	56.5	3.2	62.8		
S1	26.39	3.2	58.0	17.86	67.7	1.5	31.2					8.53	32.3	6.9	68.8		
D2	62.27	1.2	55.0	35.83	59.1	0.8	37.8					25.44	40.9	1.8	62.2		
S2	25.44	1.8	57.5	16.24	63.8	1.0	34.9					9.20	36.2	3.3	65.1		
D3	52.16	0.8	53.5	27.19	52.1	0.7	36.9					24.97	47.9	1.01	63.1		
S3	24.97	1.01	58.0	18.47	74.0	0.7	51.1					6.50	26.0	1.9	48.9		
D4	39.04	0.63	51.0	16.68	42.7	0.5	33.9					22.38	57.3	0.73	66.1		
S4	22.98	0.73	59.0	18.75	83.8	0.5	57.6					3.63	16.2	1.9	42.4		
		OVERALL SUMMARY OF TESTS 8-11 - SULPHUR															
		FEED			TAILINGS				CONS 3 + 4				CONS 1 + 2				
8-11	63.35	2.00	56.5	35.43	55.9	0.5	14.0	42.54	16.1	1.9	15.3	17.73	28.0	5.0	70.7		

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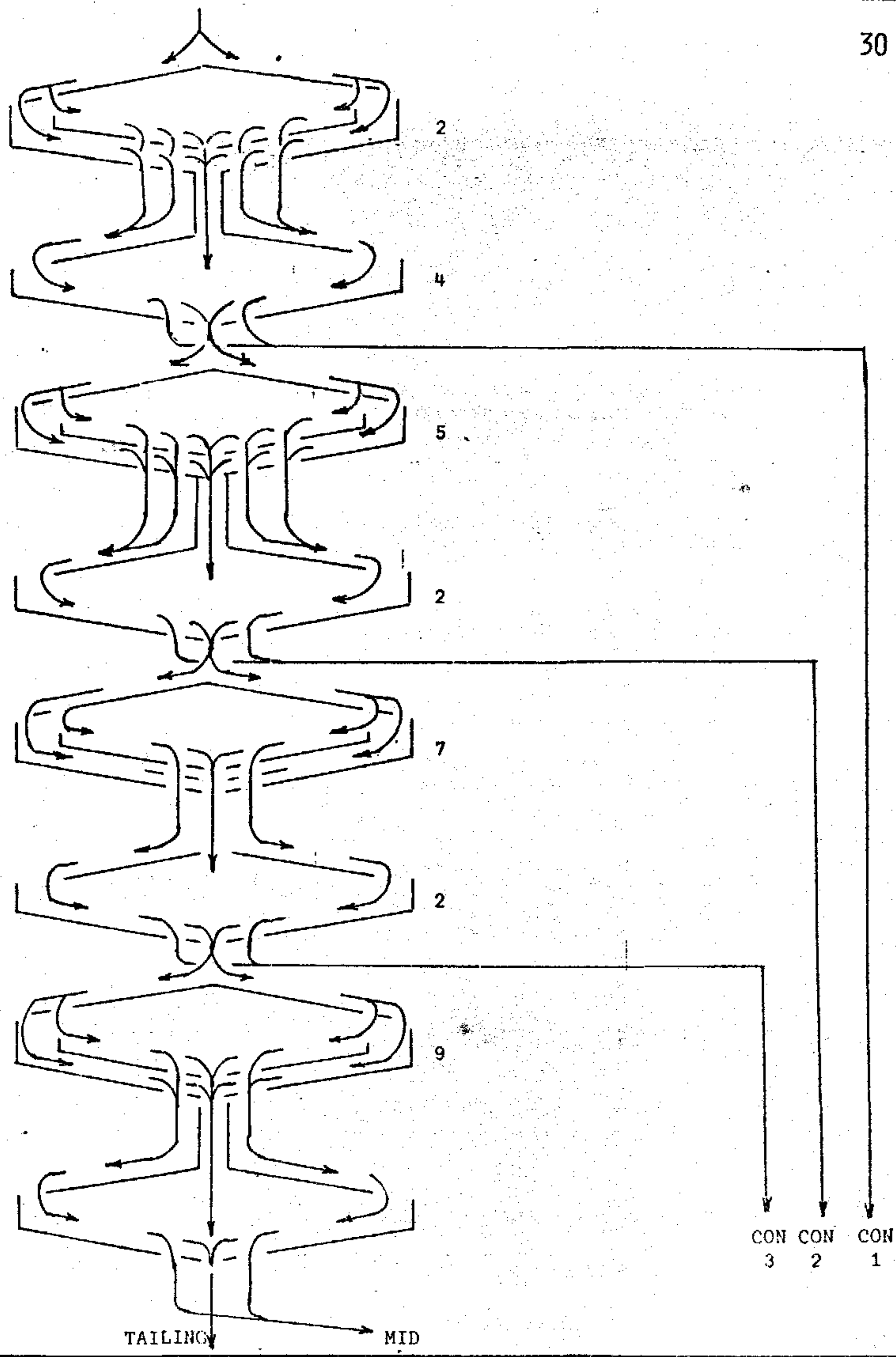

RESEARCH AND ENGINEERING DEPARTMENT
MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND

FIG.3.8

4DS
INSERT SETTINGS FOR ROUGHER TEST 12 - 14

DRAWN: M.V.A.
DATE: 16.4.75
SCALE: N.T.S.

MINERAL DEP[illegible] LIMITED, [illegible] D, AUSTRA[illegible]

TABLE [illegible]A

[illegible] Issue 3

RESEARCH AND DEVELOPMENT LABORATORIES - TEST RESULTS AND SUMMARY SHEET

PROJECT...AUSTRALIA CITIES SERVICE...TESTWORK....ROUGHER................UNIT...4DSV........................DATE..........

TEST NO.	HEAD FEED			TAILINGS				MIDDLINGS				CONCENTRATE				RECOVERY	UPGRADE RATIO
				SOLIDS		MINERAL		SOLIDS		MINERAL		SOLIDS		MINERAL			
	TPH	GRADE % H.M.	DENSITY %	TPH	%DIST	GRADE % H.M.	%DIST	TPH	%DIST	GRADE % H.M.	%DIST	TPH	%DIST	GRADE % H.M.	%DIST		
	RESULTS FOR HEAVY MINERAL																
12	63.04	22.91	57.5	33.96	53.9	12.09	28.5	9.53	15.1	24.0	15.8	20.55	31.0	41.1	55.7		
13	76.55	20.56	57.5	44.95	58.7	11.26	32.1	10.06	13.1	27.2	17.4	21.54	28.2	36.9	50.5		
14	68.26	19.78	57.5	38.02	55.7	10.83	30.5	9.98	14.6	22.6	16.7	20.26	29.7	35.2	52.8		
	RESULTS FOR SULPHUR																
12	63.04	2.62	57.5	33.96	53.9	0.6	12.4	9.53	15.1	1.8	10.4	20.55	31.0	6.5	77.2		
13	76.55	2.75	57.5	44.95	58.7	0.7	14.9	10.06	13.1	2.8	13.5	21.54	28.2	7.0	71.6		
14	68.26	2.20	57.5	38.02	55.7	0.6	16.2	9.98	14.6	1.4	10.1	20.26	29.7	5.1	73.7		
	RESULTS FOR COPPER																
12	63.04	0.18	57.5	33.96	53.9	0.15	45.3	9.53	15.1	0.19	15.7	20.55	31.0	0.23	39.0		
13	76.55	0.17	57.5	44.95	58.7	0.15	51.2	10.06	13.1	0.19	14.2	21.54	28.2	0.21	34.6		
14	68.26	0.19	57.5	38.02	55.7	0.19	54.3	9.98	14.6	0.18	13.5	20.26	29.7	0.21	32.2		

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TABLE 3.9B

TEST 13

ALL SIZES ARE MICRON SIZES

CON 1

SIZE mm	% WT.	CUM.%	% S	S DIST.	CUM.S DIST.	% Cu	DIST. Cu	CUM.Cu DIST.
+300	5.88	5.88	1.6	1.43	1.43	0.30	8.08	8.08
+212	18.10	23.97	1.0	2.75	4.18	0.23	19.06	27.14
+150	31.00	54.97	2.2	10.36	14.54	0.20	28.40	55.14
+106	26.53	81.5	7.1	28.63	43.17	0.21	25.52	81.06
+75	12.56	94.06	18.1	34.53	77.70	0.23	13.23	94.29
-75	5.94	100	24.7	22.30	100	0.21	5.71	100

CON 2

SIZE mm	% WT.	CUM.%	% S	S DIST.	CUM.S DIST.	% Cu	DIST. Cu	CUM.Cu DIST.
+300	7.03	7.03	1.30	1.81	1.81	0.24	8.79	8.79
+212	22.92	29.94	1.3	5.9	7.71	0.21	25.08	33.86
+150	31.56	61.50	1.2	7.5	15.2	0.19	31.24	65.11
+106	23.87	85.37	5.7	26.93	42.13	0.17	21.14	86.25
+75	9.91	95.27	19.2	37.65	79.78	0.19	9.81	96.06
-75	4.73	100	21.6	20.22	100	0.16	3.94	100

CON 3

SIZE mm	% WT.	CUM.%	% S	S DIST.	CUM.S DIST.	% Cu	DIST. Cu	CUM.Cu DIST.
+300	6.23	6.23	1.1	2.13	2.13	0.23	8.15	8.15
+212	22.47	28.71	1.0	6.99	9.13	0.15	19.18	27.33
+150	34.23	62.93	0.8	8.52	17.64	0.19	37.0	64.33
+106	24.39	87.32	2.8	21.24	38.89	0.16	22.2	86.53
+ 75	8.43	95.74	12.2	31.98	70.87	0.18	8.63	95.16
- 75	4.26	100	22.0	29.13	100	0.20	4.84	100

TABLE 3.9B CONT.

CON 4

SIZE mm	% WT.	CUM.%	% S	S DIST.	CUM.S DIST.	% Cu	DIST. Cu	CUM.Cu DIST.
+300	6.36	6.36	1.6	3.82	3.82	0.22	8.11	8.11
+212	23.31	29.66	0.8	7.0	10.81	0.20	27.01	35.12
+150	34.55	64.21	0.8	10.37	21.18	0.16	32.03	67.15
+106	23.23	87.44	2.1	18.3	39.49	0.15	20.19	87.34
+75	8.15	95.60	9.2	28.14	67.63	0.16	7.56	94.9
-75	4.40	100	19.6	32.37	100	0.20	5.10	100

TAILINGS

SIZE mm	% WT.	CUM.%	% S	S DIST.	CUM.S DIST.	% Cu	DIST. Cu	CUM.Cu DIST.
+300	17.88	17.88	0.9	19.32	19.32	0.15	20.56	20.56
+212	33.15	51.04	0.6	23.88	43.20	0.14	35.57	56.13
+150	29.64	80.68	0.6	21.35	64.55	0.12	27.26	83.39
+106	13.77	94.45	0.7	11.57	76.12	0.11	11.61	94.99
+75	3.44	97.88	1.6	6.60	82.73	0.11	2.90	97.89
-75	2.12	100	6.8	17.27	100	0.13	2.11	100

TABLE 3.9C

SIZE RECOVERY OF SULPHUR IN CONE PRODUCTS

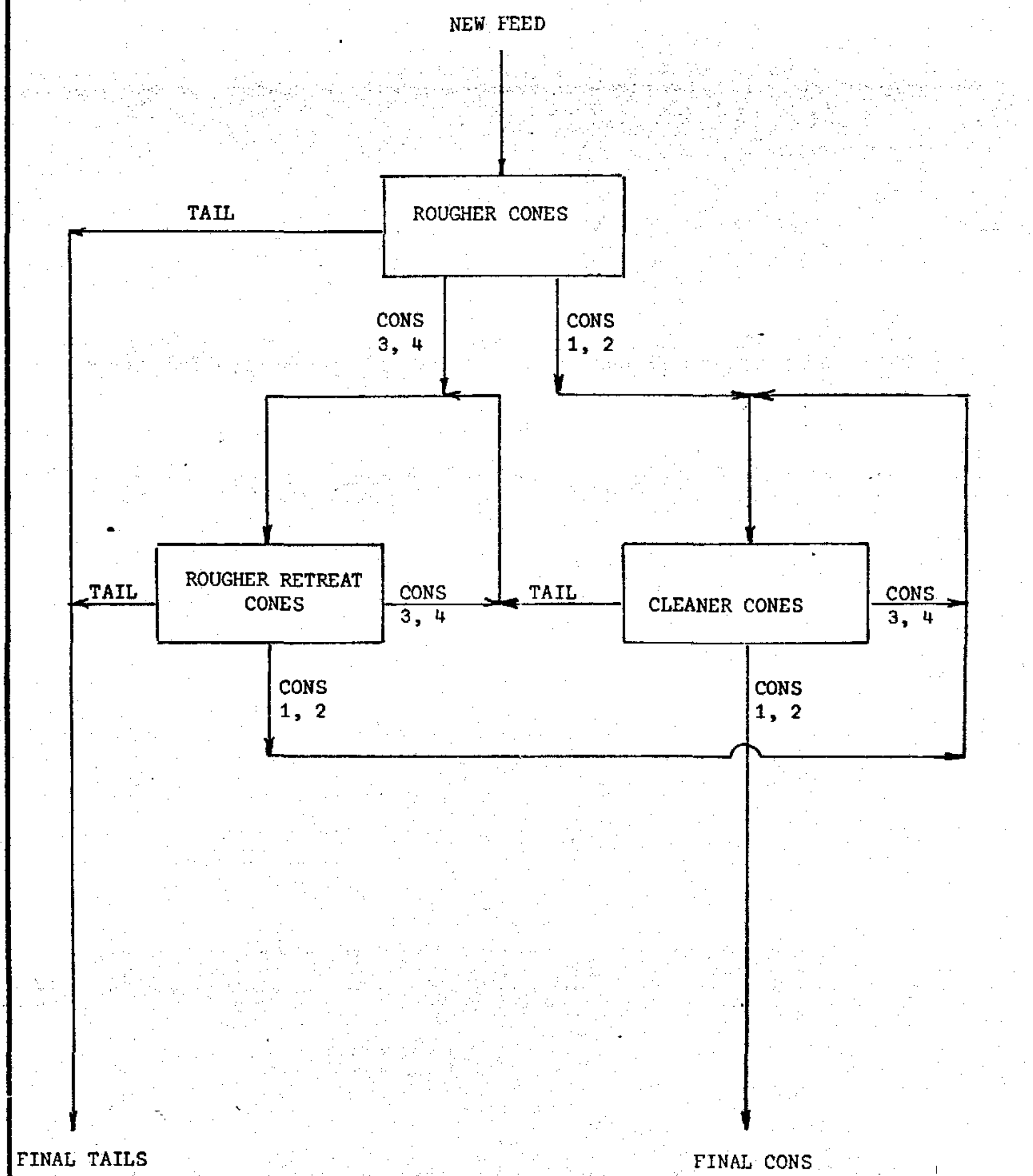
SIZE	CON 1	CON 2	CON 3	CON 4	TAILS	SULPHUR FEED DIST. SIZE
+300	10.77	9.14	4.60	3.37	72.12	3.12
+212	12.82	18.58	9.4	3.80	55.38	7.20
+150	34.9	17.04	8.26	4.07	35.67	12.74
+106	46.97	29.98	10.07	3.50	9.47	32.05
+75	45.69	33.55	12.12	4.36	4.23	24.41
-75	39.6	29.46	13.98	2.92	14.04	20.47

SIZE RECOVERY OF COPPER IN CONE PRODUCTS

SIZE	CON 1	CON 2	CON 3	CON 4	TAILS	Cu FEED DIST. SIZE
+300	11.78	9.84	5.66	2.70	70.08	15.33
+212	13.89	14.13	6.64	4.48	60.85	28.79
+150	20.09	17.14	12.43	5.16	45.18	27.67
+106	30.22	19.45	12.52	5.44	32.37	18.41
+75	39.73	22.72	12.24	5.19	20.13	6.12
-75	33.18	17.74	13.59	6.79	28.7	3.69

WEIGHT DISTRIBUTION HEAD FEED

SIZE	% WT.	CUM.% WT.
+300	7.46	7.46
+212	21.21	29.67
+150	29.33	58.00
+106	23.27	81.27
+75	9.97	91.24
-75	8.76	100



RESEARCH AND ENGINEERING DEPARTMENT **MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND**	FIG. 3.10
AUSTRALIAN CITIES SERVICE INITIAL FLOWSHEET	DRAWN: M.S.C. DATE: 9.2.76 SCALE: N.T.S.

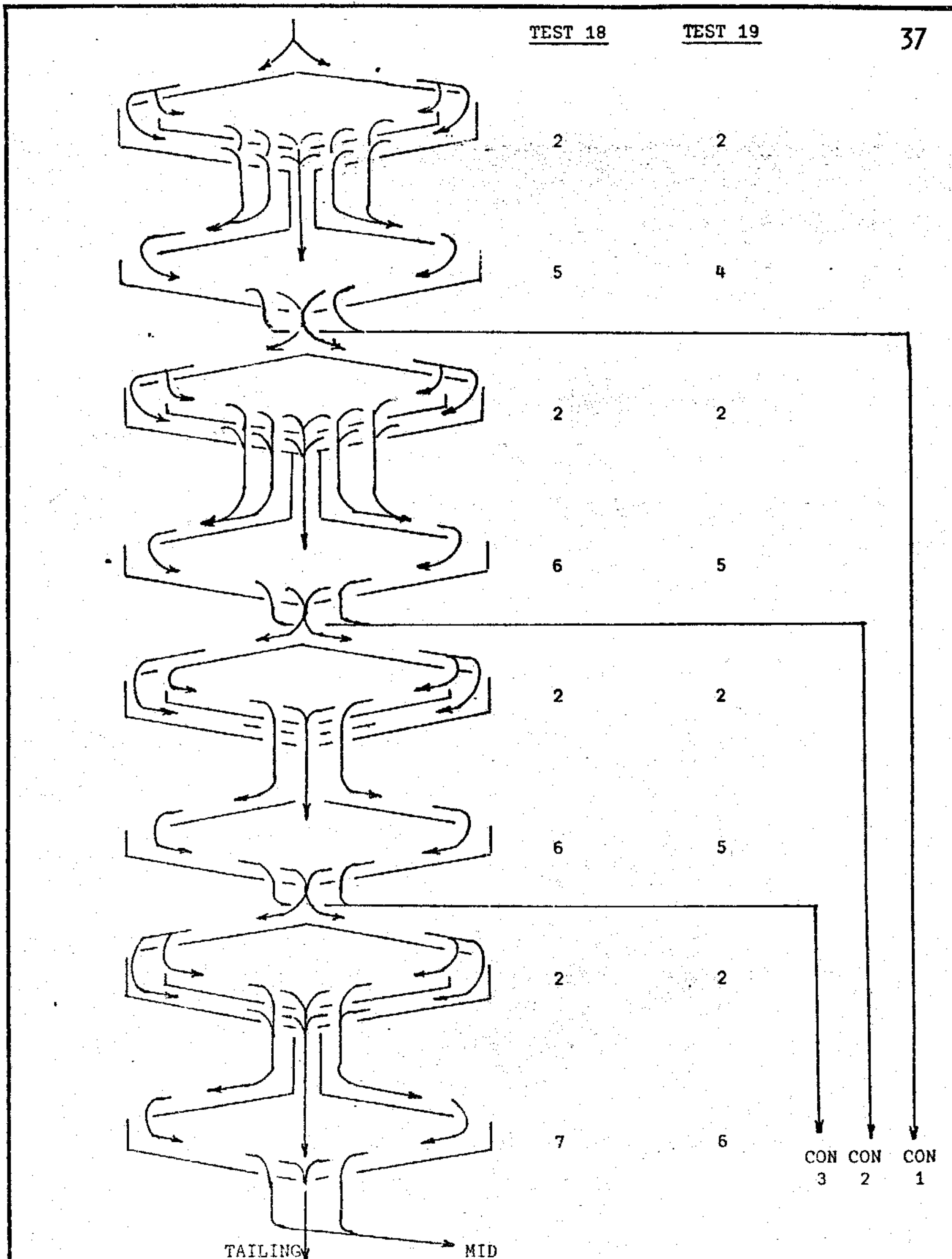
M[illegible] LIMITED, [illegible], [illegible]ND, [illegible]STR[illegible]A

TABLE 3.11 Form LRS 1 Issue 3

RESEARCH AND DEVELOPMENT LABORATORIES - TEST RESULTS AND SUMMARY SHEET

PROJECT........A.C.S.................TESTWORK.....ROUGHER RETREAT.........UNIT....4DSV.....................DATE..........

TEST NO.	HEAD FEED			TAILINGS				CONCENTRATES 3 & 4				CONCENTRATES 1 & 2				RECOVERY	UPGRADE RATIO
				SOLIDS		MINERAL		SOLIDS		MINERAL		SOLIDS		MINERAL			
	TPH	GRADE % H.M.	DENSITY %	TPH	%DIST	GRADE % H.M.	%DIST	TPH	%DIST	GRADE % H.M.	%DIST	TPH	%DIST	GRADE % H.M.	%DIST		
		SUMMARY OF CONES TESTS HEAVY MINERAL															
15	69.48	25.32	57.5	38.24	55.1	16.43	35.8	10.37	14.9	30.9	18.1	20.83	30.0	38.9	46.1		
17	61.22	24.86	57.8	32.75	53.5	14.73	31.7	9.59	15.7	30.5	19.2	18.88	30.8	39.6	49.1		
		SUMMARY OF CONES TESTS SULPHUR															
15	69.48	2.41	57.5	38.24	55.1	0.7	16.0	10.37	14.9	3.1	18.9	20.83	30.0	5.2	65.1		
17	61.22	2.56	57.8	32.75	53.5	0.7	14.6	9.59	15.7	2.3	14.1	18.88	30.8	5.9	71.3		



RESEARCH AND ENGINEERING DEPARTMENT
MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND

FIG. 3.12

CLEANER CONE TESTWORK

4DS

INSERT-SETTINGS

DRAWN: M.V.A.

DATE: 16.4.75

SCALE: N.T.S.

MINERAL DEPOSITS LIMITED, SOUTHPORT, QUEENSLAND, AUSTRALIA

TABLE 3.13

Form LFS 1
Issue 3

RESEARCH AND DEVELOPMENT LABORATORIES - TEST RESULTS AND SUMMARY SHEET

PROJECT...AUSTRALIA CITIES SERVICE TESTWORK...CLEANER CONE TESTS.........UNIT..4DSV.........................DATE..........

