814001

MACQUARIE HARBOUR TAILINGS STUDY

E.L. 2/74

MOLYCORP INC.

MARCH 1982

This report has been compiled from information in Department of Mines files. Pages 1, 2, 4 & 5 (probably title page, contents page and locality diagrams) and pages 24 - 30 were not present in the file. The author is not known.

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BACKGROUND

On May 1, 1981, Union Oil Development Corporation acquired Exploration Licenses 2/74 and 16/73 from Elisna Pty. Ltd. The Agreement which accomplished the transfer of E.L. 2/74 and 16/73 from Elisna Pty. Ltd. to Union Oil Development Corporation (hereafter Elisna and Union) contained the mechanism by which Union could reconvey the two E.L's to Elisna if Union's efforts to commercialize the property were unsuccessful. By October of 1981, Union had undertaken mineralogical and metallurgical testing and studied the markets and production economics for the minerals contained in the E.L.'s and reached the conclusion that the property did not offer a sufficient profit potential for Union to develop it. On October 15, 1981, Elisna was notified by Union that Union intended to reconvey E.L. 2/74 and 16/73 to Elisna and to terminate the Agreement.

During Union's evaluation of the property, Union used a sample obtained from Elisna. Union did not seek to determine if the sample was representative of the deposit but operated as if it was with the plan that, should the early test work prove favorable, a representative sample would be taken at a later time. Union also obtained from Elisna the results of previous work done on the property by others. This previous work helped to establish some of the procedural guidelines for scoping work on the economics study of the deposit and, additionally, was treated as background information during parts of the ore dressing and metallurgical studies conducted by Union.

In many instances the report that follows will refer to Molycorp. Molcyorp is a wholly owned subsidiary of Union Oil Company of California, as is Union Oil Development Corporation. In all instances referred to, Molycorp was acting on behalf of Union Oil Development Corporation.

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SUMMARY

1. The Union Oil Company of California acquired an option on the King River Delta deposit in Australia to evaluate the ore deposit in terms of the available technology and price of the metals.

2. According to the Project Development Group's current estimate, the total tonnage is between 85 and 95 million tons, which checks quite well with previous estimates of 100 million tons.

3. The calculated grade average of the deposit obtained by the Project Development Group agrees reasonably well to that of the analyses made by AMDEL and the Colorado School of Mines Research Institute, and are as follows:

Elements	Chemical Analysis
Sulfur (S°)	3.55%
Copper	0.17%
Cobalt	0.01%
Zinc	0.05%

4. A semiquantitative x-ray fluorescence (XRF) analysis of the material did not reveal any other minerals of interest that could be exploited commercially. A fire assay of the head sample showed gold as a possible trace ($<$0.005 oz/ton) and silver as nil.

5. Mineralogic and other examinations confirmed that approximately 20-25% of the copper minerals are present as liberated grains. Fifty plus percent (50+%) of the copper minerals are present mainly as chalcopyrite with traces of bornite; covellite and chalcocite are ultrafine with an average particle size of 10 microns. The remaining 25% of the copper is either locked, oxidized or of slag origin.

6. The cobalt occurs as a solid solution in the pyrite concentrate, assaying 0.01% cobalt. The pyrite concentrate from just about anywhere might well contain that much cobalt as a background level.

7. The flowsheet proposed by CITCO (Figure 1, p. 16) was modified to eliminate gravity separation (poor recovery of copper) and addition of unit processes to produce sulfur (S°), thus eliminating the problem of SO_2 removal.

8. In the recommended flowsheet (Figure 2, p.21), the pyrite is subjected to thermal decomposition to drive off the labile sulfur in elemental form. This eliminates about half of the sulfur normally contained in pyrite. The resulting artificial pyrrhotite (FeS) is then pressure leached in dilute sulfuric acid (Sherritt Gordon), which converts sulfide to its elemental form. The iron is removed by hydrolysis and filtration. The copper and cobalt are recovered via the solvent extraction and electrowinning route.

HISTORY OF THE PROJECT

In early 1970, the Macquarie Harbour tailing deposit attracted the attention of Mr. S.R.M. Harvey because of the unique geologic environment, congenial for formation of a deposit, particularly sulfide precipitation of base metals from solution in addition to the tailings.[1]

PHASE 1 FIELD WORK

Mr. Harvey and associates acquired Exploration License 16/73 covering the harbour in 1973. Samples were collected from the harbour floor sediments, which indicated to be of the type anticipated by Harvey and contained copper, nickel and other metals. An agreement was made with Cities Service International, Incorporated (later CITCO International Minerals Company) for further exploration.

PHASE 2 FIELD WORK

During 1974, Cities Service Company (CITCO) conducted a study of the harbour looking for copper; sixty-two sites were cored. Analysis of the core samples indicated presence of copper. It was also observed that the copper content increase as the delta was approached. Exploration License 2/74 was acquired to cover the King River Delta. The Mt. Lyell and Railway Company Ltd. had previously core drilled the delta extensively looking for pyrite. An analysis of their samples was made available.[2]

PHASE 3 FIELD WORK

CITCO felt that the King River Delta warranted further sampling on the basis of the amounts of copper found during the 1974 coring program. CITCO and Aberdare entered into an agreement, whereby CITCO would assume full project management of Phase 3 of the project.

The objects of Phase 3 were to prove the reserves by grid drilling and to develop and prove the concentration process using full scale equipment. The licenses 16/73 and 2/74 were assigned by Aberdare to a company called Elisna Pty, Ltd., which was formed as an operating company to develop the project.

Development drilling to establish tonnage and grade was completed during the first quarter of 1976. A reverse circulation drilling technique, developed and proved by the Australian mineral sands industry was used.

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(1) All numbers in parentheses correspond to the like number in the References section at the end of this report.

CITCO analyzed the data from the Phase 3 drilling program and reached the conclusion that the deposit held commercial quantities of copper, but that the deposit was not of a magnitude that would interest CITCO. It was recognized that the project was probably profitable for the gravity separation of the pyrite alone. Australian companies that might have a need for sulfur were contacted during 1977 in an attempt to sell the assets of Elisna. At about that time, CITCO decided to abandon international minerals exploration and proceeded to disband the organization. All of the stock of Elisna was assigned to Aberdare in late 1978.