TEST NO.	HEAD FEED			TAILINGS				CONCENTRATES 3 & 4				CONCENTRATES 1 & 2				RECOVERY	UPGRADE RATIO
				SOLIDS		MINERAL		SOLIDS		MINERAL		SOLIDS		MINERAL			
	TPH	GRADE %....	DENSITY %	TPH	%DIST	GRADE %....	%DIST	TPH	%DIST	GRADE %....	%DIST	TPH	%DIST	GRADE %....	%DIST		
		RESULTS FOR HEAVY MINERAL															
18	72.42	37.12	58.0	35.12	48.5	17.35	22.7	15.70	21.7	40.00	23.4	21.6	29.8	67.1	53.9		
19	78.97	35.68	58.0	34.66	43.9	15.30	18.8	18.24	23.1	45.0	29.2	26.13	33.0	56.1	52.0		
		RESULTS FOR SULPHUR															
18	72.42	6.14	58.0	35.12	48.5	0.8	6.3	15.70	21.7	4.6	21.7	21.6	29.8	15.9	77.4		
19	78.97	6.55	58.0	34.66	43.9	0.7	4.8	18.24	23.1	4.7	16.4	26.13	33.0	15.6	78.8		
		RESULTS FOR COPPER															
18	72.42	0.19	58.0	35.12	48.5	0.15	38.9	15.70	21.7	0.20	22.9	21.6	29.8	0.24	38.2		
19	78.97	0.18	58.0	34.66	43.9	0.13	31.8	18.24	23.1	0.19	23.8	26.13	33.0	0.24	44.4		

[illegible] LIMITED, [illegible]

TABLE 3.14A

Form LRS 1
Issue 3

RESEARCH AND DEVELOPMENT LABORATORIES - TEST RESULTS AND SUMMARY SHEET

PROJECT: AUSTRALIA CITIES SERVICE ..... TESTWORK: CLEANER CONE ..... UNIT: 4DSV ..... DATE: ....

TEST NO. 20-23	HEAD FEED			TAILINGS				MIDDLINGS				CONCENTRATE				RECOVERY	UPGRADE RATIO
				SOLIDS		MINERAL		SOLIDS		MINERAL		SOLIDS		MINERAL			
	TPH	GRADE % H.M.	DENSITY %	TPH	%DIST	GRADE % H.M.	%DIST	TPH	%DIST	GRADE % H.M.	%DIST	TPH	%DIST	GRADE % H.M.	%DIST		
D1	72.73	37.07	58.0	40.69	56.0	29.6	44.6					32.04	44.0	46.8	55.4		
S1	32.04	46.8	64.0	18.86	58.9	32.55	41.4					13.18	41.1	66.6	58.8		
D2	63.31	28.21	56.5	33.33	52.6	19.0	35.4					29.98	47.4	38.5	64.6		
S2	29.98	38.5	64.0	17.84	59.5	25.64	39.6					12.14	40.5	57.36	60.4		
D3	55.9	21.97	54.0	27.15	48.6	16.3	36.1					28.75	51.4	27.3	63.9		
S3	28.75	27.3	57.6	17.28	60.1	18.69	41.2					11.47	39.9	40.30	58.8		
D4	40.45	15.95	52.5	18.18	44.9	13.0	36.6					22.27	55.1	18.11	63.4		
S4	22.27	18.1	58.6	16.59	74.5	15.46	62.6					5.68	25.5	26.92	37.4		
	OVERALL SUMMARY OF HEAVY MINERAL RESULT																
				TAILING				CONCENTRATES 3 & 4				CONCENTRATES 1 & 2					
20-23	77.24	34.7	58.0	34.77	45.0	14.2	18.4	17.15	22.2	35.8	22.9	25.32	32.8	62.2	58.7		
24	73.82	33.95	58.0	31.28	42.4	12.11	15.1	16.53	22.4	32.0	21.1	26.01	35.4	61.4	63.8		

MINERAL DEPOSITS LIMITED, SOUTHPORT, QUEENSLAND, AUSTRALIA

TABLE 3.14B

Form LPS 1
Issue 3

RESEARCH AND DEVELOPMENT LABORATORIES - TEST RESULTS AND SUMMARY SHEET

PROJECT. AUSTRALIA CITIES SERVICE....TESTWORK...CLEANER STAGE.............UNIT...4DSV......................DATE.........

TEST	HEAD FEED			TAILINGS				MIDDLINGS				CONCENTRATE				RECOVERY	UPGRADE RATIO
				SOLIDS		MINERAL		SOLIDS		MINERAL		SOLIDS		MINERAL			
NO. 20-23	TPH	GRADE % S	DENSITY %	TPH	%DIST	GRADE % S	%DIST	TPH	%DIST	GRADE % S	%DIST	TPH	%DIST	GRADE % S	%DIST		
D1	72.73	6.27	58.0	40.69	56.0	3.4	30.4					32.04	44.0	9.9	69.6		
S1	32.04	9.9	64.0	18.86	58.9	4.4	22.1					13.18	41.1	17.8	77.9		
D2	63.31	3.52	56.5	33.33	52.6	1.4	21.2					29.98	47.4	5.85	78.8		
S2	29.98	5.85	64.0	17.84	59.5	2.4	24.5					12.14	40.5	10.9	75.5		
D3	55.9	1.85	54.0	27.15	48.6	0.98	25.7					28.75	51.4	2.68	74.3		
S3	28.75	2.68	57.6	17.28	60.1	1.2	26.9					11.47	39.9	4.9	73.1		
D4	40.45	1.19	52.5	18.18	44.9	0.9	33.8					22.27	55.1	1.44	66.2		
S4	22.27	1.44	58.0	16.59	74.5	0.9	46.7					5.68	25.5	3.0	53.3		
	OVERALL SUMMARY OF SULPHUR RESULTS																
20-23	77.24	6.10	58.0	34.77	45.0	0.9	6.6	17.15	22.2	4.3	15.6	25.32	32.8	14.5	77.8		
24	73.82	6.0	58.0	31.28	42.4	0.6	4.2	16.53	22.4	3.2	12.0	26.01	35.4	14.2	83.8		40

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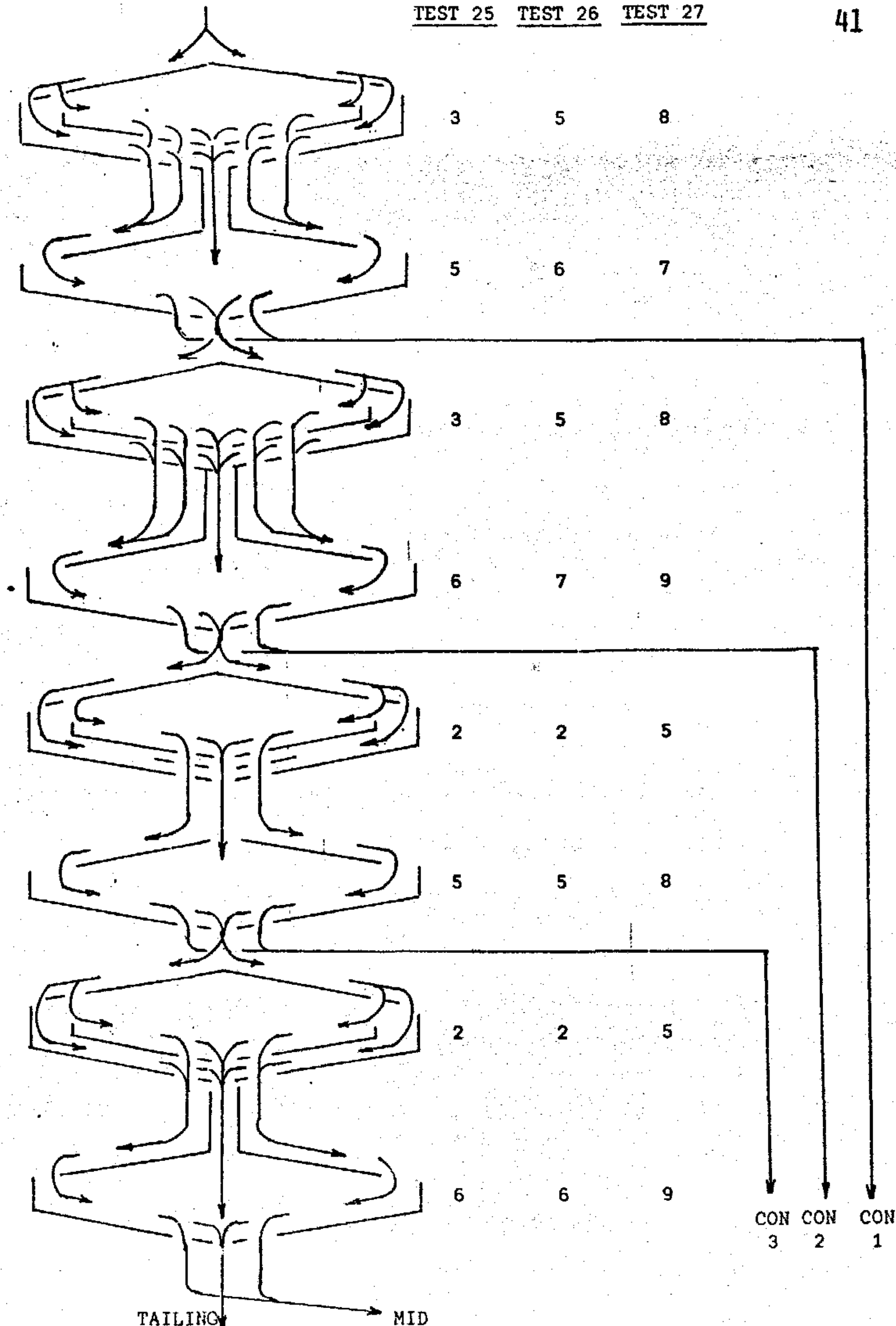

TEST 25	TEST 26	TEST 27
3	5	8
5	6	7
3	5	8
6	7	9
2	2	5
5	5	8
2	2	5
6	6	9

RESEARCH AND ENGINEERING DEPARTMENT MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND	FIG. 3.15
INSERT SETTINGS FOR TESTS 25 - 27 4DS	DRAWN: M.V.A. DATE: 16.4.75 SCALE: N.T.S.

MINERAL DEPOSITS LIMITED, ... , AUSTRALIA

TABLE 3.16

Form LPS 1
Issue 3

RESEARCH AND DEVELOPMENT LABORATORIES - TEST RESULTS AND SUMMARY SHEET

PROJECT A.C.S. TESTWORK CLEANER STAGE UNIT 4DSV DATE

TEST NO.	HEAD FEED			TAILINGS				CONCENTRATES 3 & 4				CONCENTRATES 1 & 2				RECOVERY	UPGRADE RATIO
				SOLIDS		MINERAL		SOLIDS		MINERAL		SOLIDS		MINERAL			
	TPH	GRADE % H.M.	DENSITY %	TPH	%DIST	GRADE %.H.M.	%DIST	TPH	%DIST	GRADE % H.M.	%DIST	TPH	%DIST	GRADE %.H.M.	%DIST		
		RESULT FOR H.M.															
25	70.35	33.62	57.5	36.89	52.4	13.51	21.0	11.99	17.1	38.2	19.4	21.47	30.5	65.6	59.6		
26	62.59	31.27	58.0	29.92	47.8	13.55	20.7	17.43	27.9	31.7	28.2	15.24	24.1	65.7	51.1		
27	54.39	33.63	57.5	33.63	61.8	16.33	30.0	10.37	19.1	53.1	30.1	10.39	19.1	70.2	39.9		
		RESULTS FOR SULPHUR															
25	70.35	6.33	57.5	36.89	52.4	1.1	9.1	11.99	17.1	4.7	12.8	21.47	30.5	16.2	78.1		
26	62.59	6.38	58.0	29.92	47.8	0.9	6.8	17.43	27.9	3.86	16.9	15.24	24.1	20.0	76.3		
27	54.39	7.37	57.5	33.63	61.8	1.20	10.1	10.37	19.1	11.9	30.7	10.39	19.1	22.8	59.2		

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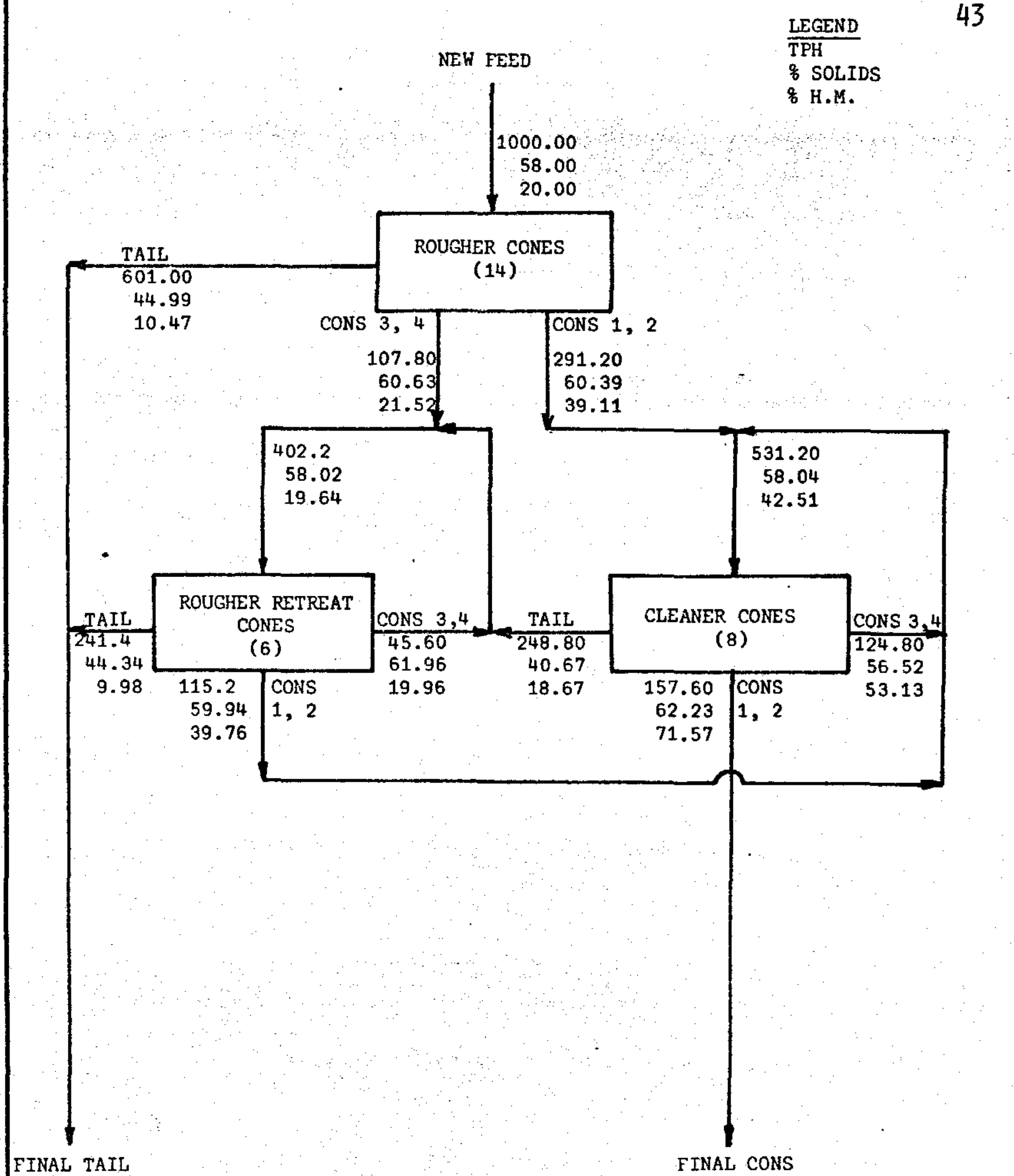
261100

099



RESEARCH AND ENGINEERING DEPARTMENT **MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND**	FIG. 4.1
AUSTRALIAN CITIES SERVICE - 1000 TPH INITIAL FLOWSHEET	DRAWN: M.S.C. DATE: 9.2.76 SCALE: N.T.S.

TABLE 4.2

CIRCUIT STUDY INSERT SETTINGS

Stage	Rougher Cone	Rougher Retreat Cone	Cleaner Cone
D1	2	2	2
S1	4	5	5
D2	2	2	2
S2	5	5	6
D3	2	2	2
S3	7	7	7
D4	2	2	2
S4	9	9	9

TABLE 4.3

CIRCUIT STUDY - STAGE RESULT - HEAVY MINERAL AND SULPHUR RESULT FROM MODELLING AND REGRESSION CURVES

Rougher Cones

PRODUCT	TPH	TPH PER CONE	% WT.	% H.M.	% S	DIST. H.M.	DIST. S
Cons 1 & 2	291.2	20.8	29.12	39.11	5.8	56.94	75.85
Cons 3 & 4	107.8	7.7	10.78	21.52	2.2	11.60	10.65
Tail	601.0	42.9	60.10	10.47	0.5	31.46	13.50
Feed	1000.0	71.4		20.00	2.23	100	100

Rougher Retreat Cones

PRODUCT	TPH	TPH PER CONE	% WT.	% H.M.	% S	DIST. H.M.	DIST. S
Cons 1 & 2	115.2	19.2	28.64	39.76	5.6	57.98	79.64
Cons 3 & 4	45.6	7.6	11.34	19.96	1.5	11.52	8.44
Tail	241.4	40.2	60.02	9.98	0.4	30.50	11.92
Feed	402.2	62.0		19.64	2.01	100	100

Cleaner Cones

PRODUCT	TPH	TPH PER CONE	% WT.	% H.M.	% S	DIST. H.M.	DIST. S
Cons 1 & 2	157.6	19.7	29.67	71.57	16.2	49.95	64.63
Cons 3 & 4	124.8	15.6	23.49	53.13	9.2	29.36	29.07
Tail	248.8	31.1	46.84	18.77	1.0	20.68	6.30
Feed	531.2	66.4		42.51	7.44	100	100

Cone Circuit Results

PRODUCT	TPH	% WT.	% H.M.	% S	DIST. H.M.	DIST. S
Final Concentrate	157.6	15.8	71.57	16.2	56.5	86.6
Final Tailing R	601.0	60.1	10.47	0.5	31.5	10.2
Final Tailing RR	241.4	24.1	9.98	0.4	12.0	3.2
Head Feed	1000.0	100.0	20.00	2.95	100	100

TABLE 4.4

CIRCUIT STUDY - STAGE RESULTS - HEAVY MINERAL AND SULPHUR FROM ACTUAL TESTWORK

Rougher Cones (Test 13)

PRODUCT	TPH (PER CONE)	% WT.	% H.M.	% S	DIST. H.M.	DIST. S
Cons 1 & 2	21.54	28.2	36.9	6.5	50.5	71.6
Cons 3 & 4	10.06	13.1	27.2	2.7	17.4	13.5
Tailing	44.95	58.7	11.26	0.7	32.1	14.9
Feed	76.55	100	20.56	2.75	100	100

Rougher Retreat Cones (Test 15)

PRODUCT	TPH (PER CONE)	% WT.	% H.M.	% S	DIST. H.M.	DIST. S
Cons 1 & 2	20.83	30.0	38.9	5.2	46.1	65.1
Cons 3 & 4	10.37	14.9	30.9	3.1	18.1	18.9
Tailing	38.28	55.1	16.43	0.7	35.8	16.0
Feed	69.48	100	25.32	2.41	100	100

Cleaner Cones (Test 18)

PRODUCT	TPH (PER CONE)	% WT.	% H.M.	% S	DIST. H.M.	DIST. S
Cons 1 & 2	21.6	29.8	67.1	15.9	53.9	77.4
Cons 3 & 4	15.7	21.7	40.0	4.6	23.4	16.3
Tailing	35.12	48.5	17.35	0.8	22.7	6.3
Feed	72.42	100	37.12	6.14	100	100

FIG. 4.5A

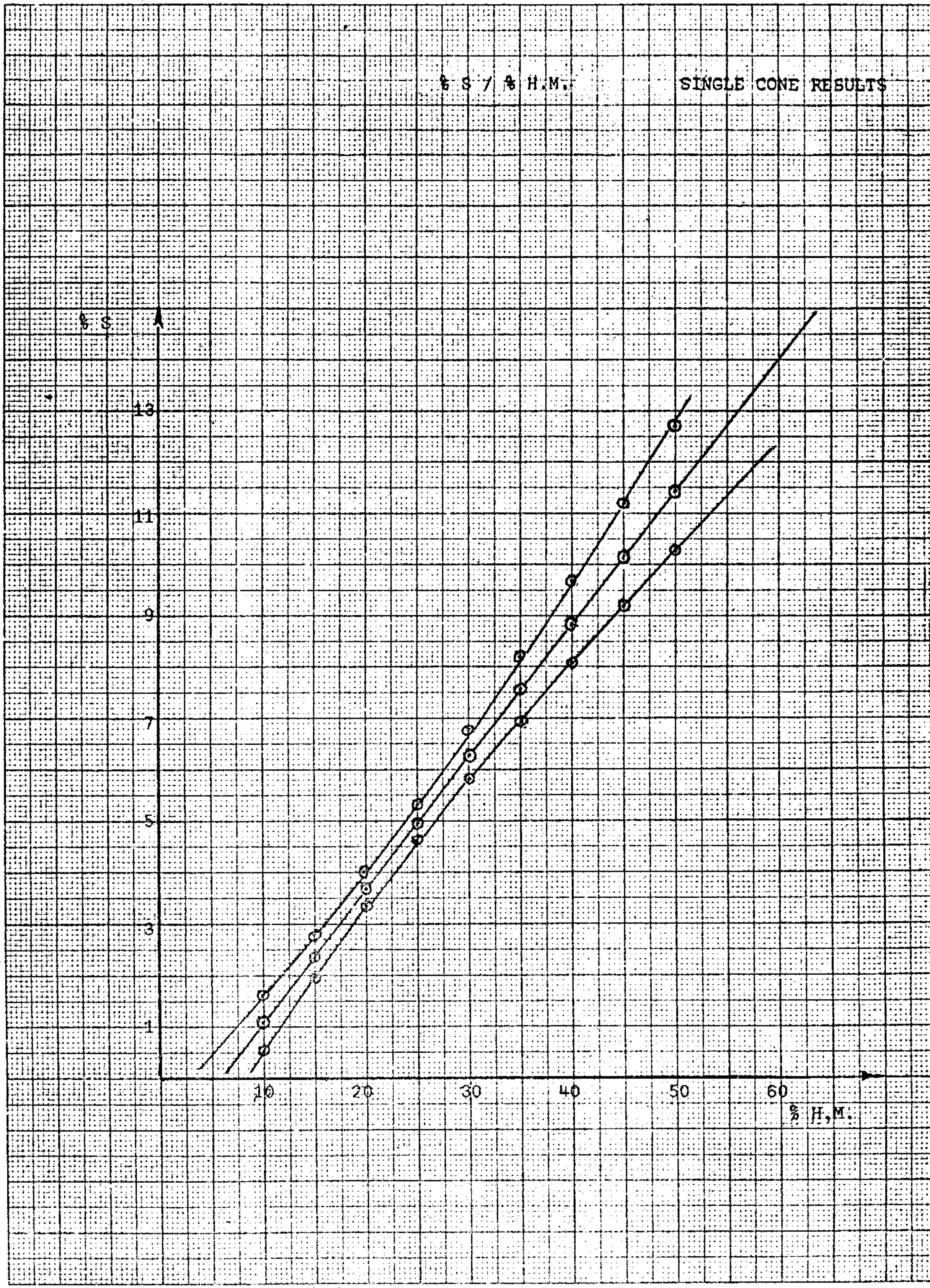


FIG. 4.5B

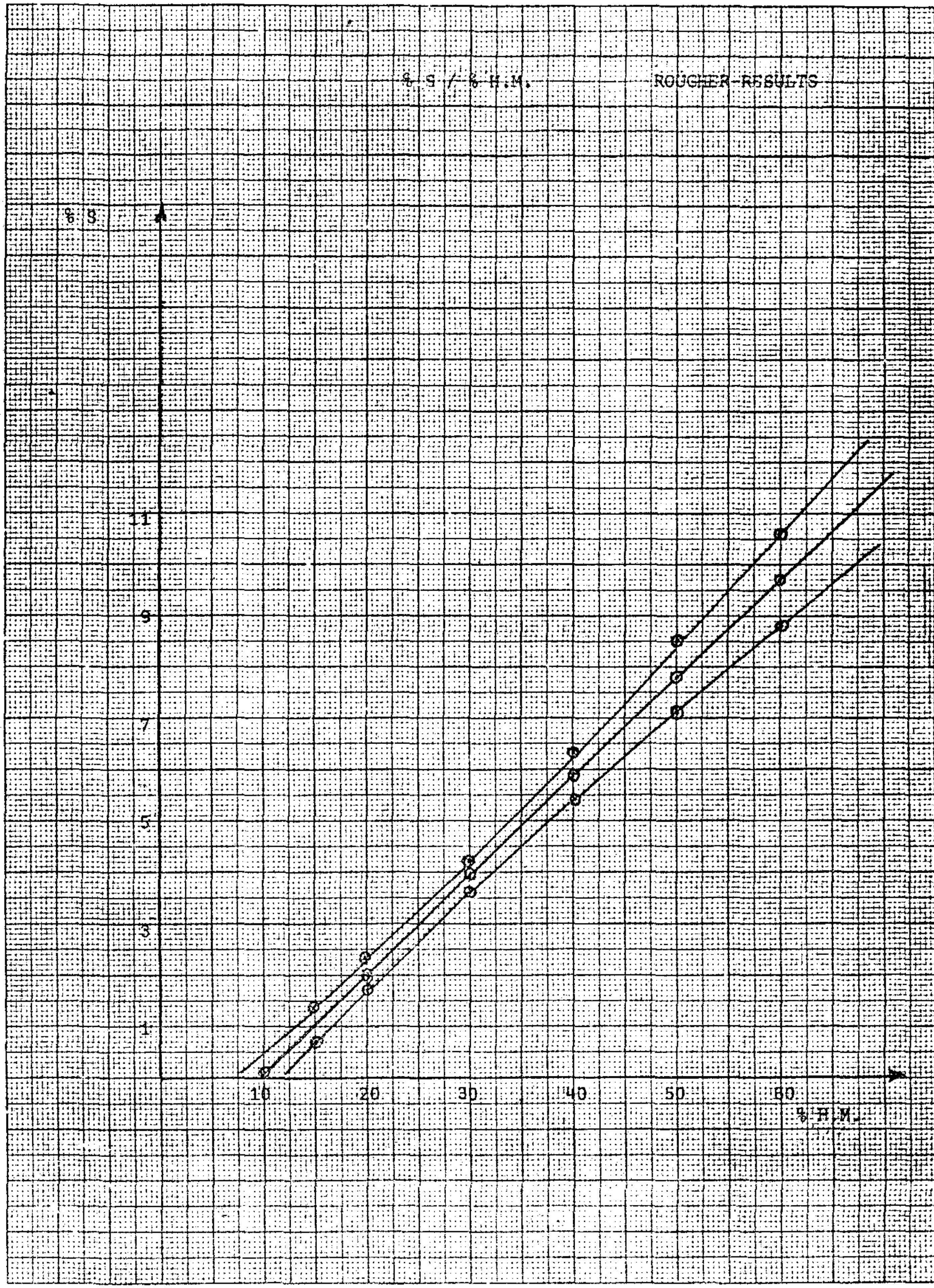


FIG. 4.5C

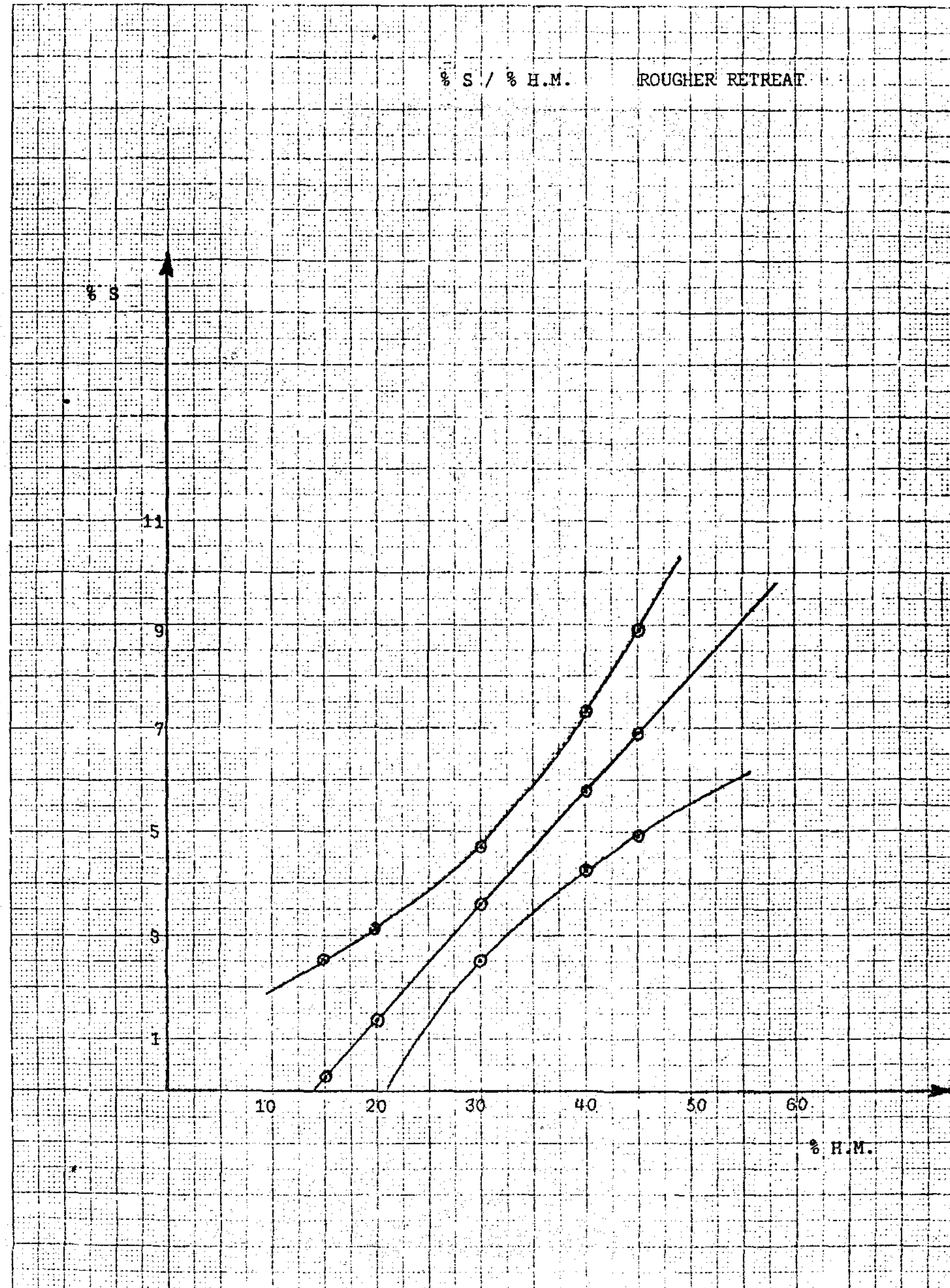
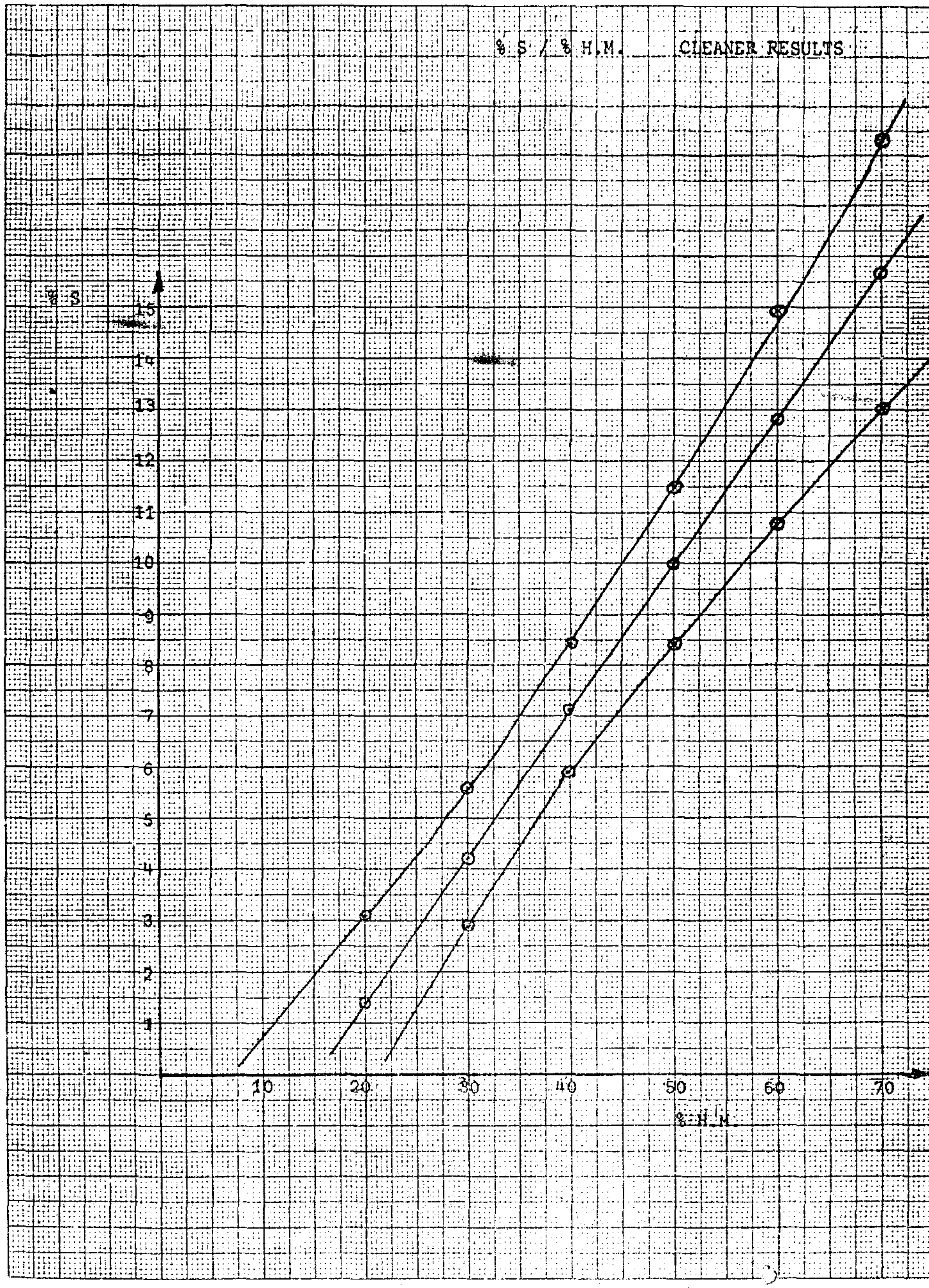


FIG. 4.5D


% S / % H.M. CLEANER RESULTS
% S
15
14
13
12
11
10
9
8
7
6
5
4
3
2
1
10
20
30
40
50
60
70
% H.M.

RAG:MN

18th July, 1975

Mr. D. N. Buchholz,
Manager,
Australia-Cities Service Inc.,
5th Floor,
151-153 Macquarie Street,
SYDNEY, N.S.W. 2000.

Dear Don,

Further to our discussions with yourself, Doug Jinks and Ron Butler on the 15th of July, there follows a proposal for cone concentrator testwork on the alluvial sulphide ore.

It is our understanding that you wish to define a gravity concentration flowsheet for the concentration of an alluvial sulphide ore, in order to produce a bulk sulphide concentrate. The ore will contain of the order 5% sulphur in the form of pyrite and other metal sulphides. A concentrate containing 45-50% sulphur should be obtained by a combination of gravity and, if necessary, flotation techniques. A plant capacity of 30-40,000 tonnes per day is indicated at this stage.

The objective is to recover at least 65% of the sulphur values and associated metal values in a final concentrate or intermediate product. You will deliver a 30 tonne sample of the ore, packed in polythene lined 44 gallon drums. The sample will be, as drilled, with excess water decanted. "Very little (<5%) - 350 mesh B.S. slimes will be present and the sample will contain substantially no material coarser than 1/8".

The testing program will be in three stages:-

1. Sufficient sample (4-5 tonnes) will be deslimed in a 6" Krebs cyclone at 50 microns; the overflow will be dewatered and stored. The deslimed sands will be utilised in a single stage cone concentrator test rig for a series of 12 tests. The results of these tests will determine the ability of the cone concentrator to successfully concentrate this sample. It will also provide data to facilitate the second stage of concentration. The samples will be evaluated by means of heavy liquid analyses and iron and sulphur chemical analyses. The cost of this stage, including preliminary reporting is estimated at $3,500.00 with analysis charges.

2. If the first stage of testwork is successful a decision will then be made to proceed to the second stage. The second stage will consist of a program of full scale cone concentrator testwork on both rougher and cleaner stages and, if indicated, recleaner stages. This program will define the concentration plant flowsheet. A minor program of spiral testwork for concentrate upgrading will be a part of this second stage. The estimated cost of the second stage including analysis is $7,000.00. Analysis techniques will be similar to those of stage 1.

...../2

3. The third stage which is integral with the second is final reporting and analysis and we anticipate that this will cost $1,500.00.

We are timing your program for the October/November period and anticipate the following time scale:-

Stage 1 - 10 working days.
Stage 2 - 20 working days.
Stage 3 - 10 working days.

We hope therefore if we can commence work in the first week of October we would be able to complete our reports by early December. You will realise of course that the costs are only estimates, but we have allowed for anticipated cost increases in the next few months.

The analytical work to be performed is aimed at supporting the cone concentrator test program. If you require more detailed analysis of some of the products this might incur additional costs.

We will of course make available to you samples of test products of whatever size you specify. We will also produce and pack a bulk sulphide concentrate for shipment for further processing. Transportation costs of these samples would however be additional to the above program.

You are welcome to have your representative on site during periods of the test program. As the program will stretch over a period of two months, this should be arranged on a mutually convenient basis.

We trust that this proposal is sufficiently definitive and we look forward to your reply.

Yours faithfully,

R. A. GRAVES
GROUP METALLURGIST

cc. Dr. D. Jinks,
Senior Project Geologist - Minerals,
Cities Service International, Inc.,
521 South Boston,
Box 37,
Tulsa, Oklahoma, 74102.
(+ copy to Sydney office).

cc. Mr. R. Butler,
Manager,
Robertson Research (Australia) Pty. Ltd., Moss Vale Road,
Bowral,
N.S.W. 2576.

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CITIES SERVICE INTERNATIONAL, INC.
521 SOUTH BOSTON
TULSA, OKLAHOMA

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TELEX: 49 7490
CABLE: CITEXTUL

November 12, 1975

REPLY TO:
P. O. BOX 37
TULSA, OKLAHOMA 74102
U.S.A.

RECEIVED
19 NOV 1975
Ans'd..............................

Dr. A. B. Holland-Batt
Senior Research Metallurgist
Mineral Deposits Limited
81 Ashmore Road
Southport, Queensland
Australia, 4215

Dear Dr. Holland-Batt:

This letter serves as confirmation of Cities Service's intention to proceed with the cone concentrator tests discussed with Mr. R. Graves on July 15, 1975 in Southport. It also serves as an acceptance of your work proposal dated July 18 and addressed to Mr. D. H. Buchholz, Regional Manager, Australia-Cities Service Inc.

I wish to apologize for the delay in acknowledging our intent in writing. As either Mr. Buchholz or myself have been in telephone contact with you, I trust that this delay has not caused any serious problems.

To confirm our recent telephone conversation, I will plan on being in Southport around the end of the first week in December to observe the full scale cone tests scheduled to be in progress at that time. If these tests are delayed, please let me know so I can adjust my schedule.

I am looking forward to seeing you and to a successful program.

Best regards.

Sincerely,

Douglas D. Jinks,
Senior Projects Geologist

DDJ:jh

cc: Mr. D. H. Buchholz
Mr. R. A. Graves

AUSTRALIA CITIES SERVICE

REPORT 15.213.1/2

SPIRAL CONCENTRATOR STUDIES AND CIRCUIT DESIGNS ON ALLUVIAL SULPHIDE ORE

M.S. CROSS B.Met.E., A.M.A.I.M.M.

14.5.76

MINERAL DEPOSITS LIMITED
RESEARCH AND ENGINEERING DEPARTMENT
SOUTHPORT QUEENSLAND
AUSTRALIA

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1. INTRODUCTION

1.1 MATERIAL

The material used for the spiral concentrator testwork was cleaner concentrates from the cone concentrator testwork as outlined in section 3.3.4 part (c) of Mineral Deposits Report No.15.213.1/1 titled "Cone Concentrator Studies and Circuit Designs on alluvial sulphide ore".

1.2 OBJECTIVES

The purpose of the testwork was to find out if the cleaner concentrates or final concentrates produced from the cone plant could be upgraded in sulphur content above 40% and maintain a reasonable recovery of sulphur in a cone-spiral wet gravity concentrator.

1.3 TERMS OF REFERENCE

The terms of reference are based on discussions held with R. Butler of Robertson-Research and M.D.L. personnel plus telex confirmation from Dr. D. Jinks of Australia Cities Service dated March 11th 1976.

1.4 SUMMARY

Sulphur — From the series of six tests conducted on the spirals, a final sulphide concentrate assaying 43.9% S was obtained.

In a spiral-cone concentrator plant a final sulphide product assaying 40% sulphur, recovery of 80% sulphur and a weight recovery between 6-8% could be achieved. Table 1.4 shows a summary of the cone plant studies and cone-spiral plant studies.

Copper — As pointed out earlier, the copper minerals are hard to identify and equally hard to recover. Final spiral concentrate assays 0.33% Cu. This would give a copper recovery of 12% into the final concentrate. From these results the copper is not associated with the sulphur minerals to a very large extent.

Process Design An integrated cone-spiral plant flowsheet has been designed. The size and circuit is shown in Fig. 4.1. The cost for such a plant excluding a dredge would be somewhere in the range of 3-4 million dollars Australian.

M. S. Cross

M.S.CROSS, B.Met.E., A.M.A.I.M.M.
SENIOR PROJECT METALLURGIST

14th May, 1976
MSC:MM

2. TEST EQUIPMENT

2.1 TEST RIGS

For the spiral concentrator testwork a two stage rig was used. The following table outlines the test rigs that were used for the testwork:-

TEST NO.	FIG.NO.	TOP SPIRAL	BOTTOM SPIRAL
1	2.1	Mk.IIA	Mk.IIA
2	2.2	Mk.IIA	Mk.IIB
3	2.2	Mk.IIA	Mk.IIB
4	2.2	Mk.IIA	Mk.IIB
5	2.2	Mk.IIA	Mk.IIB
6	2.2	Mk.IIA	Mk.IIB

2.2 ANALYTICAL SYSTEMS

2.2.1 Sampling
The spiral test rig has semi-automatic sampling systems incorporated with the rig for the tabling of timed samples.

2.2.2 Spiral products were assayed for sulphur only by X.R.F. analysis.

3. TESTWORK

3.1 PROCEDURE

Two preliminary tests were carried out to explore the best type of configuration. These are tests 1 and 2.

On examination of these products, an additional three tests were run to find peak sulphur grade. These are tests 3, 4, 5.

On completion of these three tests a final optimising test was run to verify the results.

After testwork was completed, a final spiral concentrate was produced using all the cleaner concentrate feed. This is for Australia Cities Service.

3.2 RESULTS

The results for the six spiral tests are found in Table 3.1A & B. Between tests 2 and 3 the feed in the test rig was changed due to some possible contamination.

This explains part of the head feed upgrade; the other contributing factor would be the method of taking the feed from the cleaner concentrate. A representative sample was taken from the complete cleaner concentrate product.

3.3 DISCUSSION

The results are almost self-explanatory. Test 2 indicates that a 40% plus sulphur grade product can be produced at a reasonable recovery.

As attempts to increase this grade are made, recovery of sulphur falls away without significant increases in sulphur content of concentrates. This is shown in tests 3-5.

The final test confirms the result of test 2; an excellent final product is possible with a reasonable spiral recovery.