TERRA MARINE, INCORPORATED

In mid-1979, Robin Harvey of Aberdare pointed out to Terra Marine International that the King River Delta could be developed into a profitable venture without trying to recover its copper content. The pyrite and heavy minerals can be recovered by simple gravity separation. The pyrite has a high value for its contained cobalt and possibly for its sulfur, while the tin and tungsten would be profitable in themselves.(3)

In early 1980, Union Oil was informed about the cobalt deposit, King River Delta, Macquarie Harbour, Tasmania.(4) The area is held under option of Terra Marine Pacific, Inc. of Denver.

Since the last investigation of the ore deposit by CITCO, the price of cobalt has increased substantially. Also, because of the geographical location, an attractive elemental sulfur market exists locally. Since abundant hydroelectric power is available at a reasonable rate (approx. US $0.02/kwhr), Union bought into an option to re-evaluate the ore deposit in terms of the present day technology and sales revenues of the products.

ORE DEPOSIT

TONNAGE

The reserves in the delta were estimated by CITCO at approximately 110 million tons proven ore and 28 million tons possible. No mention was made of the method of estimation, or whether the estimate included river sediments or portions in deep water.

A rough estimate was done by Union in 1980. The delta was divided up into 250 meter squares, and the average thickness of each square was calculated by the thickness at each of the four corners. This gave an estimate of 89 million tons in the delta, proper, and 50 million tons in deeper water.

Current Estimate

The polygon method of ore estimation was utilized to provide a third estimate. This method involves constructing a polygon around each drill hole such that each side of the polygon bisects the distance between the hole and its nearest neighbors. The area of influence within the polygon is considered to have the same properties as the information obtained from the drill hole. There is a limit on the radius of the polygon, which in this case was experimentally set at 250 meters and at 500 meters. A plot of the 250 meter radius polygon map may be found in Appendix B. The radius limitation affects only the outlying and edge holes and will be seen to have minimal effect on the reserve estimate. Also, holes with a total depth greater than 32 meters were excluded, as were samples from the riverbed. A list of the assays from each core hole is presented in Table B-1 in Appendix B. Tables B-2 and B-3 contain the polygonal area of influence, the ore thickness, and the calculated pounds of the various elements for each hole, for the 250 and 500 meter polygons, respectively.

Discussion

The total tonnage was calculated to be about 85 million tons, using a radius of influence of 250 meters, and 95 million tons, using a radius of influence of 500 meters. The increase is due to the extended area covered in the outlying holes and the holes at the edge of the ore deposit. The average ore thickness in those regions is small enough that the volume increase is minimal, although the area increase is substantial. Although the estimation is in agreement with previous estimates by other methods, the drill hole spacing is still too sparse to establish a confidence level in the reserve estimate. Variograms constructed with the information have no discernable sill value.

GRADE

The grade of the sample tested at the early stage of the program differed very considerably from those examined in the later stages. Consequently, interpretation and evaluation of results of tests have suffered for lack of comparable data.

The sample tested by AMDEL (1975), Mineral Deposits Ltd. (1976), and the Colorado School of Mines Research Institute (CSMRI) (1981) are similar and claimed to be representative of the bulk of the available material. The analytical results shown below on the samples received by the three organizations compare favorably.

Chemical Analysis	AMDEL (%)	CSMRI (%)	Mineral Deposits (%)
Sulfur	3.75	3.55	4.12
Copper	0.15	0.17	0.16
Cobalt	0.008	0.01	----
Zinc	0.018	0.025*	----

*by XRF

Current Estimate

Based on the chemical analysis of 113 core samples as given in Appendix B, a composite was calculated, and the results are presented as follows:

Analysis	Prorated Composite (%)
Sulfur	----
Copper	0.13
Cobalt	0.009
Zinc	0.018

The calculated average analysis, as above, are fairly well in agreement with the results of analysis from AMDEL and CSMRI, suggesting these samples are perhaps a good representation of the ore body.

A semiquantitative x-ray fluorescence (XRF) analyses of the material did not reveal any other minerals of interest that could be exploited commercially.

A fire assay of the head sample showed gold present as a possible trace ($<$0.005 oz/ton) and silver as essentially nil.

Discussion

The assay values for copper, cobalt, zinc, silver and molybdenum were contoured with respect to the core hole locations to examine the metal decompotition (Figures B-1 thru B-5, Appendix B). The plots show that copper and zinc are concentrated at the mouth of the delta, while cobalt and molybdenum are distributed more evenly over the entire deposition. There are several anomolously high silver assays, one at either end of the deposition, making the worth of the silver contour dubious.

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SIZE DISTRIBUTION

Based on Mt. Leyel Mining and Railway Company's analytical results of 476 samples, a statistical analysis was made by CITCO. It was observed that the statistical curves were composed of two sets of population, one with a normal distribution at the delta toe, and the other a skewed distribution away from the delta.[5] This suggests spatial variability of the deposit with respect to the particle size distribution and the metals content.

In fact, it has been observed by CITCO that the finer size materials of the delta are located further out from the river mouth. With finer particle size, except cobalt as noted earlier, the metal content of the material appears to increase.

Comparison of particle size distribution of the deltaic and offshore sediments, as shown below[6], indicates that over 74% of the offshore sediments (tailings) consists of particles finer than 150 mesh (Tyler). Only 41% of the deltaic sediment (tailings) are finer than 150 mesh (Tyler).

Size Fraction (Tyler mesh)	Deltaic Sample % Fraction	Deltaic Sample Cumulative %	Offshore Sample % Fraction	Offshore Sample Cumulative %
+48	7.5	7.5	---	---
+65	21.2	28.7	0.2	0.2
+100	29.3	58.0	3.3	3.5
+150	23.3	81.3	22.1	25.6
+200	10.0	91.3	29.7	55.3
-200	8.7	100.0	44.7	100.0

Similarly, in another case, the minus 120 micron (equivalent to theoretical Tyler mesh 130) fraction of the deltaic deposit accounts for approximately 31% of the total weight of the sample and contains 40.3% Mo, 73.5% Co, 29.6% Zn, 78% S, and 28.1% Cu.