4. PROCESS DESIGN

4.1 CONE-SPIRAL CIRCUIT

An integrated cone-spiral concentrator circuit is shown in Fig. 4.1. This circuit should ensure an overall recovery of sulphur at 80% and a final concentrate grade in excess of 40% sulphur. The weight recovery into final concentrates would be 6-8%.

The copper grade of the final product produced from the cone-spiral circuit was 0.33% Cu with a rather poor recovery of 12%. Table 1.4 summarises the results for both sulphur and copper.

TABLE 1.4

	WT. %	SULPHUR %	RECOVERY	COPPER %	RECOVERY
Original Feed	100	3	100	0.16	100
Final Product Cone Plant	16	16	85	0.26	26
Final Product Cone-Spiral Plant	6-8	40	80	0.33	12

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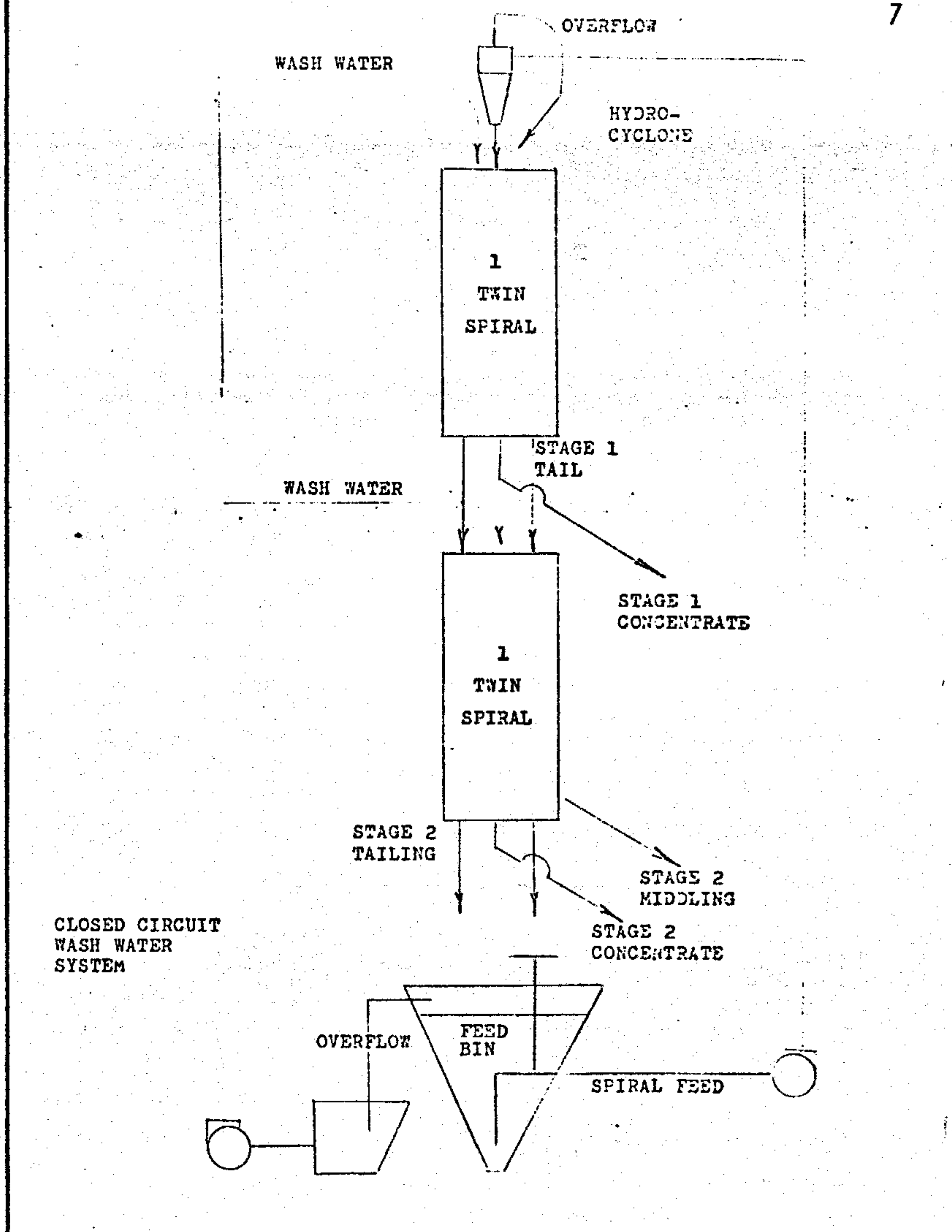
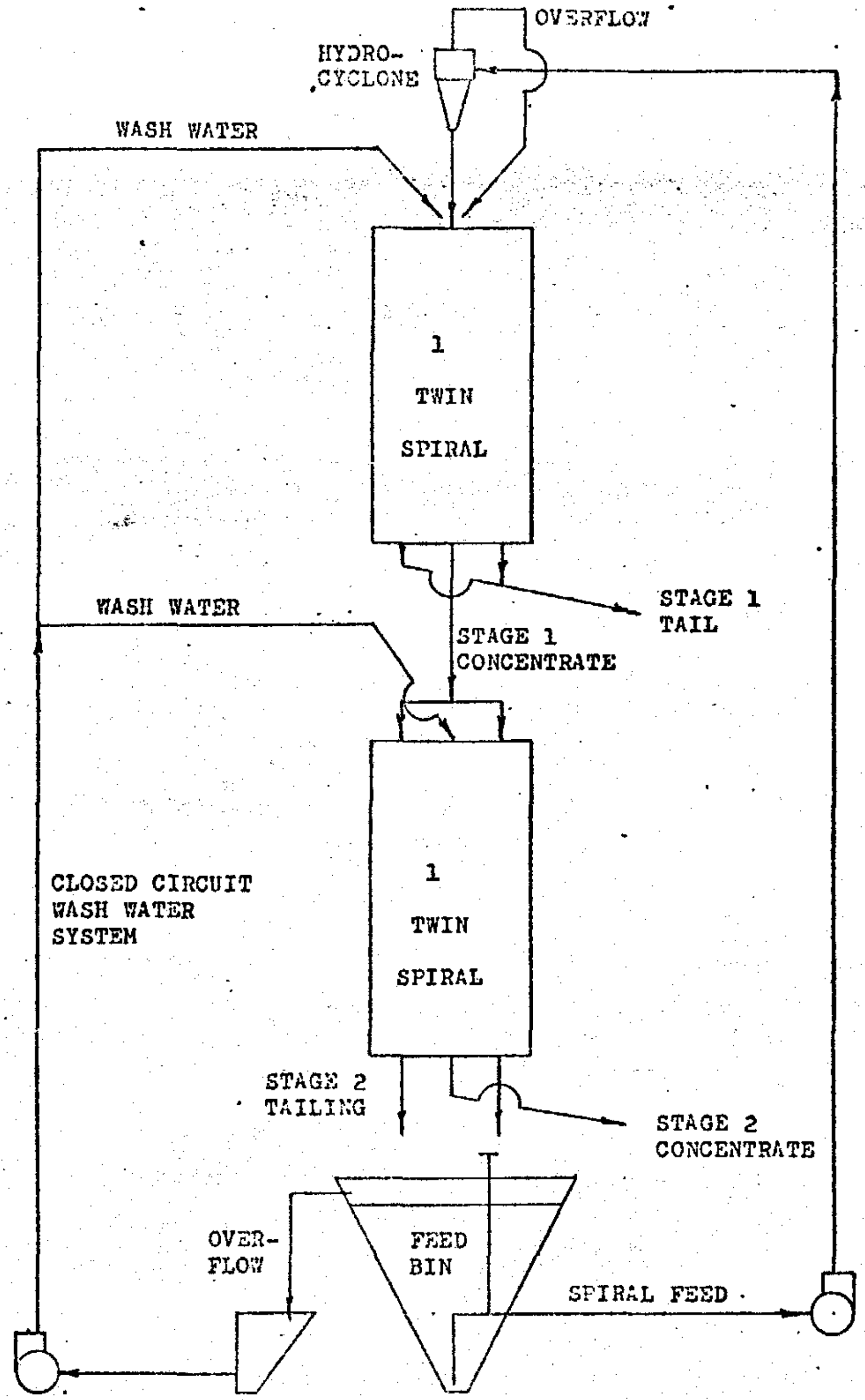
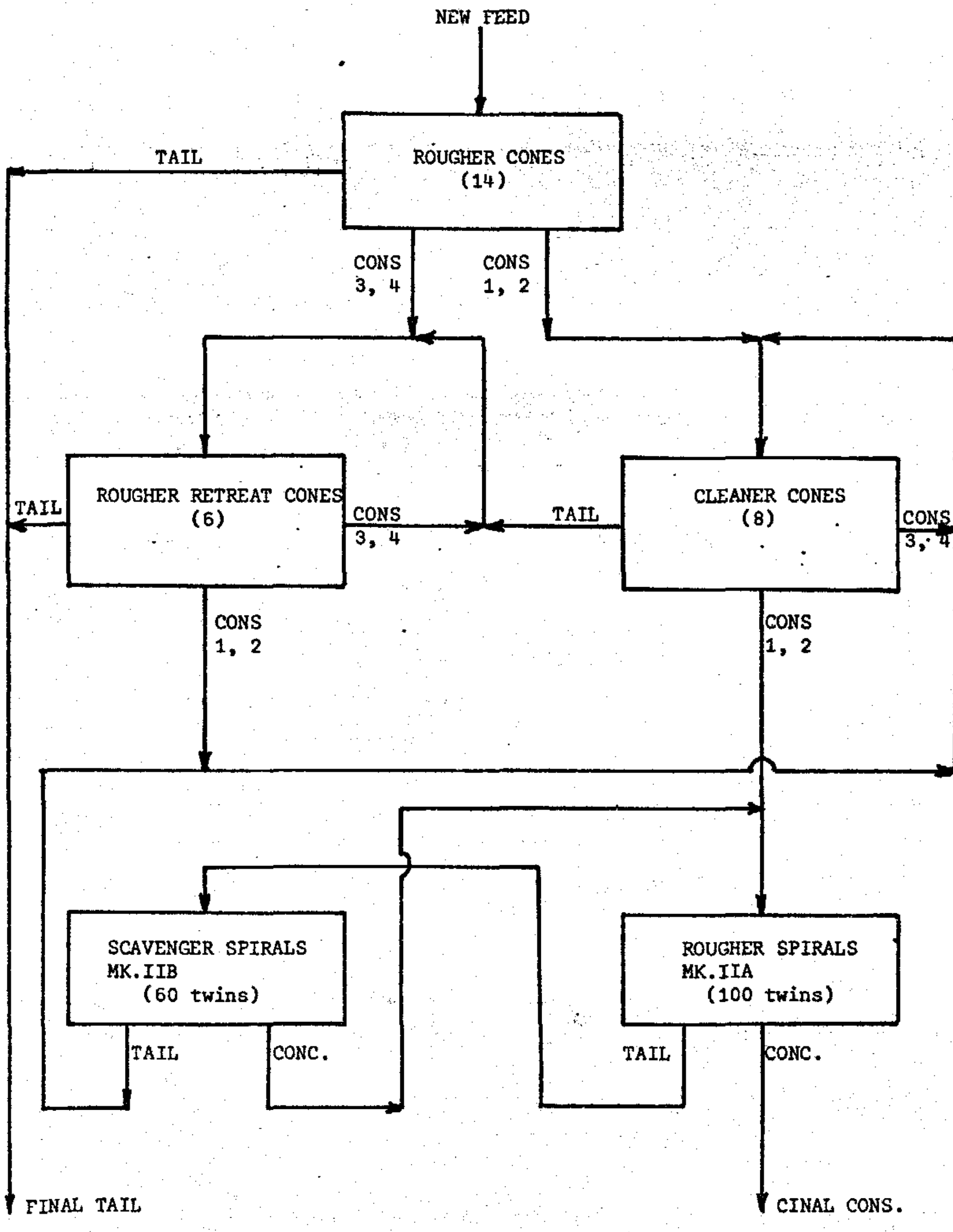


REICHERT MINING EQUIPMENT DIVISION MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND	FIG.2.2
SCHEMATIC ARRANGEMENT OF SPIRAL TEST RIG SYSTEM INDICATING CONDITIONS FOR OPERATING TWO TWIN SPIRALS IN A SERIES CONFIGURATION.	DRAWN I.J.T. DATE 1.12.71 SCALE N.T.S.



REICHERT MINING EQUIPMENT DIVISION MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND	FIG.2.1
SCHEMATIC ARRANGEMENT OF SPIRAL TEST RIG SYSTEM INDICATING CONDITIONS FOR OPERATING TWO TWIN SPIRALS IN A SERIES CONFIGURATION.	DRAWN: A.K.B. DATE: 31.5.72 SCALE: N.T.S.



RESEARCH AND ENGINEERING DEPARTMENT **MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND**	FIG. 4.1
AUSTRALIAN CITIES SERVICE CONE-SPIRAL FLOWSHEET	DRAWN: M.S.C. DATE: 9.2.76 SCALE: N.T.S.

MINERAL DEPOSITS LIMITED, SOUTHPORT, QUEENSLAND, AUSTRALIA

TABLE 3.1A Form LRS 1 Issue 3

RESEARCH AND DEVELOPMENT LABORATORIES - TEST RESULTS AND SUMMARY SHEET

PROJECT........................TESTWORK............................UNIT............................DATE.........

TEST NO.	HEAD FEED			ROUGHER TAILINGS				CLEANER TAILINGS				CLEANER CONCENTRATE				RECOVERY	UPGRADE RATIO
				SOLIDS		MINERAL		SOLIDS		MINERAL		SOLIDS		MINERAL			
	TPH	GRADE %.S..	DENSITY %	TPH	%DIST	GRADE %.S..	%DIST	TPH	%DIST	GRADE %.S...	%DIST	TPH	%DIST	GRADE %.S...	%DIST		
1	1.71	14.5	-	1.19	69.6	6.2	29.8	0.15	8.8	27.5	16.5	0.37	21.6	36.0	53.7		
				SCAVENGER TAILING				SCAVENGER CONCENTRATE				ROUGHER CONCENTRATE					
2	1.47	17.3	-	0.69	46.9	3.2	8.6	.41	27.9	18.8	30.2	0.37	25.2	42.1	61.2		
3	1.94	20.6	-	0.95	49.0	5.8	13.8	0.56	28.4	28.3	39.4	0.43	22.1	43.6	46.8		
4	2.18	21.1	-	1.12	51.4	6.2	15.2	0.63	28.9	32.3	43.5	0.43	19.7	44.3	41.3		
5	2.08	20.7	-	1.08	51.9	5.4	14.0	0.63	30.3	32.4	46.5	0.37	17.8	45.2	39.5		
6	1.97	22.3	-	0.82	41.6	3.9	6.8	0.50	25.4	23.9	27.3	0.65	33.0	43.9	65.9		

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TABLE 3.1D

Issue 3

RESEARCH AND DEVELOPMENT LABORATORIES - TEST RESULTS AND SUMMARY SHEET

PROJECT.....A.C.S.............TESTWORK.....Spirals..............UNIT..........................DATE..........

TEST NO.	HEAD FEED			ROUGHER TAILINGS				CLEANER TAILINGS				CLEANER CONCENTRATE				RECOVERY	UPGRADE RATIO
				SOLIDS		MINERAL		SOLIDS		MINERAL		SOLIDS		MINERAL			
	TPH	GRADE % H.M.	DENSITY %	TPH	%DIST	GRADE % H.M.	%DIST	TPH	%DIST	GRADE % H.M.	%DIST	TPH	%DIST	GRADE % H.M.	%DIST		
1	1.71	-	-	1.19	69.6	-	-	0.15	8.8	-	-	0.37	21.6	-	-		
				SCAVENGER TAILING				SCAVENGER CONCENTRATE				ROUGHER CONCENTRATE					
2	1.47	62.6	-	0.69	46.9	31.98	29.0	0.41	27.9	81.31	36.2	0.37	25.2	99.1	39.8		
3	1.94	65.9	-	0.95	49.0	37.75	28.05	0.56	28.4	88.31	38.67	0.43	22.1	99.0	33.28		
4	2.18	68.3	-	1.12	51.4	43.37	32.6	0.63	28.9	91.95	38.9	0.43	19.7	98.61	28.5		
5	2.08	68.1	-	1.08	51.9	43.37	33.1	0.63	30.3	92.2	41.0	0.37	17.8	99.14	25.9		
6	1.97	70.6	-	0.82	41.6	38.97	23.0	0.50	25.4	87.38	31.4	0.65	33.0	97.64	45.6		

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APPENDIX VII

THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES (AMDEL)

"Lamflo Sluice and Spiral Beneficiation Tests"

261126

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 3/798/0

29 March 1976

Mr D.H. Buchholz,
Vice President,
Australia Cities Service Inc.,
5th Floor,
151-153 Macquarie Street,
SYDNEY. NSW 2000

REPORT: ME2072/76

YOUR REFERENCE:	Telex messages dated 8/1/76, 2/2/76
MATERIAL:	Macquarie Harbour Sediment
LOCALITY:	-
IDENTIFICATION	Batch No.1 Composite Sample
DATE RECEIVED:	6/1/76
WORK REQUIRED:	Lamflo sluice and spiral beneficiation tests

Investigation and Report by:	W.G. Rogers
Operations Manager:	R.E. Wilmshurst

R.E. Wilmshurst

for F.R. Hartley
Director.

cc Dr D.D. Jinks
Senior Project Geologist Minerals,
Cities Service International Inc.
521 South Boston, Box 37,
Tulsa, OKLAHOMA 74102

Mr R. Butler,
Director and General Manager,
Robertson Research (Australia) Pty. Ltd,
ADC House, 77 Pacific Highway,
NORTH SYDNEY. NSW 2060

Pilot Plant: Osmond Place, Thebarton, South Australia, phone Adelaide 43 8053 Branch Offices: Perth and Sydney

1. INTRODUCTION

Australia Cities Service Inc., are currently carrying out a conceptional study on the feasibility of various beneficiation processes for the treatment of the coarse delta sand deposit from the Macquarie Harbour sediment.

The material in this deposit originated as a result of the deposition of the solids from the tailing pulp discharged into the Macquarie river from the Mount Lyell copper treatment plant. It contains heavy mineral sulphides, mainly pyrite (total sulphur content as much as 3.7%), together with a small amount of the unclaimed copper minerals (at least 0.15% Cu).

The objective is to produce a sulphide concentrate suitable for the manufacture of sulphuric acid and to recover portion of the copper by carrying out an acid leaching stage on the roasted sulphide product.

Results of test work carried out so far indicate treatment probably will require primary gravity concentration stages aimed at rejecting as much of the gangue material as is possible with minimal loss of the sulphide minerals and a secondary flotation stage to assist grade improvement.

Amdel's involvement in this project arose from the availability within Amdel of a Lamflo sluice together with spiral gravity separators. The sponsor was interested in investigating these separators as an alternative primary gravity beneficiation stage for the beneficiation of a composite sample of their coarse delta sand sediment and to compare the separation results obtained, in this case, with those achieved as a result of independent investigation carried out with the Reichert cone.

This report outlines the results of this investigation.

2. MATERIAL EXAMINED

This testwork was carried out with 'Batch No.1' composite sample from the Macquarie Harbour coarse delta sand sediment received 6/1/76 and contained in twenty five 200 litre polythene lined, sealed drums. The material, essentially of minus 710 m with negligible fines, had a net dry weight of 5.5 tonnes. Details of the results of size and assay analyses carried out on a head sample spear sampled from the 25 drums, are included in Tables 1 and 2. The specific gravity of the dry head sample was 2.85.

3. ANCILLARY MATERIAL

The following ragent was used for the purpose of preliminary, assessment of the test results:

Name	Composition	Use
T.B.E. (Heavy liquid)	Tetra bromo ethane	Heavy liquid separation of the heavy minerals in the dry test products at: Sp. gr. 2.95 (Tests 3 to 5) Sp. gr. 2.90 (Tests 7 to 9)

4. EQUIPMENT USED

The following equipment was used in the tests:

- Mineral Separator Spirals (2)
- Denver 1.2 x 1.2 metre conditioning tank (1)
- 2.54 x 3.81 cm Warman 3.7 kW motor fixed speed pumps (2)
- Lamflo Separator (1)
- Automatic Sample Cutters (4)
- Standard Screens (AS 1152 - 1973)

5. EXPERIMENTAL PROCEDURE AND RESULTS

5.1 Sample Preparation

The contents of the twenty five drums were dewatered and the settled solids were spread onto a concrete apron and air dried to 5% moisture. Mixing was accomplished by coning five times with a front-end loader.

Preliminary testing of a 0.5 tonne parcel of the composite sample revealed the presence of plus 1.27 cm stone pebbles which caused blockages to arise in the feed lines. Consequently it was necessary to screen the balance of the composite sample over a 0.635 cm screen to remove the pebble fraction. The weight of pebbles screened from the balance of the bulk composite sample was negligible.

5.2 Preliminary Closed Circuit Lamflo Tests

Preliminary tests with the Lamflow separator were carried out in closed circuit by returning the concentrates (3) and tailing products back to the Denver feed mixing tank charged with approximately 0.5 tonne (dry weight) of the pulped composite sample.

Tests were carried out by varying the cutter slot widths and feed pulp flowrate while maintaining the pulp density and deck slope constant.

Best performance for the Lamflo separator was obtained when using a cutter slot width of 0.635 cm (¼") as this enabled concentrate weight recoveries of the order of 50 to 70% to be obtainable when operating close to the recommended feed rates of 5.6 to 7.8 dry tonnes/hour.

Three tests, Tests 3, 4 and 5 at dry feed rates of 5.49, 6.05 and 6.5 tonnes/hour respectively were carried out when operating at the above cutter slot widths and feed solids concentration of the order of 52% solids weight. The results of these tests were assessed by carrying out T.B.E. heavy liquid separations on the test products. The results are shown in Tables 3 to 5 and graphically illustrated in Figure 1.

5.3 Production of Low Grade Bulk Lamflo Concentrate From Open Circuit Runs (Test 6)

Production of a low grade bulk Lamflo concentrate for spiral cleaning tests was carried out by running approximately 2.0 tonnes of the pulped composite delta sand sediment through the Lamflo separator when run in open circuit. To treat this quantity of the composite sample four 0.5 tonne (approx.) parcels of the sample were run through the separator separately. The average feed rate for these runs was 5.7 dry tonnes per hour at an average pulp solids concentration of 52% solids weight.

The dry solids weight distributions for the bulk concentrate and tailing products were determined by tonnage measurements on the thickened bulk Lamflo concentrate and tailing pulps.

Initial assessment of the results was done by TBE heavy liquid separations on samples of the test products. Final assessment of the Lamflo bulk concentrate and tailing products for Cu, Zn, Co, Mo and Total sulphur. The results are shown in Tables 6 and 7.

5.4 Open Circuit Spiral Cleaning Tests

Spiral cleaning of the bulk Lamflo concentrate was investigated by operating the Denver feed mixing tank in open circuit with a two-stage spiral circuit using a normal rougher/cleaner configuration with the cleaner spiral tailing returning to the rougher unit.

Four Tests, Tests 7, 8, 9 and 10 were carried out by feeding separate batches of the bulk Lamflo concentrate pulp, agitated in the Denver mixing tank, at a pulp solids concentration of approximately 52% solids weight to the rougher spiral feed box where it was diluted to approximately 30% solids weight with fresh make up water. The dry solids feed rates during these tests were within the range of 0.54 to 0.66 tonnes/hour. Wash water was added independently to each spiral at a fixed rate of 3.5 litres/minute.

Variation in the cleaner spiral concentrate weight recovery was accomplished by varying the angle of the cutters at the spiral concentrate ports.

During each test samples were taken of the bulk Lamflo concentrate (rougher spiral feed), rougher spiral tailing and cleaner spiral concentrate. These samples were assayed for Cu, Co, and Total sulphur. The results are shown in Tables 9 to 12.

Preliminary results for the T.B.E. heavy liquid separations carried out on the bulk Lamflo concentrate (rougher spiral feed), cleaner spiral concentrate and rougher spiral tailing are shown in Table 8 and graphicall-illustrated in Figure 1.

6. DISCUSSION OF RESULTS AND CONCLUSIONS

Total sulphur determinations on the Lamflo and spiral test products, were by gravimetric procedures, which were found to be more accurate than the tube furnace method.

The weight recovery results for the spiral tests have been determined from the Total sulphur analyses on the test products as these are considered more reliable than those calculated from the other elements which occur in lower concentrations.

For the spiral tests difficulty was experienced in obtaining mass balances when using the assays for the trace elements Zn and Mo. Therefore these results have not been included. However the indications were that these elements tend to report in the higher weight fractions recovered.

The beneficiation results achieved for the Total sulphur, copper and cobalt when using the Lamflo separator as an initial rougher concentration stage and spiral separators in a cleaning stage are outlined in Figures 2, 3 and 4 respectively. The best results achieved (Test 7) show a concentrate grade of 14.6% T.S. and 0.23% copper for respective recoveries of 79.7 and 23.7% from the original composite sample treated. These results correspond to a weight rejection of 80.9% of the original composite material treated. It is obvious from the test results that effective recovery of the copper constituent will not be possible when employing this type of gravity treatment circuit.

TABLE 1: RESULTS OF SIZE AND ASSAY ANALYSIS

Product (Mesh BSS)	Weight %	Assay % Cu	Assay % Total Sulphur	Assay ppm Zn	Assay ppm Co	Assay ppm Mo	Distribution % Cu	Distribution % Total Sulphur	Distribution % Zn	Distribution % Co	Distribution % Mo
Head Sample	100.0	0.15	3.75	185	80	55	100	100	100	100	100
+60	17.0	0.19	0.70	325	25	45	20.9	3.2	25.1	5.2	14.2
-60+85	29.3	0.16	0.88	185	25	45	30.4	6.9	24.7	8.9	24.5
-85+120	22.7	0.14	1.96	200	45	50	20.6	11.9	20.6	12.4	21.0
-120	31.0	0.14	9.40	210	195	70	28.1	78.0	29.6	73.5	40.3
(Calc.) Head	100.0	0.154	3.74	220	82	54	100.0	100.0	100.0	100.0	100.0

TABLE 2: DETAILED SIZING RESULTS

Product µm	Product Mesh BSS	Weight % Retained	Weight % Cumulative Passing
710	22	0.4	99.6
500	30	0.4	99.2
355	44	3.3	95.9
250	60	12.9	83.0
180	85	29.3	53.7
125	120	22.7	31.0
90	170	16.5	14.5
63	240	6.4	8.1
45	350	3.6	4.5
	-350	4.5	
		100.0	

TABLE 3: TBE ASSESSMENT OF RESULTS FOR CLOSED CIRCUIT LAMFLO TEST NO.3
Feed Rate: 5.49 dry tonnes/h (4.9 tons/h)
Feed Density: 53.7% solids weight
Deck Slope: 14°
Slot Width: 0.635 cm
Sp. Gr. TBE: 2.95

Product	Dry Solids Sp. Gr.	Weight %		Wt % Heavy Min		% Heavy Min. Distribution	
		Fract.	Cum.	Fract.	Cum.	Fract.	Cum.
Conc. 1	2.98	27.6	27.6	24.6	24.6	46.7	46.7
2	2.84	29.8	57.4	14.6	19.4	29.9	76.6
3	2.79	14.5	71.9	9.8	17.5	9.9	86.5
Tail	2.77	28.1	100.0	7.0	14.5	13.5	100.0
Feed (calc)	2.85	100.0		14.5		100.0	

TABLE 4: TBE ASSESSMENT OF RESULTS FOR CLOSED CIRCUIT LAMFLO TEST NO.4
Feed Rate: 6.05 dry tonnes/h (5.4 tons/h)
Feed Density: 52.8% solids weight
Deck Slope: 14°
Slot Width: 0.635 cm
Sp. Gr. TBE: 2.95

Product	Dry Solids Sp. Gr.	Weight %		Wt % Heavy Min		% Heavy Min. Distribution	
		Fract.	Cum.	Fract.	Cum.	Fract.	Cum.
Conc 1	2.97	23.3	23.3	25.1	25.1	43.3	43.3
2	2.83	27.3	50.6	15.2	19.8	30.7	74.0
3	2.80	12.7	63.3	10.8	18.0	10.2	84.2
Tail	2.78	36.7	100.0	5.8	13.5	15.8	100.0
Feed (calc)	2.84	100.0		13.5		100.0	

TABLE 5: TBE ASSESSMENT OF RESULTS FOR CLOSED CIRCUIT
LAMFLO TEST NO,5
Feed Rate: 6,5 dry tonnes/h (5.8 tons/h)
Feed Solids Concentration: 52.4% solids weight
Deck Slope: 14°
Slot Width: 0.635 cm
Sp. Gr. TBE: 2,95

Product	Dry Solid Sp. Gr.	Weight %		Wt % Heavy Min		% Heavy Min. Distribution	
		Fract,	Cum,	Fract,	Cum,	Fract,	Cum,
Conc 1	3.02	20.3	20.3	28.4	28,4	40.2	40.2
2	2.87	26.5	46.8	16,1	21,4	29.8	70.0
3	2.83	10.8	57.6	11,5	19.6	8,7	78,7
Tail	2.78	42.4	100.0	7.2	14.3	21,3	100.0
Feed (calc)	2,86	100,0		14.3		100,0	

TABLE 6: TBE ASSESSMENT OF RESULTS FOR OPEN CIRCUIT
LAMFLO TEST NO,6
Feed Rate: 5.7 dry tonnes/h (5,1 tons/h)
Feed Solids Concentration: 52% solids weight
Deck Slope: 14°
Slot Width: 0,635 cm
Sp. Gr. TBE: 2.90

Product	Dry Solids Sp. Gr.	Weight %*	Wt % Heavy Minerals	% Heavy Minerals Distribution
Bulk Conc	2.90	69,2	21.4	87,9
Tail	2,76	30,8	6,6	12,1
Feed (calc)	2.86	100,0	16.8	100,0

* Calculated by carrying out tonnages tests on Lamflo products,

TABLE 7: ASSAY RESULTS FOR OPEN CIRCUIT LAMFLO TEST NO.6

Product	Weight %	Assay					Distribution %				
		%		ppm			Cu	Total Sulphur	Zn	Co	Mo
		Cu	Total Sulphur	Zn	Co	Mo					
Bulk Conc	69.2	0.18	4.55	250	110	50	74.2	92.0	70.1	86.1	73.7
Tail.	30.8	0.14	0.89	240	40	40	25.8	8.0	29.9	13.9	26.3
Feed (calc)	100.0	0.17	3.42	247	88	47	100.0	100.0	100.0	100.0	100.0

TABLE 8: TBE ASSESSMENT OF RESULTS FOR SPIRAL TESTS (TESTS 7 to 9)
Rougher Spiral Feed Concentration: 30% Solids weight
Av Dry Feed Rate: 0.58 tonnes/h
Wash Water Feed Rate: 3.5 litres/min each spiral
No Spiral Stages: 1 Rougher
1 Cleaner, cleaner tail recycled to rougher spiral feed box
Sp. Gr. TBE: 2.90

Test No.	Product	% Weight		% Wt Heavy Mineral	% Recovery Heavy Mineral	
		Feed	Orig. Ore		Feed	Orig. Ore
7	B. Lamflo Conc.	100.0	69.2	19.0	100.0	87.9
	Cl. Sp. Conc	31.0	21.4	52.2	85.1	74.8
	Ro. Sp. Tail	69.0	47.8	4.1	14.9	13.1
8	B. Lamflo Conc.	100.0	69.2	20.0	100.0	87.9
	Cl. Sp. Conc	80.4	55.6	24.4	98.0	86.1
	Ro. Sp. Tail	19.6	13.6	2.0	2.0	1.8
9	B. Lamflo Conc.	100.0	69.2	18.8	100.0	87.9
	Cl. Sp. Conc	46.6	32.2	36.2	89.9	79.0
	Ro. Sp. Tail	53.4	37.0	3.6	10.1	8.9
10	B. Lamflo Conc.	100.0	69.2	18.4	100.0	87.9
	Cl. Sp. Conc	45.1	31.2	37.6	92.2	81.0
	Ro. Sp. Tail	54.9	38.0	2.6	7.8	6.9

TABLE 9: ASSAY RESULTS FOR OPEN CIRCUIT SPIRAL CLEANING TEST NO,7

Product	Weight % (Feed)	Assay % T,S,	Assay % Cu	Assay ppm Co	Distribution % (Feed) T,S,	Distribution % (Feed) Cu	Distribution % (Feed) Co
Spiral Cl. Conc	27.6	14.6	0.23	290	86.6	31.9	76.3
Ro. Spiral Tail	72.4	0.86	0.14	30	13.4	68.1	23.7
Spiral Feed	100.0	4.65	0.16	95	100.0	100.0	100.0

TABLE 10: ASSAY RESULTS FOR OPEN CIRCUIT SPIRAL CLEANING TEST NO.8

Product	Weight % (Feed)	Assay % T.S.	Assay % Cu	Assay ppm Co	Distribution % (Feed) T.S.	Distribution % (Feed) Cu	Distribution % (Feed) Co
Spiral Cl. Conc	86.1	5.10	0.18	115	96.5	52.9	96.6
Ro. Spiral Tail	13.9	1.15	0.16	30	3.5	47.1	3.4
Spiral Feed	100.0	4.55	0.17	105	100.0	100.0	100.0

TABLE 11: ASSAY RESULTS FOR OPEN CIRCUIT SPIRAL CLEANING TEST NO.9

Product	Weight % (Feed)	Assay % T.S.	Assay % Cu	Assay ppm Co	Distribution % (Feed) T.S.	Distribution % (Feed) Cu	Distribution % (Feed) Co
Spiral Cl. Conc	46.5	8.75	0.21	180	90.4	61.8	88.5
Ro. Spiral Tail	53.5	0.81	0.13	25	9.6	38.2	11.5
Spiral Feed	100.0	4.50	0.17	105	100.0	100.0	100.0

TABLE 12: ASSAY RESULTS FOR OPEN CIRCUIT SPIRAL CLEANING TEST No.10

Product	Weight % (Feed)	Assay			Distribution % (Feed)		
		%		ppm Co	T.S.	Cu	Co
		T.S.	Cu				
Spiral Cl. Conc	40.5	9.67	0.19	200	89.1	47.5	84.2
Ro. Spiral Tail	59.5	0.81	0.14	25	10.9	52.5	15.8
Spiral Feed	100.0	4.4	0.16	95	100.0	100.0	100.0