Although no analysis is available for the comparable offshore sample, it is anticipated that the offshore sample, in addition to containing a larger percentage of fines, will also have a higher metals content.

The gradation of coarse to fine particle sizes of the deposit with distance from the delta would certainly pose problems for gravity separation operations because of greater recovery loss of the metals when the finer sizes are treated.

MINERALOGY

Mineralogical examination of the head samples and concentrates was conducted by Robertson Research, Australia, Lakefield Research of Canada, and the Colorado School of Mines Research Institute, USA.

The highlights of the conclusions drawn from ore microscopy and mineral processing results regarding the nature of the mineralization are presented as follows:

- For coarser material such as the deltaic sample, over 90% of the sulfur was recovered as a sulfide concentrate, assaying 15% S. It was estimated microscopically that 30-50% of the sulfides were free. The rest were locked with gangue.
- For finer material such as the offshore sample, 70% of the sulfur was recovered as sulfide, assaying 30% S. Ninety-five percent (95%) of the concentrate was liberated pyrite.
- Copper is present in several forms. Only about 20-25% occurs in the form of predominantly liberated pyrite-chalcopyrite (residual primary) mineralization.
- About 10% of the total copper in the ore is locked with pyrite.
- About 2-4% of the amount of copper present is in the form of oxide copper mineral, or as slag.
- The remaining 50 plus percent (50+%) of copper mineralization appears to be present as secondary sulfide.

Thus, there are two distinct sulfide populations:[7]

1. The residual primary sulfides, accounting for all the pyrite and perhaps (though not proven) cobalt, some of the copper and zinc, and most of the sulfur.
2. Secondary sulfices, mostly ultrafine, accounting for a considerable portion of the copper (50% or more) and zinc concentrates.

The ultrafine nature of the secondary sulfides is corroborated by the mineralogic examination of a cleaner sulfide concentrate. The major copper mineral present in the concentrate was chalcopyrite, but trace amounts of bornite, chalcocite and covellite were also present. More than 45% of the chalcopyrite was present as fine grained inclusions in pyrite (37 microns or less). Another 10% was present as attached films on pyrite. The average grain size of the chalcopyrite was smaller than 10 microns.[8]

From mineralogic observations above, the difficulty of recovering copper better than 50% by conventional gravity separation equipment is apparent. In order to achieve a sustained 80 plus percent (80+%) recovery of copper by flotation alone, the cost would be prohibitive.

The presence of cobalt as a discrete identifiable species was not observed. Qualitative electron microprobe analysis of grains of pyrite indicated a cobalt peak, and hence, claimed that cobalt is associated with the pyrite. The cobalt assay of the pyrite concentration, determined by chemical method by CSMRI, was 0.072%. According to the CSMRI report, "It seems likely that a pyrite concentrate from just about anywhere might well contain that much cobalt as background level."

FLOWSHEET

PREVIOUS WORK

The Phase 1 and 2 work, described earlier, were directed mainly to ascertain by sampling and analysis whether or not the deposit is of commercial importance. Preliminary process studies were carried out by Hazen Research, Golden, Colorado and the Colorado School of Mines Research Institute, Golden Colorado.[9] The data for the above studies was not available.

Virtually all metallurgical and processing investigations were conducted during Phase 3 field work, spearheaded by CITCO and performed by the following organizations in the areas described:

- AMDEL, Adelaide, Australia: heavy liquid separation and Lamflo sluice and spirals
- Robertson Research, Bowral, Australia: mineralogy and flotation[11]
- Mineral Deposits, Southport, Australia: Riechert cone[12] and spiral studies[13] and circuit designs
- Lakefield Research of Canada Ltd., Lakefield, Ontario: mineralogy and flotation[14]

AMDEL

The best beneficiation results achieved for the total sulfur, copper and cobalt using the Lamflo separator as an initial rougher concentration and spiral separators in a cleaning stage is shown as follows:[15]

	Weight % (feed)	Assay: Percent Total S	Assay: Percent Cu	Assay: ppm Co	Percent Recovery of Feed: Total S	Percent Recovery of Feed: Cu	Percent Recovery of Feed: Co
Concentrate	19.1	14.6	0.23	0.03	79.7	23.7	74.5

It is clear from the test results that effective recovery of copper and perhaps cobalt will not be possible when employing the combination of Lamflo sluice and apirals type of gravity treatment circuits.

Robertston Research

Analysis by Robertson Research revealed that gravity concentration effectively recovers the pyrite, but not the copper. It is capable of producing a 40% sulfur product accounting for 80% of the sulfide, but only about 20% of the copper.[16]

It was also concluded that a relatively simple flotation procedure works better and is capable of producing a comparable sulfur grade and recovery with 50% of copper or better, and that no combination of gravity and flotation processing would appear to offer any advantage over flotation alone.

Mineral Deposits Ltd.

The results of Riechert cone concentration studies indicated a three stage cone circuit with the following results:[17]

- A 16% sulfur concentrate at a recovery of 86.5% with a weight recovery of 16%.
- A copper recovery of approximately 22% in the final concentrate at a grade of 0.26% copper.
- The copper minerals were difficult to find and identify.

A combination of spiral-cone concentration studies revealed that:[18]

- A final sulfide product, assaying 40% sulfur, at an 80% recovery of sulfur and a weight recovery between 6-8% could be achieved.
- Copper minerals were hard to recover. A concentrate assaying 0.33% Cu, at a recovery of 12% copper into the concentrate was achieved.
- From the results above, it was concluded that copper is not with the sulfur minerals to a large extent.

Lakefield Research of Canada

A considerable number of flotation tests were 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. Most of the roasting tests conducted resulted in SO_2 production, perhaps leading to a cumbersome and expensive SO_2 removal process. Most of the roasting work was aimed at sulfation roasting at the laboratory scale. Previous experience at CSMRI indicates satisfactory sulfation roasting at a commercial scale is difficult to achieve.