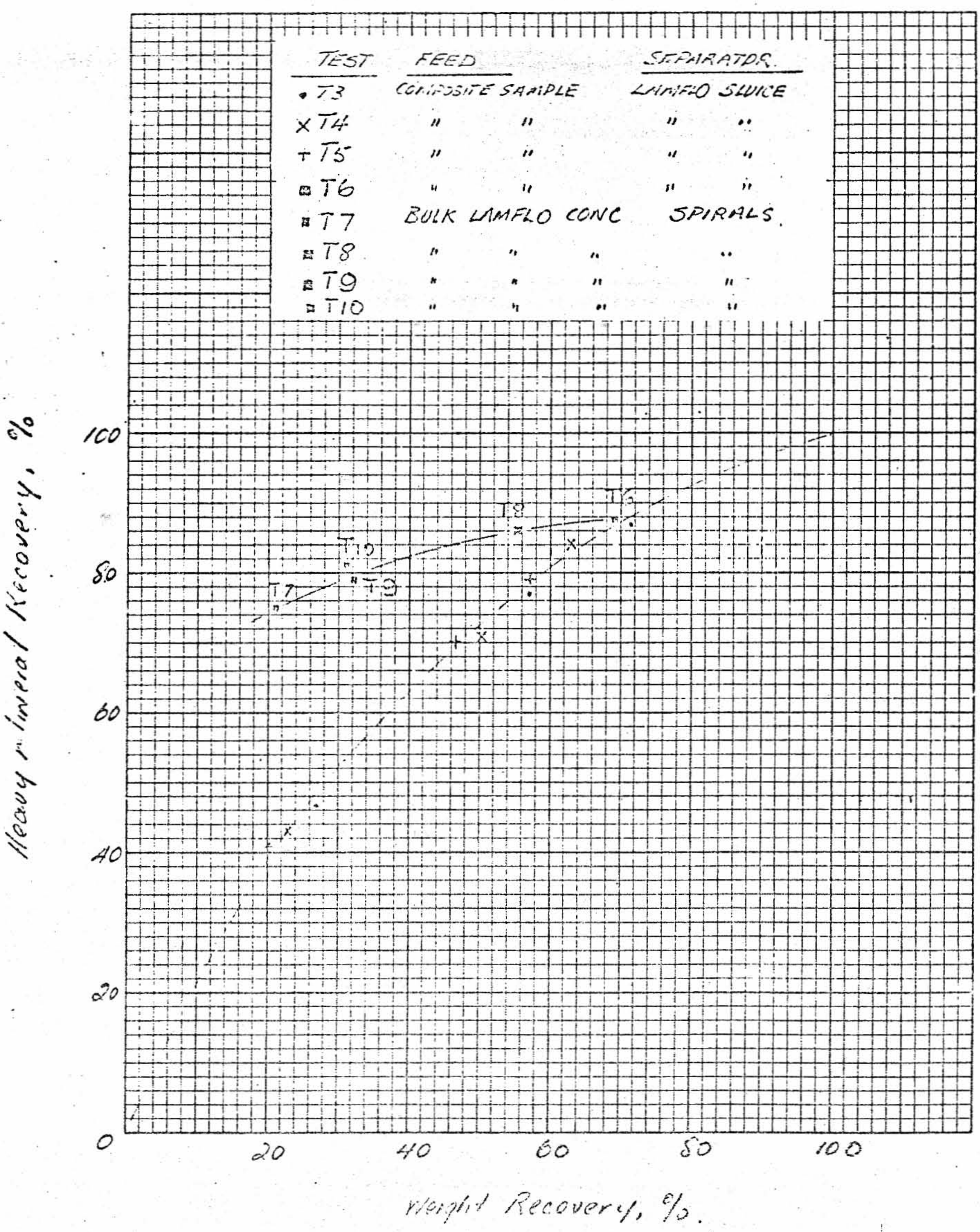

FIG 1. ASSESSMENT OF LAMFLO AND SPIRAL TEST PRODUCTS BY HEAVY LIQUID SEPARATION

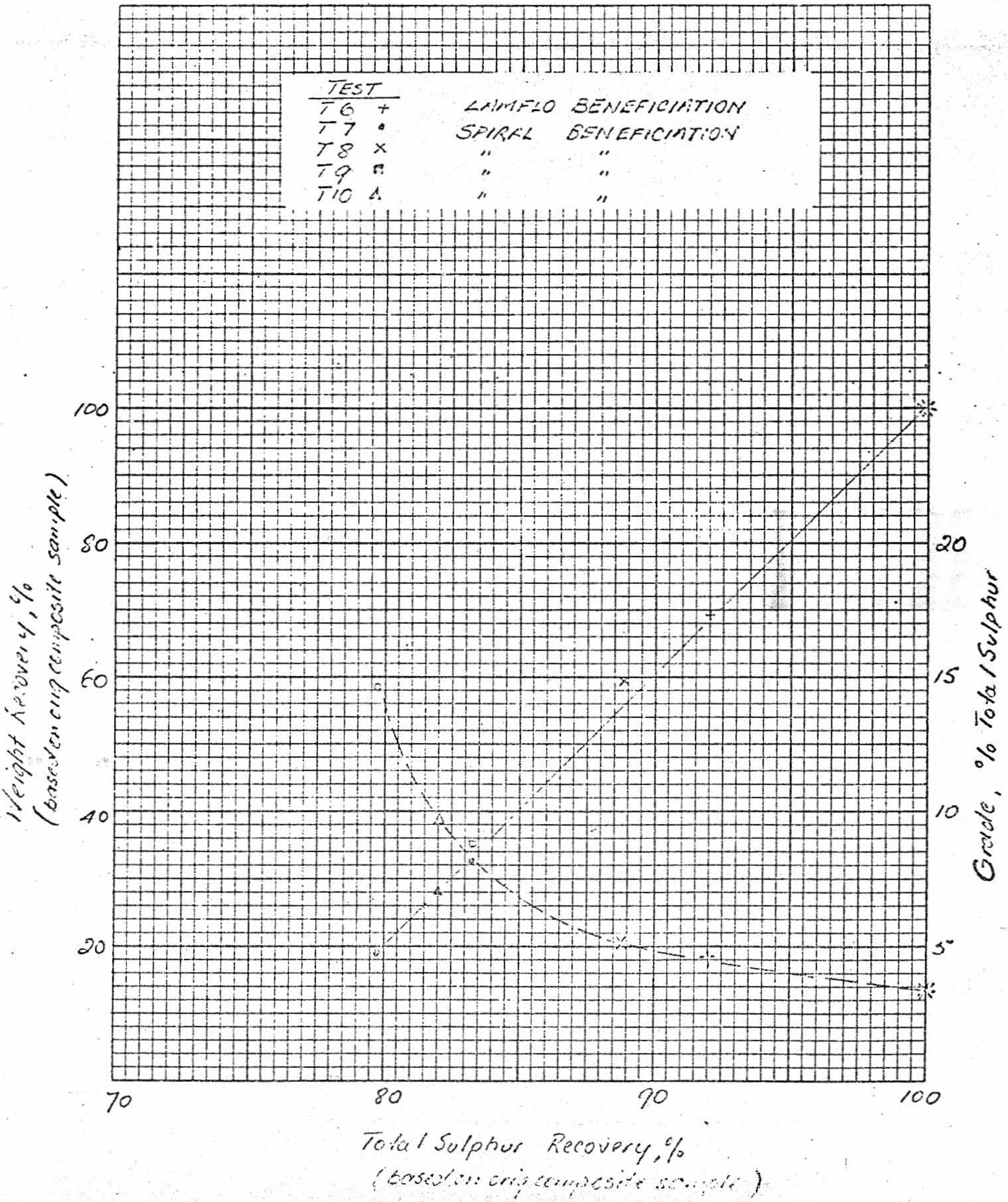

FIG 2. TOTAL SULPHUR BENEFICIATION OBTAINABLE AS A RESULT OF LAMFLO AND SPIRAL TREATMENT OF COMPOSITE SAMPLE

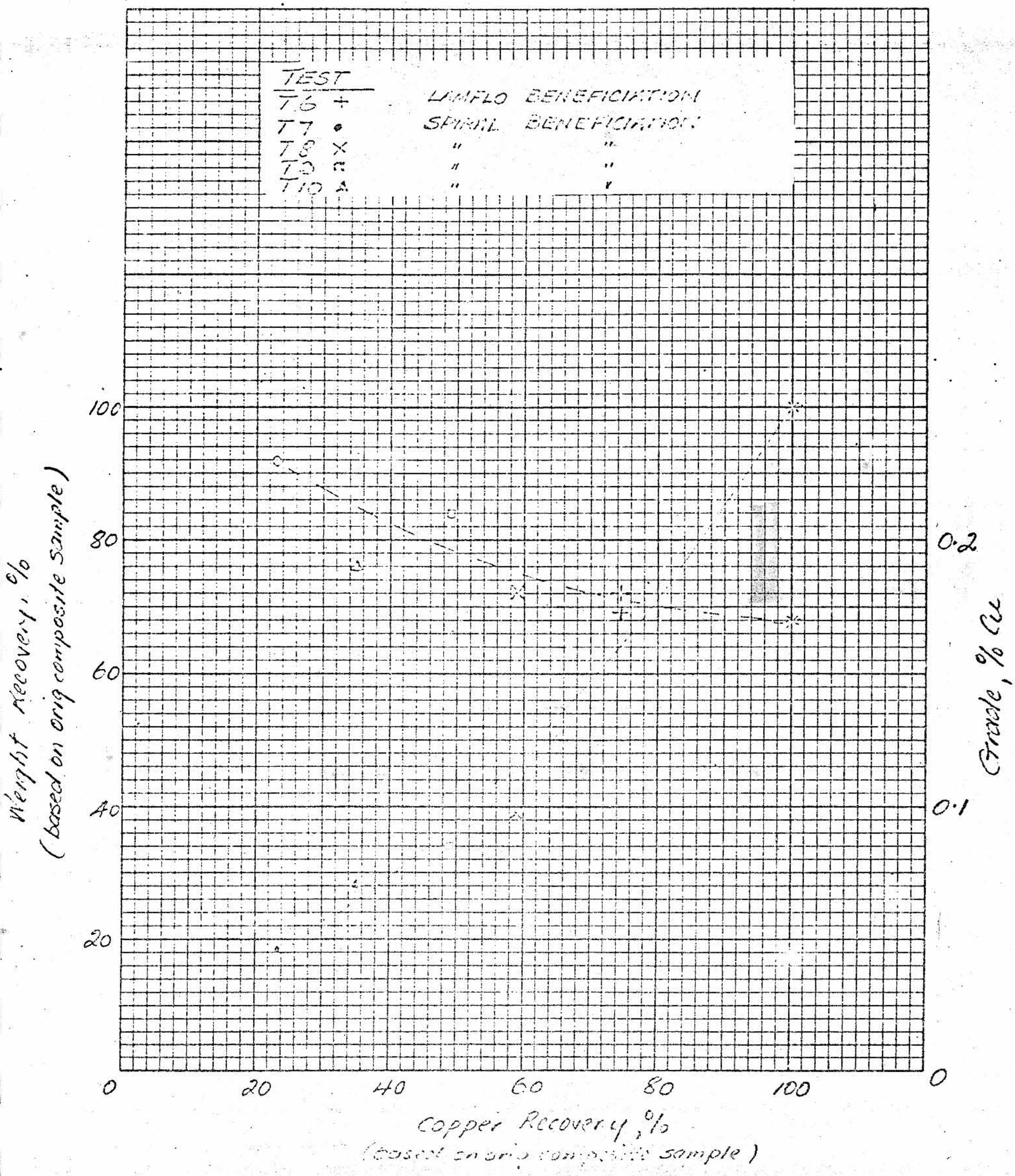

FIG 3. COPPER BENEFICIATION OBTAINABLE AS A RESULT OF LAMFLO AND SPIRAL TREATMENT OF COMPOSITE SAMPLE

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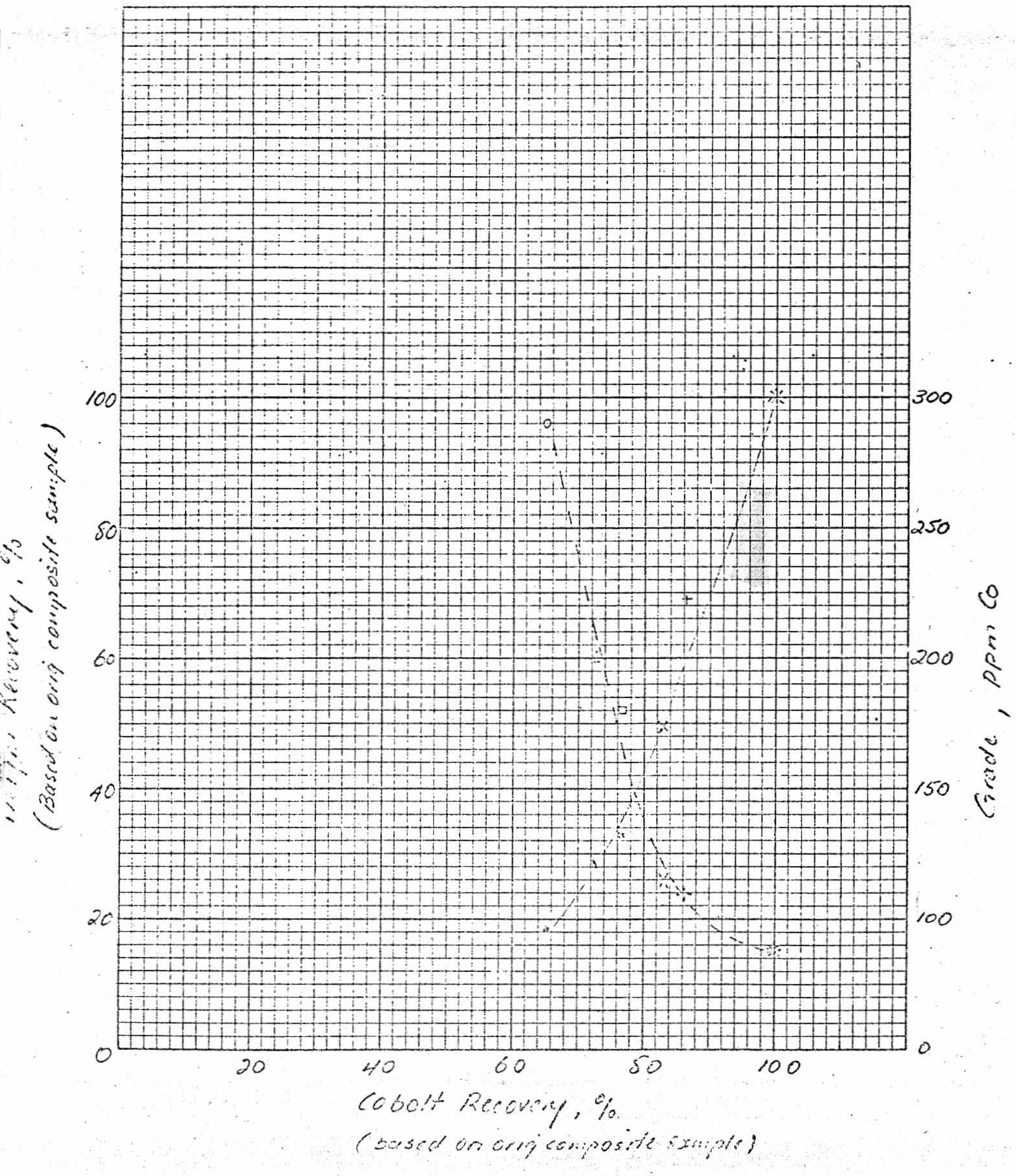


FIG 4. COBALT BENEFICIATION OBTAINABLE AS A RESULT OF LAMFLO AND SPIRAL TREATMENT OF COMPOSITE SAMPLE

APPENDIX VIII

LAKEFIELD RESEARCH OF CANADA LIMITED

"The Recovery of Sulphides"

An Investigation of

THE RECOVERY OF SULPHIDES

from an A.C S. Feed sample

submitted by

CITIES SERVICE INTERNATIONAL INC.

Progress Report No. 4

Project No. L.R 1784

NOTE:

This report refers to the samples as received.

LAKEFIELD RESEARCH OF CANADA LIMITED
Lakefield, Ontario
May 12, 1976

261143

I N T R O D U C T I O N

One March 29, 1976, Dr. D.D. Jinks advised via telephone that a small sample of ore was being shipped to Lakefield for testing. The objective of the testwork was to determine the amount of sulphides, in particular chalcopyrite, which could be recovered by flotation.

The sample, weighing 2.5 kg. represented typical feed from the Macquarie Harbour sulphide project, and was forwarded by Robertson Research (Australia) Pty. Limited, Bowral, N.S.W.

LAKEFIELD RESEARCH OF CANADA LIMITED

A.G. Scobie, P. Eng.,
Manager

D.M. Wyslouzil, P. Eng.,
Chief Metallurgist

Investigation by: D.R. Shaw

S U M M A R Y

1. Head Analysis

The analysis of the A.C.S. Head Feed sample, as calculated from the two flotation tests, was as follows:

Copper (Cu)	0.17 %
Sulphur (S, Total)	3.61 %

1.1. Size Analysis

A.C.S. Head Feed

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 20	0.1	0.1	99.9
28	0.2	0.3	99.7
35	1.1	1.4	98.6
48	9.2	10.6	89.4
65	25.1	35.7	64.3
100	34.0	69.7	30.3
150	16.7	86.4	13.6
200	8.0	94.4	5.6
270	3.1	97.5	2.5
400	1.3	98.8	1.2
- 400	1.2	100.0	-
Total	100.0	-	-

2. Metallurgical Results

Two tests were performed to evaluate the response of this material to flotation. In both tests a rougher concentrate was removed from the material "as received". As shown in the size analysis, the particle size distribution was typical of a deslimed feed, and hence the sulphides were present as coarse free particles or were finely disseminated in coarse quartz. The free sulphides floated rapidly.

Summary - Continued

2. Metallurgical Results

Table 1

Flotation of A.C.S. Feed (Test 41)

Product	Weight %	Assays, %		% Distribution	
		Cu	S	Cu	S
Concentrate	7.0	0.65	40.6	28.5	85.8
Tailing	93.0	0.12	0.51	71.5	14.2
Head	100.0	0.16	3.34	100.0	100.0

The total sulphur recovery of 85.8 % and the grade of 40.6 % S were satisfactory considering the coarseness of the feed, but the copper recovery was low.

To determine the amount of copper which could be recovered by regrinding and scavenging, the rougher tailings from Tests 40 and 41 were reground to 71 % and 87 % minus 200 mesh, respectively.

Low-grade scavenger concentrates were removed (low-grade with respect to sulphur analysis) but the tailings assays were lowered to 0.02 % Cu (Test 41) and 0.03 % Cu (Test 40), respectively. The combined rougher and scavenger concentrates had the following composition.

Table 2

Combined Rougher and Scavenger Concentrate

Test No.	% -200 mesh after regrind	Weight %	Assays, % Cu	Assays, % S	% Recovery Cu	% Recovery S
40	71.6	15.3	0.99	24.1	84.6	95.0
41	87.4	23.8	0.61	13.7	90.5	97.9

In order to get maximum information from the two tests that could be performed with the available material, the cleaner circuits were carried out as follows:

Summary - Continued

2. Metallurgical Results

Test 40: Rougher concentrates 1 and 2 were combined and cleaned twice by reflotation. The cleaner concentrate was then reground and again cleaned twice. The analyses of the concentrates after each stage are shown in Table 3. A complete flowsheet of this test is shown in Figure 1.

Table 3
Cleaning of Concentrates of Test 40

Product	Regrinding	Weight	Assays, %		% Distribution	
		%	Cu	S	Cu	S
Cleaner Conc.	No	10.3	1.4	35.5	80.9	94.2
Re-cleaner Conc.	Yes	6.3	1.75	48.2	68.6	79.6

The above products were the end-members of two-stage operations, and hence there are intermediate products available, containing sulphur and copper, which could be further processed to increase the recovery. It is clear, however, that from the original A.C.S. Feed, without further grinding, the copper recovery would be low.

Test 41: The rougher concentrate was reground and cleaned separately, whereas the scavenger concentrate was cleaned without further grinding. The results were interesting in that they showed that from the scavenger concentrate a copper concentrate could be produced assaying 5.64 % Cu and containing 54 % of the total copper. The reground rougher concentrate was upgraded to over 50 % S as shown in Table 4.

Summary - Continued

2. Metallurgical Results

Table 4

Selected Products from Test 41

Product	Weight %	Assays, % Cu	Assays, % S	% Distribution Cu	% Distribution S
Rougher Concentrate 1	7.05	0.65	40.6	28.5	85.8
After regrinding and cleaning	4.43	0.93	51.3	25.6	68.1
Scavenger Concentrate	16.78	0.59	2.38	62.0	12.1
After cleaning	1.53	5.64	21.7	53.7	10.0
Combined rougher + scav. conc.	23.83	0.61	13.7	90.5	97.9
Combined cleaner concentrate	5.96	2.02	41.3	79.3	78.1

The final results of these two tests were similar. A satisfactory pyrite recovery was possible from the coarse feed material, but since the chalcopyrite was preferentially associated with gangue minerals in the form of fine inclusions, the recovery was low.

Grinding to about 70 % - 80 % minus 200 mesh would be required to achieve a good copper recovery. Regrinding of the concentrates would be required to obtain a high sulphur-low silica product.

Figure 1

Flowsheet Test 40

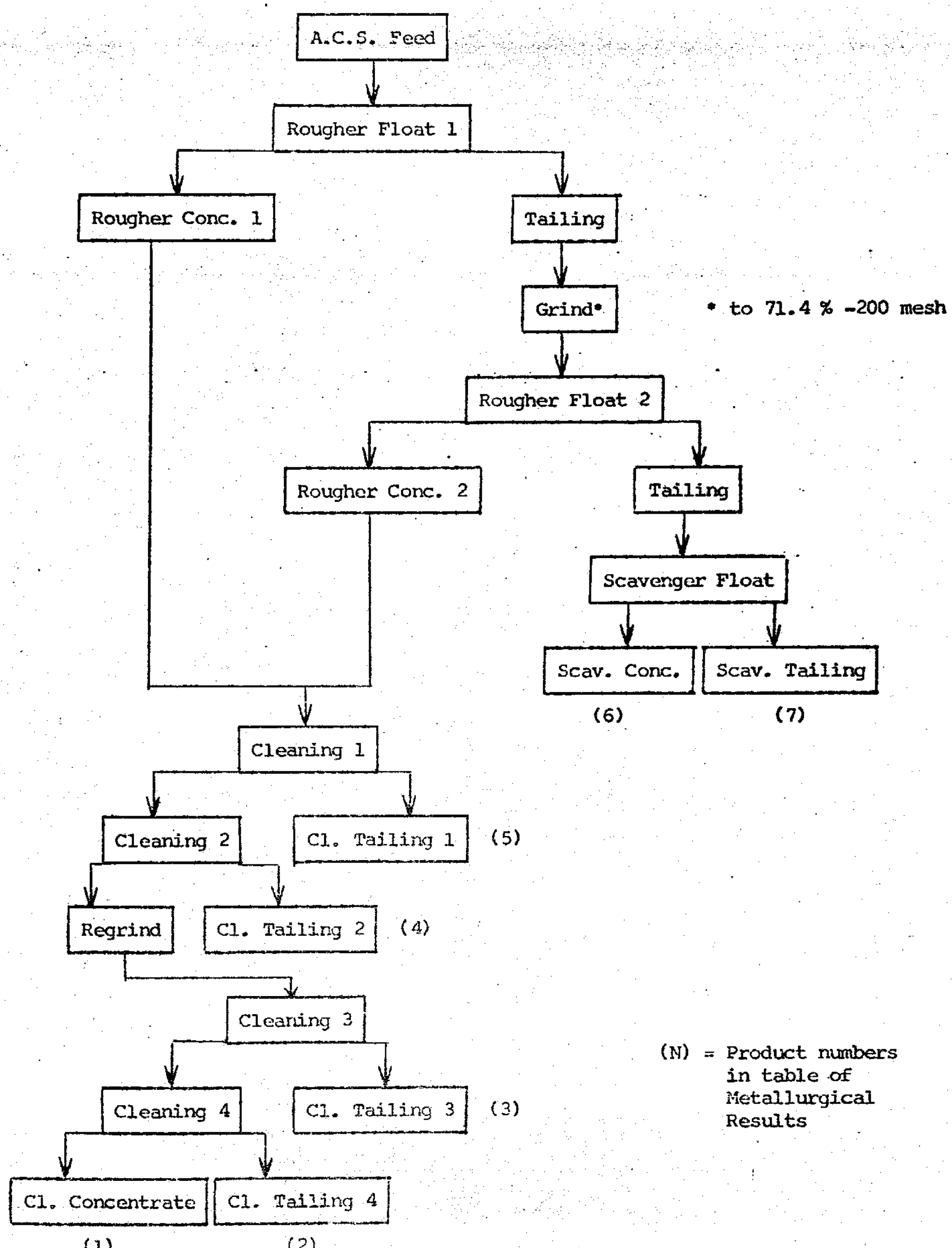
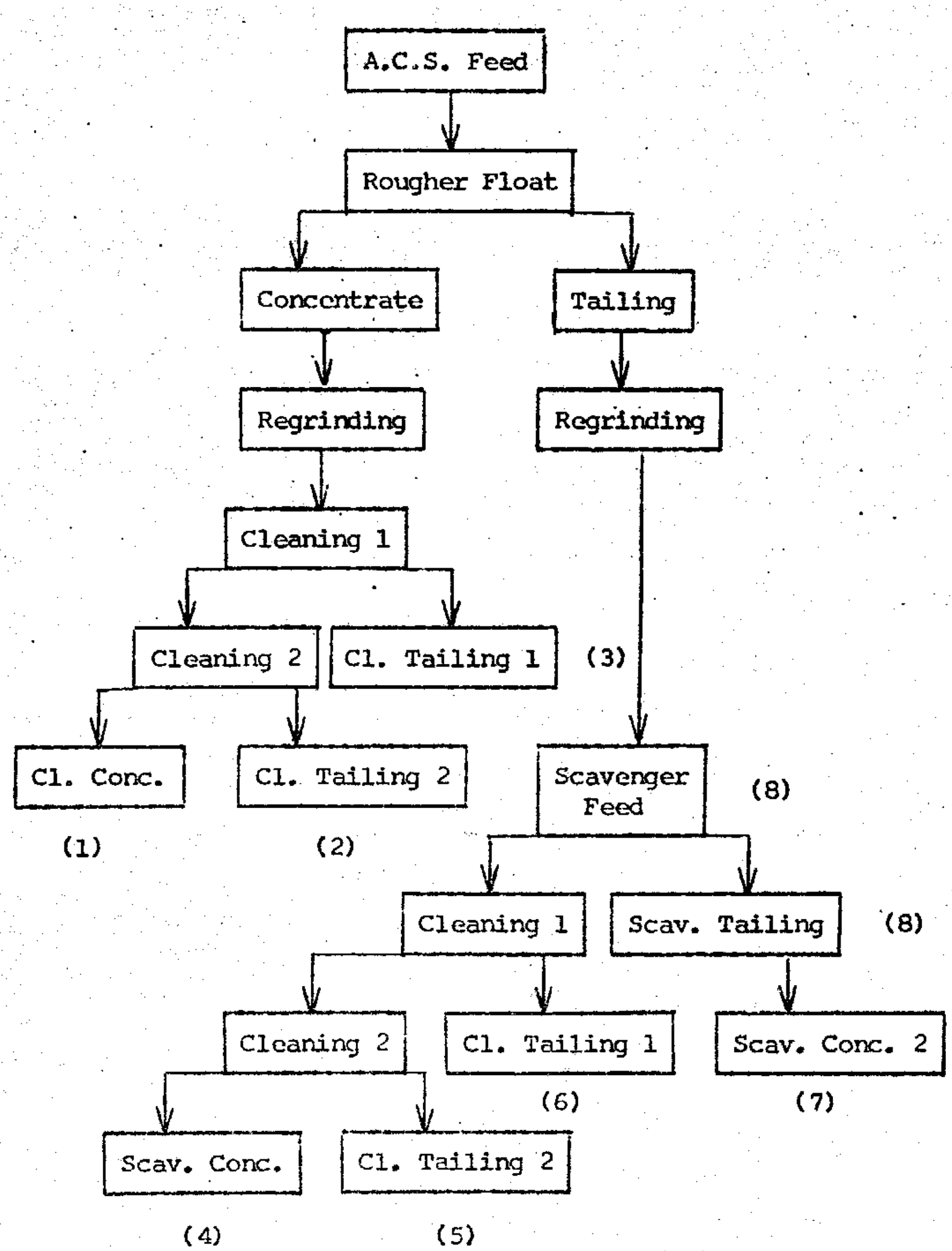


Figure 2

Flowsheet Test 41



DETAILS OF TESTS

Test No. 40

Purpose: To investigate the recovery of a sulphide concentrate from the A.C.S. head feed sample.

Procedure: Float a sulphide rougher concentrate. Grind tailing float a second rougher concentrate and scavenge once. Clean the combined rougher concentrate twice.

Feed: 500 grams A.C S. Head Feed sample.

Observations: Feed - 50 - 60 % coarse grained free pyrite
~ 20 % coarse middling
~ 20 % fine inclusions inclusions in non-sulphide gangue (quartz) considerable tarnished pyrite, especially in finer sizes and middlings.

Ro. 1 - Pyrite responded well, free and coarser middlings.

Ro. 2 - Pyrite responded well more (tarnished) and finer pyrite

Scav. Conc. - Few sulphide (chalcopyrite)

Scav. Tail. - Clean of sulphide, except for very fine inclusion in quartz.

Cleaning - good

Conditions:

Stage	Reagents Added, pounds per ton		Time, minutes			pH
	Z - 6	DF-250	Grind	Cond.	Froth	
Sulphide Rougher (1)	0.04	0.02	-	2	3	6.5
Grind - rougher tailing	-	-	20	-	-	-
Sulphide Rougher (2)	0.04	0.02	-	1	4	-
Scavenger	0.02	0.01	-	1	2	-
Cleaning Combined Rougher Concentrates						
1st Cleaner	0.01	0.01	-	1	3	-
2nd Cleaner	-	0.01	-	1	2	-
45.0 grams 2nd cleaner concentrate reground and cleaned further						
Regrind	-	-	5	-	-	-
3rd Cleaner	0.01	0.01	-	1	2	-
4th Cleaner	-	-	-	1	1½	-

Test No. 40 - Continued

Conditions:

Stage	Flotation	Grind and Regrind
Equipment	250 g D-1	Abbe Pebble Mill
Speed: r.p.m.	950	-
% Solids	Ro. - 30	-

Metallurgical Results

Product	Weight	Assays, %		% Distribution	
	%	Cu	S	Cu	S
1. Re Cleaner Conc.	6.33	1.75	48.2	68.9	79.6
2. Cleaner Tailing 4	0.89	1.06	27.0	5.9	6.3
3. Cleaner Tailing 3	3.05	0.32	10.5	6.0	8.3
4. Cleaner Tailing 2	0.99	0.30	1.73	1.7	0.4
5. Cleaner Tailing 1	4.00	0.09	0.38	2.0	0.4
6. Scavenger Conc.	4.90	0.12	0.39	3.3	0.5
7. Scavenger Tailing	78.81	0.027	0.22	12.1	4.5
Head (Calculated)	100.00			100.0	100.0

Calculated Grades and Recoveries

Products 1 and 2	7.22	1.66	45.6	74.8	85.9
Products 1 to 3	10.30	1.40	35.5	80.9	94.2
Products 1 to 4	11.29	1.30	32.5	82.6	94.6
Products 1 to 5	15.29	0.99	24.1	84.6	95.0
Products 1 to 6	20.19	0.76	18.4	87.9	95.5
Products 6 and 7	84.71	0.03	0.23	15.4	5.0

Test No. 40 - Continued

Screen Analysis

Scavenger Tailing

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	0.2	0.2	99.8
100	1.7	1.9	98.1
150	7.1	9.0	91.0
200	19.4	28.4	71.6
270	18.1	46.5	53.5
400	16.7	63.2	36.8
- 400	36.8	100.0	-
Total	100.0	-	-

Test No. 41

Purpose: To investigate the effects of regrinding the first sulphide rougher concentrate and of a longer grinding time for the rougher tailing on the recovery of copper and pyrite.