The results of the Lakefield tests showed that an acceptable grade of sulfide concentrate with approximately 50% copper recvoery was achieved in the simplest possible flotation with:

- Short retention time,
- Minimal cleaner flotation time,
- Minimal reagent requirement,
- No grinding.

Production of flotation concentrates with as high a recovery of copper as 80% was achieved, but at the expense of excessive reagent consumption very long rougher and several stages of cleaner flotation time, and excessive grinding and regrinding, spiraling costs.

Colorado School of Mines Research Institute (CSMRI)

Using the same sample that was used by AMDEL, further tests designed to explore the potential for the recovery of cobalt were conducted by CSMRI recently (June 16, 1981). Their findings summarized below[19] were discouraging, to say the least, for further initiation of chemical processing.

- The heavy liquid test (sp.gr. 3.3) resulted in a heavy fraction of 6.6% of the initial weight of the ore, assaying 0.073% cobalt and recovering 63% of the cobalt from the ore.

- Tests utilizing standard pyrite flotation procedures on underground material with or without scrubbing showed a sulfur recovery of over 97.7%, copper recovery of 25% to 47.1%, cobalt recovery of 64% to 77.8% to the concentrate produced. The percent weight of the rougher concentrate produced ranged from 818% to 12.% (of the ore).

- Optical mineralogic examination of a polished section of the heavy fraction from the heavy liquid spearation tests, assaying 0.073% cobalt, did not reveal specific occurrences of cobalt minerals.

- A fire assay of the head sample showed occurrence of gold as a possible trace (0.005 oz/ton) and silver as essentially nil.

CITCO Flowsheet

Based on the previous work, the process flowsheet, as proposed by CITCO, shown in Figure 1, can be divided into four major sections:

- Mining by suction dredging.

- Preconcentration using Reichert cone and/or Lamflo sluice.

- Final concentration by flotation.

- Hydrometallurgy consisting of roasting, leaching, extraction and electrowinning.

Discussion

The preconcentration by gravity followed by final concentration by flotation appears elaborate and costly. Also, it would have an adverse impact on the recovery of metals such as copper and cobalt. Furthermore, as concluded earlier (p. 13), no combination of gravity and flotation processing would appear to offer any advantage over flotation alone.

The modification suggested is to include one step flotation method for rougher concentration without grind, followed by cleaning of the rougher concentrate with mild regrinding.

The hydrometallurgical treatment for dissolution of values as proposed by CITCO flowsheet does not specify the process, and consequently, is left vague. This gray area gives one room for speculation in capital and operating costs, and gives an impression that economic viability may be at hand.

As will be seen later, the technologically feasible roasting leaching unit processes adopted from Sherritt Gordon for our estimation purposes accounts for over 73% of the total capital investment and approximately 43% of the total operating costs.

The remaining part of the process such as solvent extraction and electrowinning of copper and cobalt values are well established commercial processes, and perhaps will be the least expensive method of extraction and purification.

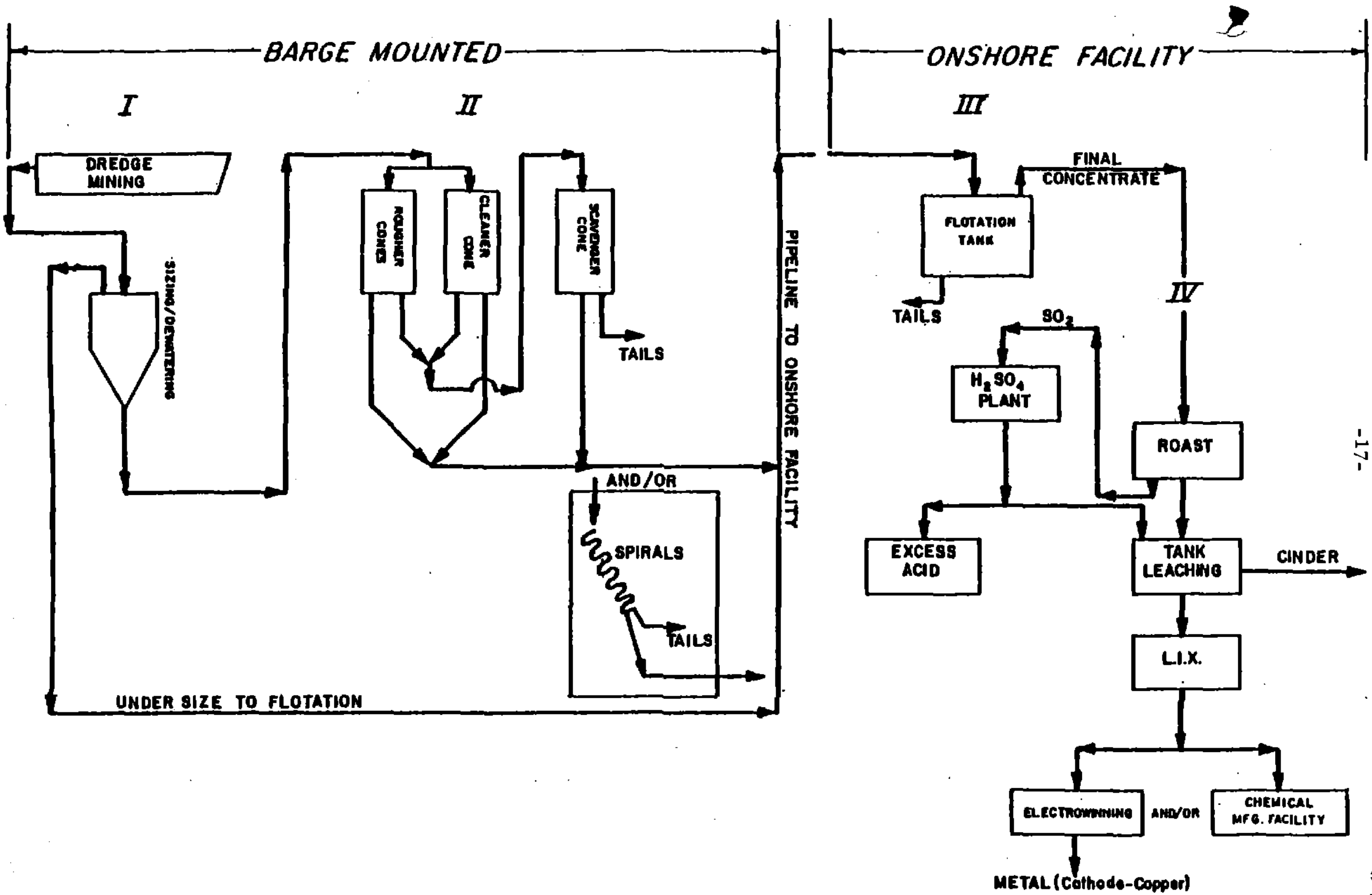

FIGURE 1. PROCESS FLOWSHEET AS PROPOSED BY CITCO

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LITERATURE SURVEY

The choice of a flowsheet for evaluation of the Tasmanian cobalt ore was guided by the following:

1. The combination of unit processes to be used are technically sound and commercially proven.

2. The process would be capable of producing elemental sulfur in addition to production of copper, cobalt and other metals and not SO_2 gas or sulfate. Since there is no market for sulfuric acid in Australia, the SO_2 must be disposed of in some other way, creating a problem with air pollution. SHOULD READ "ONLY A REMOTE MARKET"

A quick literature survey(20-41) on the state of the art of sulfide hydrometallurgy suggested a pressure hydrometallurgical approach in treating the Tasmanian sulfide concentrates for recovering copper and cobalt besides sulfur.

The advantages of a pressure hydrometallurgical process are:

- It is possible to treat ores or concentrates with complex metal content such as solid solution of cobalt in pyrite.

- Elimination of the problem of air pollution by sulfur dioxide or elimination of the necessity of producing unwanted sulfuric acid.

- Pressure hydrometallurgy lends itself to automation, can be centrally controlled, and thus, labor requirements can be kept to a minimum.

The disadvantages associated with the pressure hydrometallurgical process are that it is quite energy intensive and requires a much higher capital investment that other methods.

For our case, the pyrite is subjected to thermal decomposition to drive off the labile sulfur in elemental form. This eliminates about half the sulfur normally contained in pyrite. The resulting artificial pyrrhotite (FeS) is then pressure leached in dilute sulfuric acid, which converts sulfide sulfur to its elemental form. The iron is removed by hydrolysis and filtration, and the copper and cobalt are recovered via the solvent extraction and electrowinning route.

There are other methods of producing elemental sulfur, but none of these processes are beyond the laboratory stages, and as yet are not commercially proven.

CONSULTATION WITH EXPERTS

In order to assess the potential treatment process and arrive at a more realistic estimation of the capital and operating costs for DCF ROI analysis, a number of experts were consulted. A summary of their remarks and comments are included below.

Dan H. Kentro (Consultant, Molycorp)

- The examination work done under the auspices of Cities Service appears adequate enough to permit a preliminary evaluation as to the economic feasibility of the project, before beginning more intensive (and expensive) work to demonstrate the practicality of a process to treat these tailings to produce marketable products.

- Assuming a 25,000 ton per day dredging operation (9 million tons per year) to reclaim the tailings for processing and a flotation concentrator to produce a pyrite concentrate of 90% pyrite content (48% sulfur), with respective recoveries of 50%, 75% and 75% for copper, cobalt and pyrite into the concentrate. The result would be the production of 525,000 tons per year concentrate, assaying:

48%	sulfur (252,000 tons/year)
1.4%	copper (7,350 tons/year)
0.10%	cobalt (525 tons/year)

- Operating costs for dredging and concentration (excluding depreciation) are estimated at $2.25 per ton of concentrator feed, or $20,250,000 per year, equivalent to $80+ per ton of recovered contained sulfur. Capital costs for the dredge (30,000 yards per day capacity), flotation concentrator, ancillary facilities would be in the range of $35 to $50 million. With a 10 year life for the operation, depreciation charges would amount to $3.5 to $5.0 million per year, equivalent to $14 to $20 per ton of recovered sulfur contained. A total cost of $94 to $100 per ton of sulfur content in the pyrite concentrate.

- The copper and cobalt content of concentrate is so low that recoveries of 50% or lower can be expected in attempting to process the concentrate chemically to produce saleable copper and cobalt products. At 50% recovery, $0.90 price for copper and $20 price for cobalt, the result would be a sales value of $17,000,000, equivalent to $32+ per ton of concentrate processed. Although there are several chemical process approaches that can be suggested for treatment of the concentrate, it is doubtful that the $17,000,000 sales value is sufficient to cover operating plus depreciation costs, and at the same time, provide sufficient profit to justify the investment in a chemical processing plant.

Cominco

Following a successful pilot campaign utilizing Sherritt Gordon's zinc pressure leaching process of converting zinc sulfide concentrates to zinc sulfate solution and elemental sulfur, Cominco began building the world's first commercial zinc pressure leaching plant. The plant started operating by late 1980 and is continuing operation.

A meeting was arranged with Cominco's Research and Development Division personnel at Trail, British Columbia on June 30, 1981. The meeting was attended by Drs. Hirsch, Fairweather, Swinkels and Parks from Cominco, and Messrs. Leyel, Ackerman and Zaman from Molycorp.

The objective of the meeting was to get acquainted with their pressure leaching process and discuss the practicality of applying a similar process to the Tasmanian cobalt ore. The highlights of the discussion are given as:

- With the metal values in Tasmanian ore, the application of Sherritt's process would be very costly and uneconomical.

- Cominco has quite extensive commercial operating experience on the recovery of metal values by roasting the ore to make a slag and a matte followed by separation of the matte containing the metal values. The matte containing the metal values is then autoclave leached, separated and purified. This process, Cominco suggested, would perhaps be less expensive than Sherritt's. Outokumpu operated a plant to recover metals this way in Kokkola, Finland; however, this plant was shut down in 1975 because of the economics.