Procedure: As for Test No. 40, but with following modifications:

1) regrind first sulphide rougher concentrate
2) increase rougher tailing regrind time from 20 to 30 minutes.
3) clean rougher and scavenger concentrate separately.

Feed: 500 grams A.C.S. Head Feed sample.

Conclusions: Rougher and scavenger similar to Test 40, except that with increased fineness of tailing grind more obvious chalcopyrite response. Copper seems to respond well to cleaning, but pyrite recovery may be significantly lower due to dilution. Considerable silica in scavenger cleaner concentrates, may need further cleaning conditions (larger feed sample) or require small addition of fines depressants (Na_2SiO_3).

Conditions:

Stage	Reagents Added, pounds per ton		Time, minutes			pH
	Z - 6	DF-250	Grind	Cond.	Froth	
Sulphide Rougher	0.04	0.02	-	2	3	6.6
Grind - rougher tailing	-	-	30	-	-	-
Scavenger (1)	0.04	0.02	-	1	4	-
(2)	0.02	0.01	-	1	3	-
Regrind Rougher Concentrate		-	5	-	-	-
Clean rougher and scavenger (1) concentrates separately						
Ro. 1st Cleaner	0.01	-	-	1	2	-
Ro. 2nd Cleaner	-	0.01	-	1	2	-
Scav. 1st Cleaner	0.01	-	-	1	2	-
Scav. 2nd Cleaner	-	-	-	1	2	-

Stage	Flotation	Grind
Equipment	250 g D-1	Abbe Pebble Mill
Speed: r.p.m.	950	-
% Solids	Ro. - 30	-

Test No. 41 - Continued

Metallurgical Results

Product	Weight %	Assays, % Cu	Assays, % S	% Distribution Cu	% Distribution S
1. Rougher Cleaner Conc.	4.43	0.93	51.3	25.6	68.1
2. Rougher 2nd Cl. Tail.	0.29	0.34	29.5	0.6	2.6
3. Rougher 1st Cl. Tail.	2.33	0.16	21.6	2.3	15.1
4. Scavenger Cl. Conc.	1.53	5.64	21.7	53.7	10.0
5. Scavenger 2nd Cleaner Tailing	0.98	0.24	1.19	1.5	0.4
6. Scavenger 1st Cleaner Tailing	5.02	0.07	0.32	2.2	0.5
7. Scavenger Conc. No. 2	9.25	0.08	0.42	4.6	1.2
8. Scavenger Tailing	76.17	0.020	0.10	9.5	2.1
Head (Calculated)	100.00	0.16	3.34	100.0	100.0

Calculated Grades and Recoveries

Products 1 and 2	4.72	0.89	50.0	26.2	70.7
Products 1 to 3	7.05	0.65	40.6	28.5	85.8
Products 4 and 5	2.51	3.53	13.7	55.2	10.4
Products 4 to 6	7.53	1.22	4.78	57.4	10.9
Products 4 to 7	16.78	0.59	2.38	62.0	12.1
Products 1 to 7	23.83	0.61	13.7	90.5	97.9

Test No. 41 - Continued

Screen Analysis

Scavenger Flotation Tailing

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	0.1	0.1	99.9
100	0.4	0.5	99.5
200	2.2	2.7	97.3
200	9.9	12.6	87.4
270	16.0	28.6	71.4
400	20.7	49.3	50.7
- 500	50.7	100.0	-
Total	100.0	-	-

LAKEFIELD RESEARCH OF CANADA LIMITED
Lakefield, Ontario
May 12, 1976 / dmm

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APPENDIX IX

ROBERTSON RESEARCH (AUSTRALIA) PTY. LIMITED

'Macquarie Harbour Sulphides:

Mineral Processing Investigations: June 1975-May 1976"

261157 156

ROBERTSON RESEARCH (AUSTRALIA) PTY. LIMITED

REPORT NO. 423

PROJECT NO. 756/9511

MACQUARIE HARBOUR SULPHIDES:

MINERAL PROCESSING INVESTIGATIONS : JUNE 1975-MAY 1976

by

R. Butler, A.R.S.M., B.Sc. (Eng.)
J.W. Bruinsma, Geol. Drs. (Amsterdam), A.M.A.I.M.M.
K. Andrews, B.Sc. (Met.), B. Econ., J.M.A.I.M.M.

Prepared for:

Cities Service International Inc.,
Box 37,
Tulsa, OKLAHOMA.

14th July 1976.

SUMMARY

(1) This report assesses the gravity and flotation testing programmes carried out in Australia on samples from Macquarie Harbour. Testing concentrated on the search for a low cost treatment method capable of producing a commercial sulphide concentrate without grinding of the "ore."

(2) The samples investigated contained 3.75% sulphur (principally as pyrite) and 0.16% copper, with minor amounts of other base metals.

(3) Gravity concentration is capable of producing a commercial sulphide concentrate (approximately 40% S) with good recovery (approximately 80%) but copper recovery is poor (∿20%).

(4) Froth flotation will produce a similar or better sulphur grade and recovery, but copper recovery is much better (>50%).

(5) Further improved copper recoveries may be possible in a usable sulphide concentrate, but would involve an increasingly sophisticated plant. The possibilities of flotation in this respect were not fully investigated.

(6) The treatment problems arise from the nature of copper mineralisation, which appears to occur as primary and secondary sulphides, liberated and in composites, as well as a minor amount in slag and oxide form. In marked distinction, almost all the sulphur is present as primary pyrite only.

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3.	MINERALOGY	5
4.	TESTWORK : GRAVITY PROCESSING	9
	FLOTATION	11
5.	CONCLUSIONS	13
	APPENDIX : FLOTATION RESULTS	14

1. INTRODUCTION

Australia Cities-Service have rights to a large alluvial deposit containing approximately 5 percent of pyrite and significant base metal values, in Macquarie Harbour, Tasmania. The material is unconsolidated deltaic sands and fines, mostly lying below high water mark, at the mouth of the King River, with more than 50 million tons to dredgable depth. Most of the values will have originated as tailings from mine workings at Mount Lyell, but it is believed that some may be naturally derived detrital material. It also appears that some of the base metals may be present in secondary form after solution transportation.

The basic thinking behind the investigation reported herein, was that it might be possible to produce a pyrite concentrate suitable for acid manufacture by a large-scale low cost dredging and gravity processing operation. It was reasoned that the base metal (and any precious metal) values would be concentrated in the pyrite concentrate and could be economically recovered from the pyrite cinder by leaching or volatilization (e.g. Dowa process), subsequent to burning the pyrite in the acid plant, with the final cinder after removal of copper, zinc and other base metals, being suitable for blast furnace feed. This process is operated commercially in Japan, Germany and the U.S.A.

Processing testwork was undertaken at the following Australian laboratories, with the objective of establishing a viable treatment method:

- Amdel, Adelaide : Lamflo sluice and spirals
- Mineral Deposits, Southport : Reichert cone and spirals
- Robertson Research, Bowral : Flotation testing, mineralogy, etc.

In addition, flotation testwork was undertaken in Canada at the laboratories of Lakefield Research. The testwork at Australian laboratories was coordinated by Mr. R. Butler, chief metallurgist, Robertson Research Australia.

This report summarizes and interprets all the work carried out and results achieved. Fuller details of the work undertaken by Mineral Deposits and Amdel are contained in the following reports:

- AMDEL ME 2072/76 — March 1976 — "Lamflo Sluice and Spiral Beneficiation Tests" - W.G. Rogers.
  see appendix 76-1180
- MIN. DEPOSITS 15.213.1/1 — April 1976 — "Cone Concentrator Studies and Circuit Designs on Alluvial Sulphide Ore" - M.S. Cross
- MIN. DEPOSITS 15.213.1/2 — May 1976 — "Spiral Concentrator Studies and Circuit Designs on Alluvial Sulphide Ore." - M.S. Cross

2. SAMPLES TESTED

The samples tested by the respective laboratories consisted of:

Amdel : 25 x 44 gal drums

Mineral Deposits : Approx. 80 x 44 gal drums from Holes 211-221 (plus 130 other drums)

Lakefield : 1 x 110 pound wet pulp sample

Robertson Research : ex Mineral Deposit Composite

The composition of the various composites reported by the laboratories was:

	Amdel	Min.Dep.	Lakefield	RRA
Size Analysis (cumulative %)				
+300 micron		7.5		14.4
+210 "	31.3	28.7	0.2	
+150 "	56.0	58.0	3.5	78.8
+105 "	77.5	81.3	25.6	
+ 75 "	88.9	91.3	55.3	96.8
Analysis (%)				
Sulphur	3.75	4.12	3.70	3.30
Copper	0.15	0.16	0.13	0.17
Zinc	0.018		0.012	
Cobalt	0.008			
Molybdenum	0.005			

COMMENTS:

The samples tested by Amdel and Mineral Deposits were similar, and said to be representative of the bulk of the available material. In both cases, the material under test varied from time to time as the bulk samples were used drum by drum. The RRA sample was a head sample taken from one particular Mineral Deposits test run.

The Lakefield sample was much finer and presumably typical of a section of the delta further out from the river mouth.

The samples tested at this stage of the programme differed very considerably from those examined in the preliminary stages. This was not unexpected insofar as the earlier samples were either superficial surface pit or random drill hole samples. Initial testing at Lakefield in late 1974 was on very fine slime material, with the following composition:

Sizing :	14.5 percent retained at 75 micron
Analysis :	4.8% Sulphur 0.09% Copper

Preliminary testwork at Mineral Deposits in early 1975 was undertaken on very coarse material with the following properties:

Sizing :	92.0 percent retained at 75 micron
Analysis :	4.4% Sulphur 0.17% Copper

Generally, however, the results between the different laboratories were consistent, with the coarser fractions tending to show a greater amount of unliberated composite material.

3. MINERALOGY

Mineralogical examination was carried out by Lakefield Research and Robertson Research, on various samples. The following summary details the conclusions which can be drawn from ore microscopy and mineral processing results regarding the nature of the mineralization:

1. The bulk of the sulphur content is present as liberated pyrite - chalcopyrite

   (a) In Lakefield Test 5, 70 percent of the sulphur was recovered as a concentrate assaying 50% S. 95 percent of the concentrate was liberated pyrite, chalcopyrite or pyrite + chalcopyrite composites.

   (b) On the much coarser Mineral Deposits preliminary sample, 90 percent of the sulphur was recovered as a concentrate assaying 15% S. It was estimated microscopically that 30-50 percent of the sulphides were free.

   (c) On the bulk sample tested by Mineral Deposits, composites appeared to account for less than 20 percent of the sulphide present. Most of the composites were coarser size gangue particles with only minor amounts of sulphide, as inclusions.

   (d) On the same bulk sample, RRA achieved by flotation a 93 percent recovery of sulphur at a grade of 32.6% S.

2. Copper is present in several forms: only about 20-25 percent occurs in the form of predominantly liberated pyrite - chalcopyrite (residual primary) mineralization.

   (a) In the original Lakefield sample, minor amounts of chalcocite, covellite, bornite are reported.

   (b) In the preliminary Mineral Deposits gravity testing, only about 50 percent of the copper content reported with 90 percent of the sulphides in the gravity concentrate.

   (c) In the Amdel gravity testwork, the best results gave a recovery of only 24 percent of the copper with 80 percent of the sulphides.

(d) Similarly, in the Mineral Deposits study, copper recovery of about 26 percent was associated with a recovery of 85 percent of the sulphides.

(e) Gravity concentration would be expected to make a substantial recovery of copper occurring as discrete grains of copper mineral above about 20-40 micron size and as inclusions associated with other heavy particles (e.g. pyrite).

3. <u>A greater proportion of the total chalcopyrite content is locked than of the pyrite. This is estimated as about ten percent of the total copper content.</u>

(a) It must be remembered that the deposit is largely the residue from copper and gold winning activities over the years. Whilst there has been some sporadic recovery of pyrite values, in general the ratio of pyrite to chalcopyrite will be considerably greater than that of the original Mt. Lyell mineralization.

(b) It is suggested that the natural ratio of pyrite to chalcopyrite will probably be preserved in the composition of sulphide inclusions. As the natural ratio is approximately 20:1, and gravity concentration losses are only about 10-20 percent, with perhaps 0.70 percent as pyrite inclusions, i.e. 7-10 percent of total pyrite, losses as chalcopyrite inclusions are probably of the order of 0.04 percent, i.e. 0.015 percent Cu.

(c) Mineralogical examination of tailings from tabling and cone tests at Mineral Deposits confirmed the presence of some losses as fine pyrite/chalcopyrite inclusions. The quantity was relatively minor. A greater loss of pyrite occurred as composite coarse (+150 micron) particles, associated with gangue. Such particles tended to be predominantly gangue with minor sulphides. Pyrite, chalcopyrite and chalcocite were reported as occurring in this form, but relative proportions could not be estimated.

4. <u>Only a minor (negligible?) amount of copper is present in the form of oxide copper mineral, or as slag.</u>

(a) Only copper sulphide minerals were observed in mineralogical examinations at Lakefield and Robertson Research.

(b) Only 2-4 percent of the copper present in the RRA samples is soluble in dilute acid at ambient temperature conditions. These conditions would dissolve oxide minerals (chrysocolla only partly) but not copper sulphides.

(c) A very small amount of granulated slag was observed in the coarse size fractions. It constituted a very small fraction of the whole sample (<0.1% perhaps) and therefore could not account for any significant portion of the copper content.

(d) Siderite is present, and constitutes approximately 3 percent of the head. There is no appreciable concentration of copper with the siderite.

5. The remaining "50 plus" percent of copper mineralization appears to be present as secondary sulphide but has not been fully identified. It may occur in several forms.

(a) Secondary minerals, bornite, chalcocite and covellite were identified in the polished sections prepared by Lakefield Research.

(b) Chalcocite was also identified in the polished grain mounts examined by Robertson Research. Approximately twice as much chalcopyrite was present, as chalcocite.

(c) In both cases, the samples examined would have been granular and free of slimes. The finer fractions show a slight concentration of copper, which is presumably present as the sulphide, chalcocite. The Mineral Deposits head sample assaying an average 0.16% Cu, contained 0.20% Cu in the minus 20 micron fraction, but with a fairly uniform copper distribution throughout the size range.

(d) The flotation testwork undertaken by Lakefield and RRA gave much better recovery of copper than gravity concentration for similar sulphide sulphur recovery and grade. This is consistent with the additional flotation recovery of fine chalcocite, and is further supported by the slow flotation of copper (which is characteristic of fine chalcocite).

	Grade		Recovery	
	Cu	S	Cu	S
5 min.	0.79	25.8	54.3	92.3
20 min.	0.64	17.2	85.4	97.8

It seems reasonable to postulate the following genesis for the deposit as it exists:

(1) The deltaic deposit has been formed from material carried down the river, derived by natural weathering and mining operations in the vicinity of Mt. Lyell.

(2) The material brought down from the mining operations will contain pyrite but will generally be depleted in free chalcopyrite particles. Such chalcopyrite as is present will largely be associated with pyrite. Other base metals will also tend to be closely associated with the pyrite.

(3) Some copper (and other base metals) has been introduced as fine slag particles. It is likely that the river waters have tended to be acidic and carry anomalous copper values.

(4) Under the likely conditions existing in the delta, especially below the normal water table (tending to be reducing/ acidic), copper could be precipitated as the secondary minerals, chalcocite, etc., in the form of ultrafine particles or as coatings on other minerals. Zinc is similarly mobile, but cobalt is largely fixed within the pyrite lattice.

(5) Thus there are two distinct sulphide populations:

RESIDUAL PRIMARY SULPHIDES accounting for all the pyrite, and the cobalt content, some of the copper and zinc (most of the sulphur).

SECONDARY SULPHIDES, mostly ultrafine and in process of formation, accounting for a considerable portion of the copper and zinc.

(6) From the foregoing, the following possible distribution of the copper content can be hypothesized:

Chalcopyrite/pyrite, gravity recoverable	- 20-25%
Chalcopyrite/pyrite, as minor constituent (inclusion) irrecoverable by gravity or flotation	- 10%
Oxide copper	- 2-4%
Secondary sulphide copper - fine chalcocite - recoverable by flotation only	- 60%

4. MINERAL PROCESSING TESTWORK

Processing investigations followed two distinct lines of thought, in an attempt to achieve a satisfactory concentration of the sulphides. Preliminary estimates indicated a very clear advantage to large scale gravity processing, using current beach sand technology, if satisfactory recoveries could be achieved. Operating costs (dredging and gravity concentration) of the order of $0.30 per tonne dredged might be achieved, compared with $0.50-0.70 , or even higher, if flotation had to be employed. In addition, capital costs for a gravity plant would probably be considerably lower.

4.1. GRAVITY PROCESSING

Investigations were undertaken by the following laboratories:

- AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES : Beneficiation by Lamflo sluice and spirals.
- MINERAL DEPOSITS : Beneficiation by Reichert cone concentrator and spiral.

In both laboratories, satisfactory concentration of the pyrite could be achieved, but recovery of the copper content was very poor, with no significant upgrading. The results of testwork at each laboratory have been reported in detail elsewhere.

At Amdel, Lamflo sluice concentrators were used to produce a primary (rougher) concentrate which was further upgraded by a Mineral Separator spiral to give a final concentrate. A number of tests were run to evaluate different equipment parameters. In the most promising test, the sulphur recovery was 80 percent, the copper recovery only 24 percent. The details are as follows:

	Grade		Recovery		
	S	Cu	Wt.	S	Cu
Feed	3.42	0.17			
Lamflo sluice Conc.	4.55	0.18	~~6.93~~ 69.2	92.0	74.2
Spiral concentrate	14.6	0.23	19.1	79.7	23.7

More thorough testing to achieve a higher grade concentrate was not carried through because of the obvious difficulty of recovering the copper values in this circuit.

At Mineral Deposits, Reichert cone concentrators were used to achieve a primary bulk separation of the heavy mineral values, with further upgrading of the concentrate by means of spirals. A considerable number of tests were carried out to establish equipment parameters, and to develop a flowsheet for concentrating the pyrite content. In the most satisfactory tests, a final concentrate assaying 43.9% S was achieved, but copper recovery was very poor. From these tests, the predicted performance for a closed cycle operation were:

	Grade		Recovery		
	S	Cu	Wt.	S	Cu
Feed	3	0.16			
Reichert cone conc.	16	0.26	16	85	26
Spiral concentrate	40	0.33	7	80	12

These results serve to confirm the conclusion drawn from the gravity concentration testwork at Amdel, that the pyrite fraction can be satisfactorily concentrated by gravity methods, but that the bulk of the copper content is not recoverable by gravity methods.

The poor recovery of copper confirms the view that only about 25 percent of the copper occurs in close association with the pyrite content, or as free granular sulphide mineral (chalcopyrite). The 75 percent unrecovered must occur either in the fine fraction (below 30-40 micron sizing) not amenable to concentration by gravity sluice, or as a relatively minor component of light granular material, which is rejected by the sluice. Some undoubtedly occurs in this latter form as very minor inclusions (chalcopyrite) in gangue particles, but should not be significantly more than the amount of pyrite occurring in the same mode. It must thus be concluded that the bulk of the copper unrecovered by gravity processing occurs in the fines or possibly as very fine surface coating material on coarser gangue particles.

4.2. **FLOTATION**

A considerable number of flotation tests have been carried out by Lakefield Research, on two samples. The possibilities of bulk flotation, bulk flotation with middlings regrind, selective copper flotation and various leaching and roasting tests were investigated.

Flotation tests were undertaken in the RRA laboratory as part of the investigation of copper not recovered by the gravity concentration processes. Six tests were completed:

(1) FLOTATION OF SULPHIDES IN SLIMES. This test was undertaken on decanted slimes from the Mineral Deposits test rig. Most of the values would not be recovered by gravity concentration. Flotation recovered 97 percent of the sulphide sulphur and 59 percent of the copper.

(2) FATTY ACID FLOTATION OF SLIMES. This test was undertaken on the tailings from the previous test. The objective was to concentrate the carbonate content (siderite, principally) together with any copper carbonates associated with it. The flotation was unsuccessful in concentrating the copper.

(3) FATTY ACID FLOTATION OF SLIMES. Flotation of the siderite fraction from the decanted slimes resulted in no significant concentration of copper, all products having approximately the same assay.

(4) AMINE FLOTATION. The possibility existed of some of the copper content occurring in intimate association with the illite/mica/chlorite content. Amine flotation to concentrate these minerals resulted in only minor concentration of the copper content. It can be concluded that a small amount of copper occurs in this form. Its recovery would not be economic.

(5) SULPHIDE AND CARBONATE FLOTATION. This flotation was carried out on a representative sample. Using standard flotation technqiues and relatively short flotation times, 54 percent of the copper reported with 92 percent of the sulphides. There was only minor concentration of the copper with the siderite concentrate.

(6) SULPHIDE FLOTATION. This test repeated the conditions of test 5 sulphide float with extended flotation time. The longer float resulted in a recovery of 85 percent of the copper and 97 percent of the sulphur in a concentrate after rejecting 79 percent of the feed.

Fuller details of each of these tests are included in Appendix 1. Some of the tests were designed to provide answers to some of the problems regarding the location of the copper losses. In undertaking sulphide flotation tests, it was considered appropriate to bias the tests towards the practical possibilities of a bulk flotation of unground material, using the simplest reagent combination possible.

The results of the flotation tests show that an acceptable grade of sulphide concentrate with over 50 percent copper recovery is achieved in the simplest possible flotation circuit (short retention time, minimal cleaner flotation and reagent requirement, no grinding). The extension of flotation capacity achieves a greater copper recovery -- 70 or 80% -- depending on retention time allowed, but this would result in dilution of the concentrate and necessitate additional cleaner flotation capacity. (Test 6)

The low copper recovery results from a variety of factors. Some of the copper is present in a form (fine inclusions, slag, etc.) in which it is not amenable to concentration by gravity or flotation processes. A considerable proportion is amenable to flotation, but floats slowly, requiring a long contact time. This is consistent with the presence of fine secondary copper mineral, such as chalcocite -- it is an axiom that large particles float faster than the smaller -- though it has been difficult to confirm this by mineralogical examination.

5. CONCLUSIONS

The overall practical conclusions regarding the treatment of Macquarie Harbour sulphides are:

(1) Gravity concentration effectively recovers the pyrite content but not the copper content. It is capable of producing a 40 percent sulphur product accounting for 80 percent of the sulphide, but only about 20 percent of the copper.