- Consensus of Cominco was that processing Tasmanian cobalt ore in any one of the above methods would be a marginal operation at best.

Sherritt Gordon

Following Cominco, a meeting was also arranged with the personnel from Sherritt Gordon at Fort Saskatchewan, Alberta on July 7, 1981.

The response from Sherritt Gordon on the feasibility of treating the Tasmanian cobalt ore was quite similar to that of Cominco. In fact, consensus was that Sherritt Gordon would not recommend pursuing the project further because of the poor grade of the ore with respect to copper and cobalt.

PROPOSED FLOWSHEET

The proposed flowsheet shown in Figure 2 covers the process from dredging through the production of copper, cobalt and sulfur. The recovery of zinc, silver, molybdenum or other materials has not been considered because of the presence at a low concentration in the ore. The treatment of tailings has not been addressed. It could be a potential problem, but without further knowledge of the requirements, impoundment was specified. No cost was generated for an impoundment.

Dredging

The material deposited in the delta is fairly unconsolidated and relatively clean, such that a barge mounted suction dredge could be used. The design criteria used in estimating the dredge costs were: 25,000 tons/day, 350 days/year. The maximum depth was limited to 32 meters, and the pipeline to the processing facility onshore was estimated to run a maximum of 2 km with 1.5 km floating. Ellicott furnished capital and operating estimates for the dredge, pipeline and support facilities.

Flotation

For reasons discussed earlier (p. 15), flotation was chosen over cones or spirals for the gravity separation. Although the cones or spirals would recover the coarse sulfides, the finer fraction of sulfide will be a total loss for such gravity operation. Further, there seems to be no combination of methods that would have an advantage over flotation. The flotation cells and equipment were designed to handle 25,000 per day feed, producing 1,600 tons per day through two stages of flotation.

The first stage unit would have a flotation time of 10 minutes and would produce 2,500 tons per day. This concentrate would be reground to -200 mesh. The second stage unit would have a flotation time of about 6 minutes. Anticipated recovery from this section is:

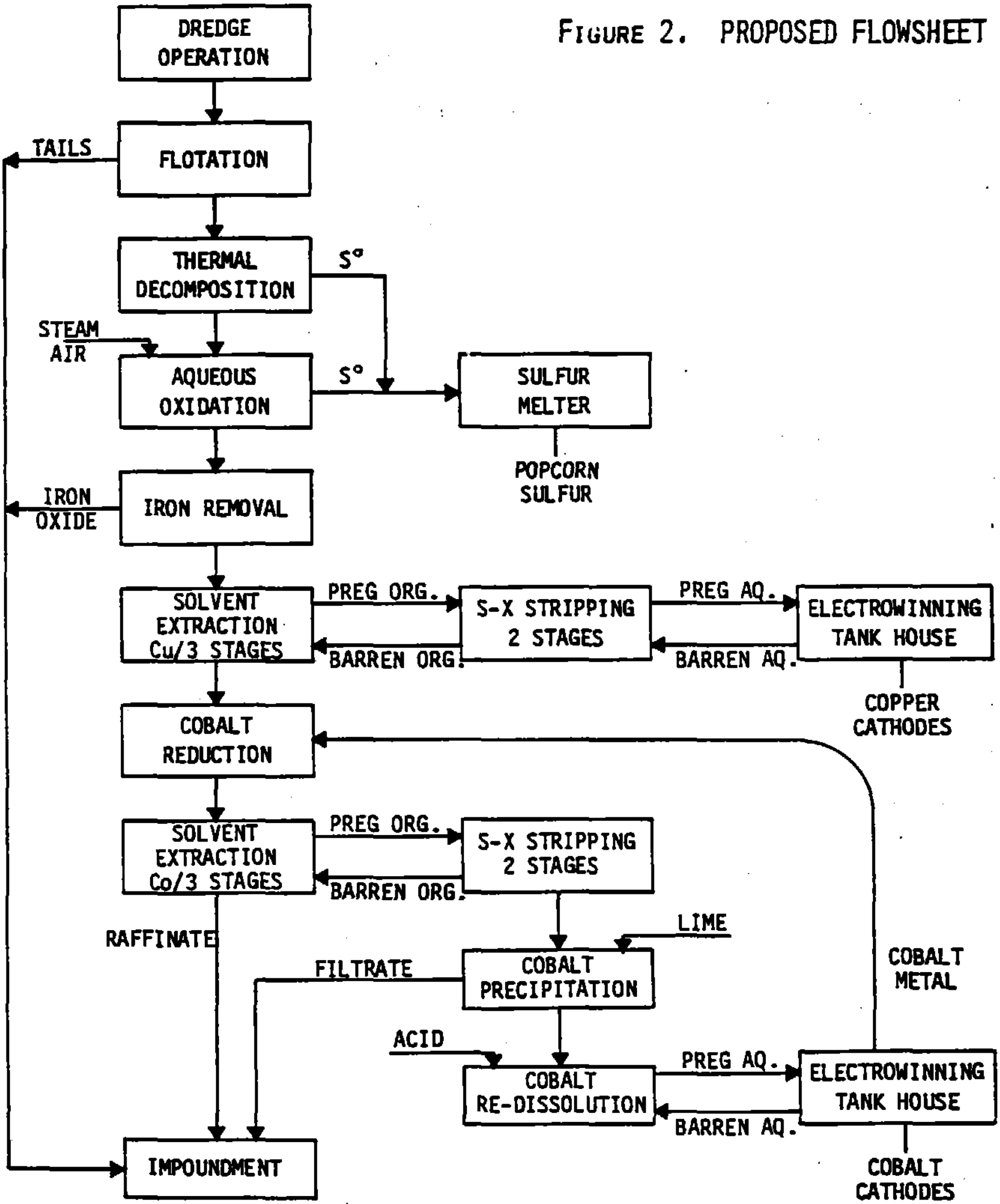
S°	80%
Cu	60%
Co	75%

Estimates were obtained for the flotation system from the Denver Equipment Company and the Galigher Company.

Sulfur Production Alternatives

The concentrate from the flotation section will consist almost entirely of pyrite, which is 53% sulfur. To solubilize the contained metals, the sulfide mineral must be converted to an oxide ore. Considering limitation to conventional technology, the alternatives are to roast the ore, driving the sulfur off as SO_2 or to produce elemental sulfur by thermal decomposition of pyrite to pyrrhotite and aqueous oxidation of the pyrrhotite to iron oxide. The roasting alternative is more widely used and is probably cheaper to build and operate; however, the problem of disposing of the SO_2 is enormous. Conversion of SO_2 to sulfuric acid would make 2,700 tons of acid a day, which presents a shipping problem. The SO_2 could be scrubbed out of the system with lime, making gypsum, but there is no ESTABLISHED convenient source of limestone. There are several processes that convert SO_2 to

FIGURE 2. PROPOSED FLOWSHEET



elemental sulfur, but they all require a quantity of fuel to reduce the sulfur, and fuel is at a premium in Tasmania. In short, there seems to be no way to economically dispose of the sulfur dioxide.

The thermal decomposition/aqueous oxidation route of Sherritt Gordon produces elemental sulfur directly from the sulfide ore. The low temperature thermal decomposition of the pyrite removes the labile sulfur atom to form pyrrhotite, which is then oxidized with air in an aqueous suspension at 300° F and 1000 psig. The leach liquor is separated from the tails, and the iron is hydrolyzed and filtered off.

Capital and operating expenses for the thermal decomposition and aqueous oxidation section were obtained from a report by Sherritt Gordon (see Appendix E).

Copper Purification

The solution from the aqueous oxidation section contains about 1.0 grams per liter copper. Studies of various operating and solvent extraction plants indidate that 20:1 concentration is routine for an SX circuit. Three stages of extraction and two stages of stripping were designed with settling areas of 1.75 ft^2/gpm for both stripping and extraction. Costs were obtained from a number of sources, detailed in Appendix C. The 20 grams per liter pregnant aqueous reports to the electrowinning tankhouse, where 41,500 lbs/day cathode copper is produced. Electrowinning is a common method of producing high quality copper. Cost estimates were obtained from General Mills Chemical Company and from a report by McKee for California Nickel.

Cobalt Purification

The raffinate from the copper SX circuit reports to the cobalt reduction tower, where the cobaltic ion is reduced to cobaltous ion by contact with cobalt metal. This step is essential to the proper functioning of the SX circuit. Information on cobalt SX circuits is somewhat less available. A 20:1 concentration factor was also assumed for this case, with three stage extraction and two stage stripping.

The pregnant aqueous stream coming off the cobalt SX circuit must be concentrated further for efficient operation of an electrowinning operation. The concentration is accomplished by precipitating cobalt with lime and redissolving to a concentration of 30 grams per liter before going to the tankhouse. The flow going into the tankhouse is nearly of a pilot scale. A portion of the cathode cobalt produced will be recycled to the cobalt reduction tower.

Waste streams are generated at four points in the flow diagram. First, nearly all of the material dredged will be rejected from the flotation section. Second, approximately 47% of the weight of the remaining pyrite will be removed by the iron hydrolysis and filtration. Third, the remainder of raffinate from the cobalt and copper SX circuits will go to the tails, and last, the filtrate from the cobalt precipitation unit will be shipped to the tails. All streams are assumed to be impounded, although further processing on some may be desirable or necessary.

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REFERENCES

1. The King River Delta Mineral Deposit: Preliminary Report by Terra Marine International, Denver, Colorado, p. 2.

2. Ibid, p. 3.

3. Ibid, p. 4.

4. Cobalt Deposit, King River Delta, Macquarie Harbour, Tasmania, memo to H.E. Lindsey, W.R. Moran, from R.E. Besley, Union Oil Development Corporation, dated January 16, 1980.

5. Ibid, Appendix 4, p. 21.

6. Tasmania Pyrite Project: CITCO Final Report, June 1977, p. 19.

7. Macquarie Harbour Sulphides: Mineral Processing Investigations by R. Buller et al, Robertson Research (Australia) Pty. Ltd., Report No. 423, Project No. 756/9511, July 14, 1976, p. 8.

8. Microscopic Examination of a Copper Concentrate, Report No. 2, Lakefield Research of Canada, April 9, 1975, p.3.

9. Reference 7, Appendix 4, p. 21.

10. Lamflo Sluice and Spiral Beneficiation, The Australian Mineral Development Laboratories (AMDEL), March 29, 1976.

11. Reference 7.

12. Cone Concentrator Stuides and Circuit Designs on Alluvial Sulphide Ore by M.S. Cross, Mineral Deposits Ltd., Research and Engineering Department, Report No. 15.213.1/1, April 29, 1976.

13. Spiral Concentrator Studies and Circuit Designs on Alluvial Sulphide Ore by M.S. Cross, Mineral Deposits Ltd., Research and Engineering Department, Report No. 15.213.1/2.

14. An Investigation of the Recovery of Sulphides: Progress Reports No. 1-4, Lakefield Research of Canada, May 12, 1976.

15. Reference 7, Table 9 (attached).

16. Reference 11, p.13.

17. Reference 12, p. 2.

18. Letter Report on Preliminary Studies with a Tasmanian Tailings Sample, Colorado School of Mines Research Institute, July 16, 1981, pp. 1-2.

19. Ibid, Appendix.

20. Review of Developments in Hydrometallurgy, 1979, by Milton E. Wadsworth, Journal of Metals, April 1980, pp. 27-30.

21. Review of Developments in Hydrometallurgy, 1980, by Milton E. Wadsworth, Journal of Metals, April 1981, pp. 36-40.

22. Direct Pressure Leaching of Zinc Blend with Simultaneous Production of Elemental Sulfur, by H. Veltman and G.L. Bolton, Erzmetall, 1980, v. 33, no. 2, pp. 76-83.

23. Oxidising Pressure Leaching of Roasted Pyrite Concentrate, by M.H. Tikkanen and E. Vuoristo, op.cit. (6), pp. 250-268.