(2) A relatively simple flotation procedure works better and is capable of producing a comparable sulphur grade and recovery, with 50 percent recovery of the copper or better.

(3) A product which is satisfactory except possibly in terms of copper recovery can be produced by flotation without grinding, with a single reagent addition, with short retention time (5 minutes' roughing) and using a simple single-stage cleaning circuit.

(4) Improvement in the copper flotation recovery can be achieved by longer flotation, but at the expense of product grade. Additional cleaning stages may improve the grade without excessive loss of copper values, but this possibility has not been thoroughly investigated.

(5) No combination of gravity and flotation processing would appear to offer any advantage over flotation alone.

APPENDIX 1

The following diagrams summarise the conditions and results of the flotation testwork. Each test is shown in the form of a flowsheet identifying successive products.

Results at each stage are given as percent and shown in boxes, using this key:

Weight	Copper Grade	Sulphur Grade
	Copper Recovery	Sulphur Recovery

In some cases a single set of figures only is given. In these instances, only copper values are included.

Reagent quantity and point of addition are all indicated in the following manner:

HMP 25 ⟶

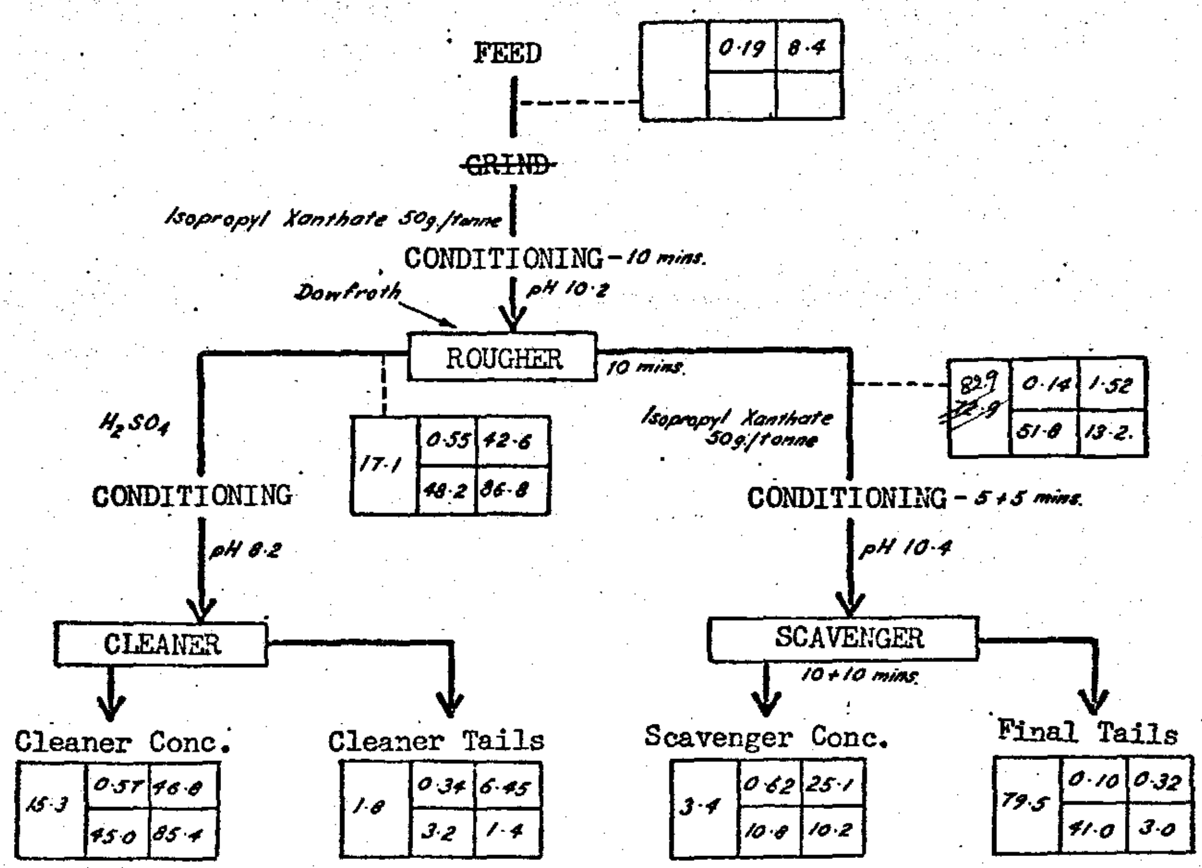
which signifies:

Sodium Hexametaphosphate 25 g/t

Frother quantities are not shown, since frother is only added as required to maintain quality of froth.

261174

TEST 9511-1



PURPOSE: To investigate the recovery of fine sulphides, not recovered by the gravity concentrator.

FEED: Reichert cone decanted slimes (N.B. - 8.4% S 0.19% Cu)

GRIND: No grind

FLOTATION CELL: Denver 500 gm

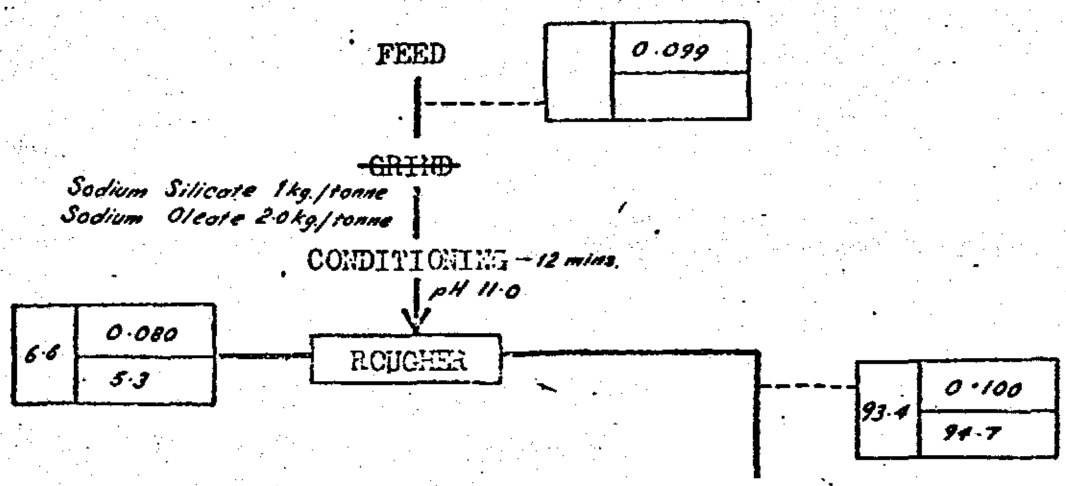
SPEED: 2500 rpm

PULP DENSITY: 25% solids

COMMENTS: A good recovery of sulphides is achieved in a simple rougher float, with scavenging raising recovery to 97 percent at a combined grade of 39.7% S. The recovery of copper is lower and slower. Of particular significance is that the highest grade of copper is achieved in the scavenger concentrate. This suggests that flotation of the copper sulphides (chalcocite?) is not complete.

261175

TEST 9511-2



PURPOSE: To check the possibility that some of the copper contained in the slime tails after flotation of sulphides may be locked with the siderite/calcite.

FEED: Tailings from sulphide flotation of slimes (Test 9511-1).

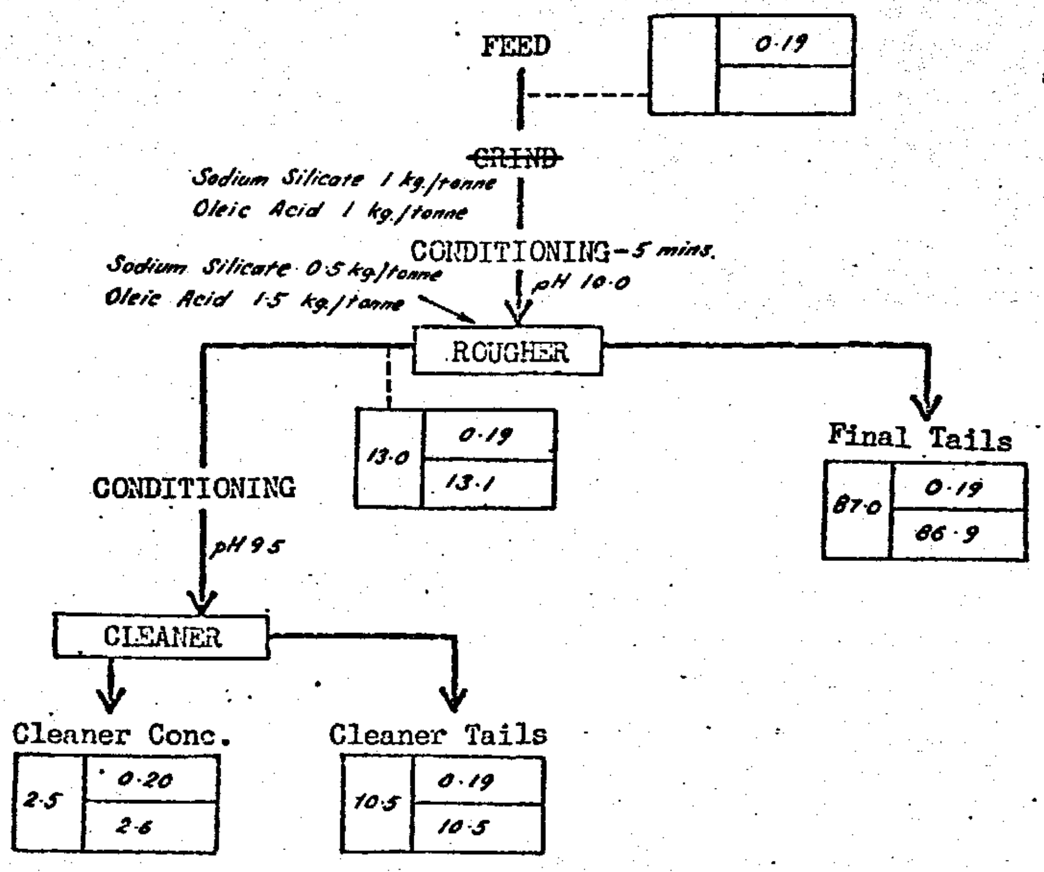
GRIND: No grind.

FLOTATION CELL: Denver 250 gm.

COMMENTS: Although a siderite/calcite concentrate was produced, there is no significant copper content in it.

261176

TEST 9511-3



PURPOSE: To repeat test 2, using slimes decanted from the Reichert cone test as feed.

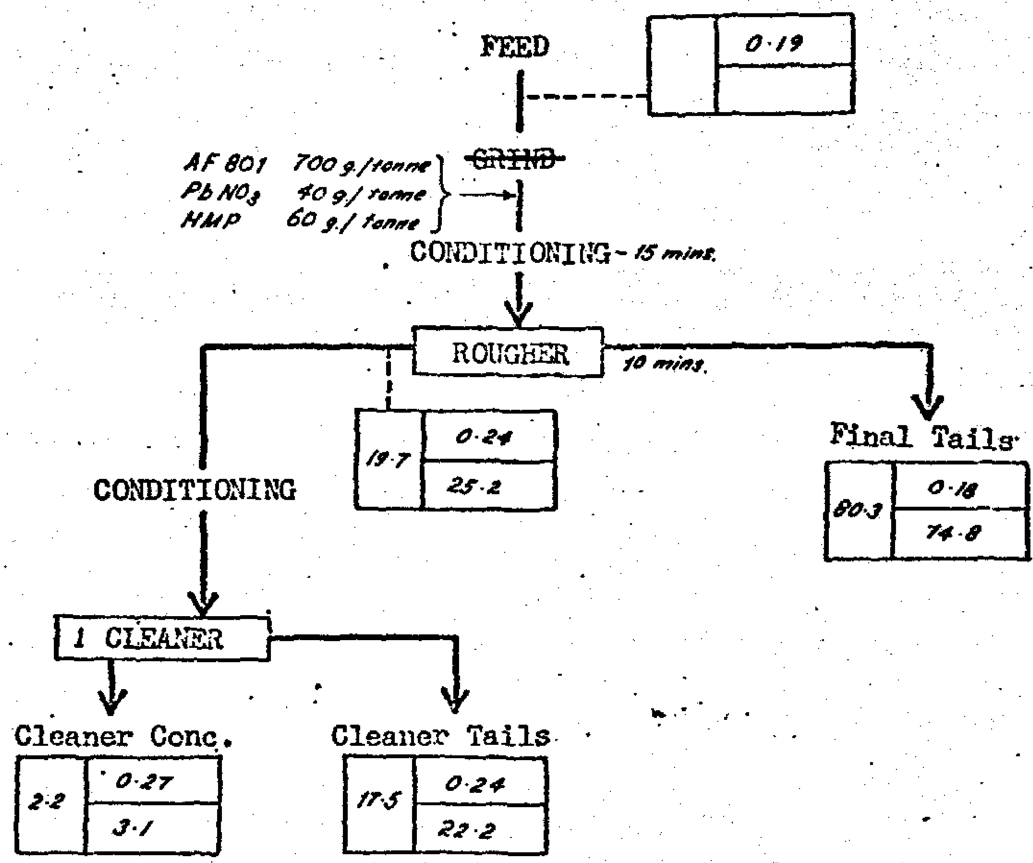
FEED: Reichert cone decanted slimes

GRIND: No grind

FLOTATION CELL: Denver 500 gm

COMMENTS: Again there is no significant concentration of copper with any of the products.

TEST 9511-4



PURPOSE: To determine what portion of the copper lost in slimes was associated with the illite/mica chlorite fraction. An amine reagent was used to selectively float these minerals.

FEED: Reichert cone decanted slimes.

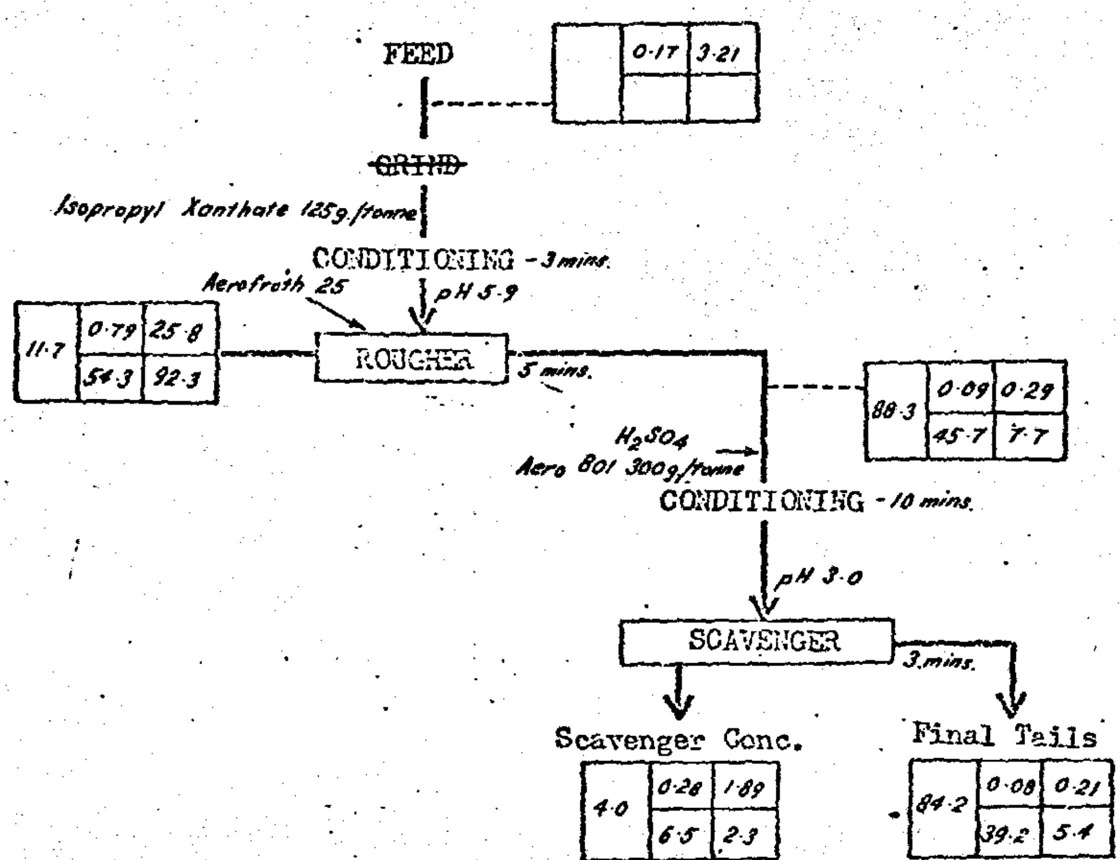
GRIND: No grind

FLOTATION CELL: Denver 250 gm

COMMENTS: There is a minor amount of copper associated with the clay fraction. The copper has probably been adsorbed from solution.

261178

TEST 9511-5



PURPOSE: To float sulphide and copper values from the representative head sample using:

- Xanthate float - to recover sulphides
- Aero 801 float - to recover siderite, calcite etc.

FEED: Representative head sample of material tested at Mineral Deposits.

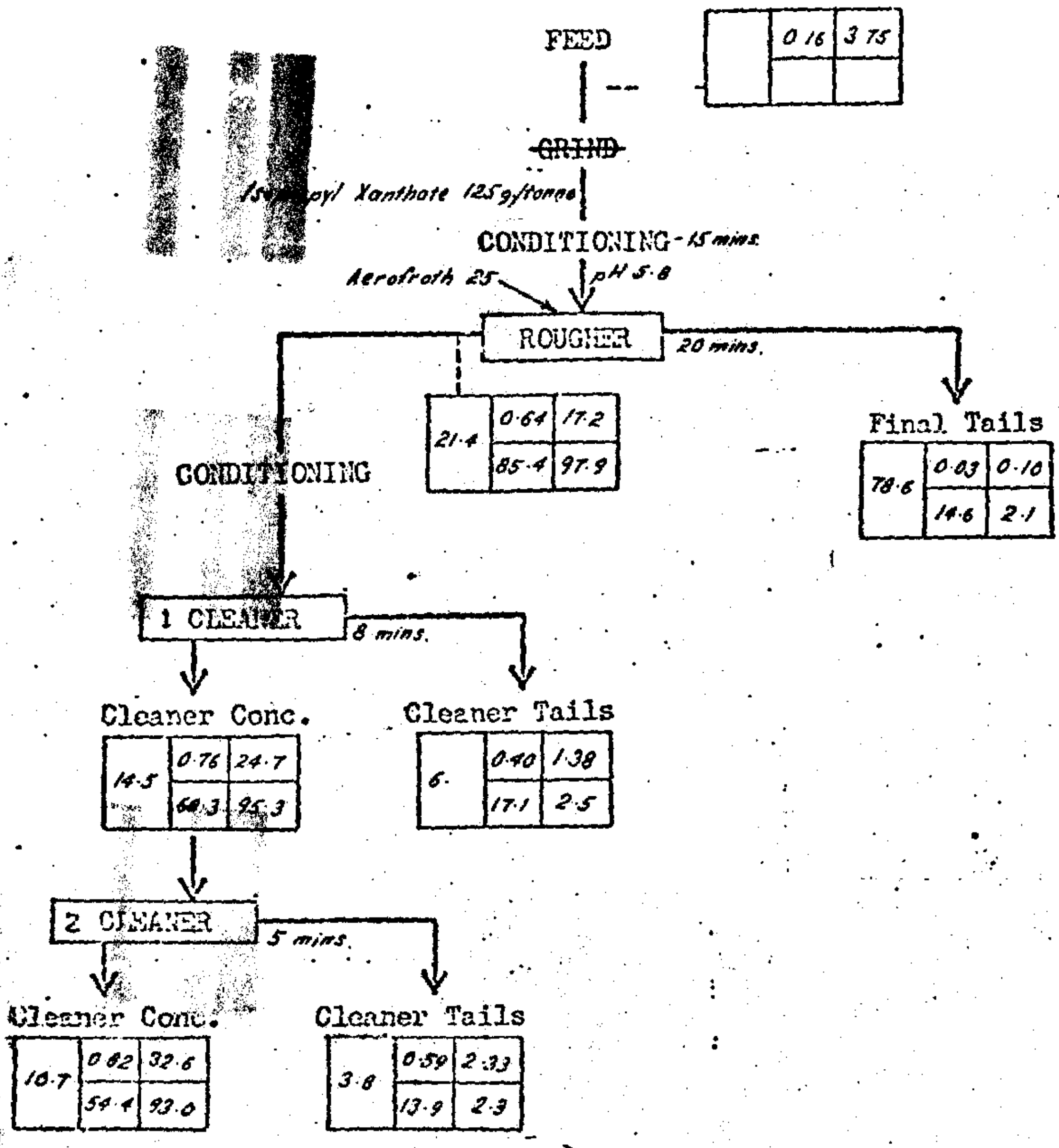
GRIND: No grind.

FLOTATION CELL: Denver 250 gm.

PULP DENSITY: 20% solids.

COMMENTS: In a single flotation step, recovery of copper and grade of sulphides is better than achieved by gravity treatment. Some copper reports with the siderite concentrate, but this may well be sulphides continuing to float (see Test 1) rather than copper-bearing siderite.

TEST 9511 6



PURPOSE: To repeat the sulphide flotation of Test 9511-5, with longer rougher flotation, to permit flotation of fine chalcocite and other copper sulphides which tend to be slow floaters.

FEED: Representative head sample of material tested at Mineral Deposits.

GRIND: No grind.

FLOTATION CELL: Denver 250 gm.

COMMENTS: Rougher flotation time has been quadrupled with marked improvement in copper recovery. This is entirely consistent with presence (and recovery) of fine chalcocite. Although some of this additional copper was dropped on cleaning, it is believed that an improved recovery/grade could be achieved with further testing of the cleaning steps.

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TO OCEAN BEACH
LONG BAY
STRAHAN
TO QUEENSTOWN
TOMMIES BAY
REGATTA PT.
LETTS BAY
DEAD HORSE PT.
BOUY PT.
NUDIST PT.
No 8 BEACON
MAGAZINE IS.
TO OCEAN BEACH
CAT IS.
1 FATHOM LINE
KING RIVER ORANGE BEACON
BLUE BEACON
KING RIVER ROAD
TIPOOKANA
NECK IS.
SWAN BASIN
KING POINT
PINE COVE
8·00W
7·00W
6·00W
5·00W
4·00W
3·00W
2·00W
1·00W
0·0
1·00E
2·00E
3·00E
17·00N
16·00N
15·00N
14·00N
13·00N