24. Recovery of Elemental Sulphur from Pyrite and Pyrrhotite, by K.W. Downes and R.W. Bruce, CIM Bulletin, v. 48, 1955, pp. 127-132.

25. Iron and Sulphur from Sulphidic Iron Ores, by W. Kunda, B. Rudyk and V.N. Mackiw, CIM Bulletin, v. 61, 1968, pp. 819-835.

26. Economics of Iron and Sulphur Recovery from Pyrites, by B.R. Mehta and P.T. O'Kane, CIM Bulletin, v. 61, 1968, pp. 836-846.

27. Reducing Pressure Leaching of an Ilmenite Concentrate, by M.H. Tikkanen, T. Tyynela and E. Vuoristo, op.cit. (6), pp. 269-283; Finnish Pat. 32,894, 1963.

28. Direct Leaching Zinc Sulphide Concentrates by Sherritt Gordon, by F.A. Forward and H. Veltman, Journal of Metals, 12, 1959, pp. 836-840.

29. Recovery of Zinc from Zinc Sulphides by Direct Pressure Leaching, P. Kawulka, W.J. Haffenden and V.N. Mackiw, U.S. Patent 3,867,268, Fevruary 18, 1975.

30. Leaching of Metal Sulphides, H.E. Hirsch, J.F. Higginson, E.G. Parker and G.M. Swinkels, U.S. Patent No. 4,192,851.

31. Pilot Plant Demonstration of Zinc Sulphide Pressure Leaching, by E.G. Parker and S. Romanchuk, Proceedings, World Symposium on Metallurgy, TMS-AIME, 109th AIME Annual Meeting, Las Vegas, February 1980, pp. 407-425.

32. Two-Stage Pressure Leaching Process for Zinc and Iron Bearing Mineral Sulphides, H. Veltman, G.J.J. Mould and P. Kawulka, U.S. Patent 4,004,991, January 25, 1977.

33. Acid Pressure Leaching Zinc Concentrates Produces By-Product Elemental Sulphur, by B.N. Doyle, I.M. Masters, I.C. Webster and H. Veltman, presented at Eleventh Commonwealth Mining and Metallurgical Congress, Hong Kong, May 6-12, 1978.

34. Pressure Leaching Process for Complex Zinc-Lead Concentrates, by G.L. Bolton, N. Zubryckyj and H. Veltman, presented at Thirteenth International Mineral Processing Congress, Warsaw, Poland, June 1979.

35. Treatment of Copper-Zinc Concentrates by Pressure Hydrometallurgy, by D.J.I. Evans, S. Romanchuk and V.N. Mackiw, CIM Bulletin, v. 57, no. 628, 1964, pp. 857-866.

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36. Direct Acid Pressure Leaching of Chalcocite Concentrate, by H. Veltman, S. Pellegrini and V.N. Mackiw, J. Metals, v. 19, no. 2, 1967, pp. 21-25.

37. Copper and Elemental Sulphur from Chalcopyrite by Pressure Leaching, by A.I. Vizsolyi, et al, J. Metals, v. 19, no. 11, 1967, pp. 52-59.

38. Oxidation Leaching of Copper Sulphide in Acidic Pulps at Elevated Temperature and Pressure, by M.H. Stanczyk and C. Rampacek, Washington, U.S. Bureau of Mines, Rep. Invest. 6193, 1966.

39. Research at Warren Spring Laboratory --- Pressure Hydrometallurgy, by R. Derry, Chemy. Ind., December 6, 1969, p. 1759.

40. Hydrometallurgical Treatment of Oxidised Nickel-Cobalt Concentrate, by D. Maschmeyer and B. Benson, CIM Bulletin, v. 58, 1965, pp. 931-938.

41. Reduction Roasting and Acid Pressure Leaching of Calcined Cobalt Concentrates, by H. Veltman, D.J.I. Evans and V.N. Mackiw, CIM Bulletin, v. 57, 1964, pp. 1281-1287.

APPENDIX A

LIST OF SOURCES FOR REVIEW OF PREVIOUS WORK

APPENDIX A

Source Material for Review of Past Work

Tasmania Tailings

1. The King River Delta Mineral Deposit: Preliminary Report by Terra Marine International, Denver, Colorado.

2. Core Sample Assay: Cities Service Drilling Program (see Appendix B)

3. McQuarie Harbour Sulphides: Mineral Processing Investigations: by R. Buller et al., Robertson Research (Australia) Pty. Ltd., Report No. 423, Project No. 756/9511, July 14, 1976.

4. Cone Concentrator Studies and Circuit Designs on Alluvial Sulphide Ore, by M.S. Cross, Mineral Deposits Limited, Research and Engineering Department, Report No. 15.213.1/1, April 29, 1976.

5. Spiral Concentrator Studies and Circuit Designs on Alluvial Sulphide Ore, by M.S. Cross, Mineral Deposits Limited, Research and Engineering Department, Report No. 15.213.1/2.

6. Tasmania Pyrite Project: Citco International Minerals Company, June 1977.

7. An Investigation of the Recovery of Sulphides: Progress Reports No. 1-4, Lakefield Research of Canada, May 12, 1976.

8. Lamflo Sluice and Spiral Beneficiation: The Australian Mineral Development Laboratories (AMDEL), March 29, 1976.

9. McQuarie Harbour - Cobalt: Memo to John Ackerman et al by R.E. Beasley, dated March 12, 1980. Attached to this memo is a write-up by Mr. Harvey of Terra Marine Pacific, Inc. about Tasmanian tailings deposit.

10. McQuarie Harbour Cobalt: Australia Sulfur Industry: Memo to John Ackerman by R.E. Beasley dated December 16, 1980.

11. Mineral Deposits, King River Delta, Tasmania, Australia, Memo to W.R. Moran from P.C. Carlos dated February 29, 1980.

12. Letter to John Ackerman from S.R.M. Harvey of Aberdare, Inc. dated June 2, 1981.

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APPENDIX B

CHEMICAL ANALYSIS OF CORE SAMPLES

and

CALCULATION OF TOTAL ORE AND THE MINERAL CONTENT
(113 Cores)

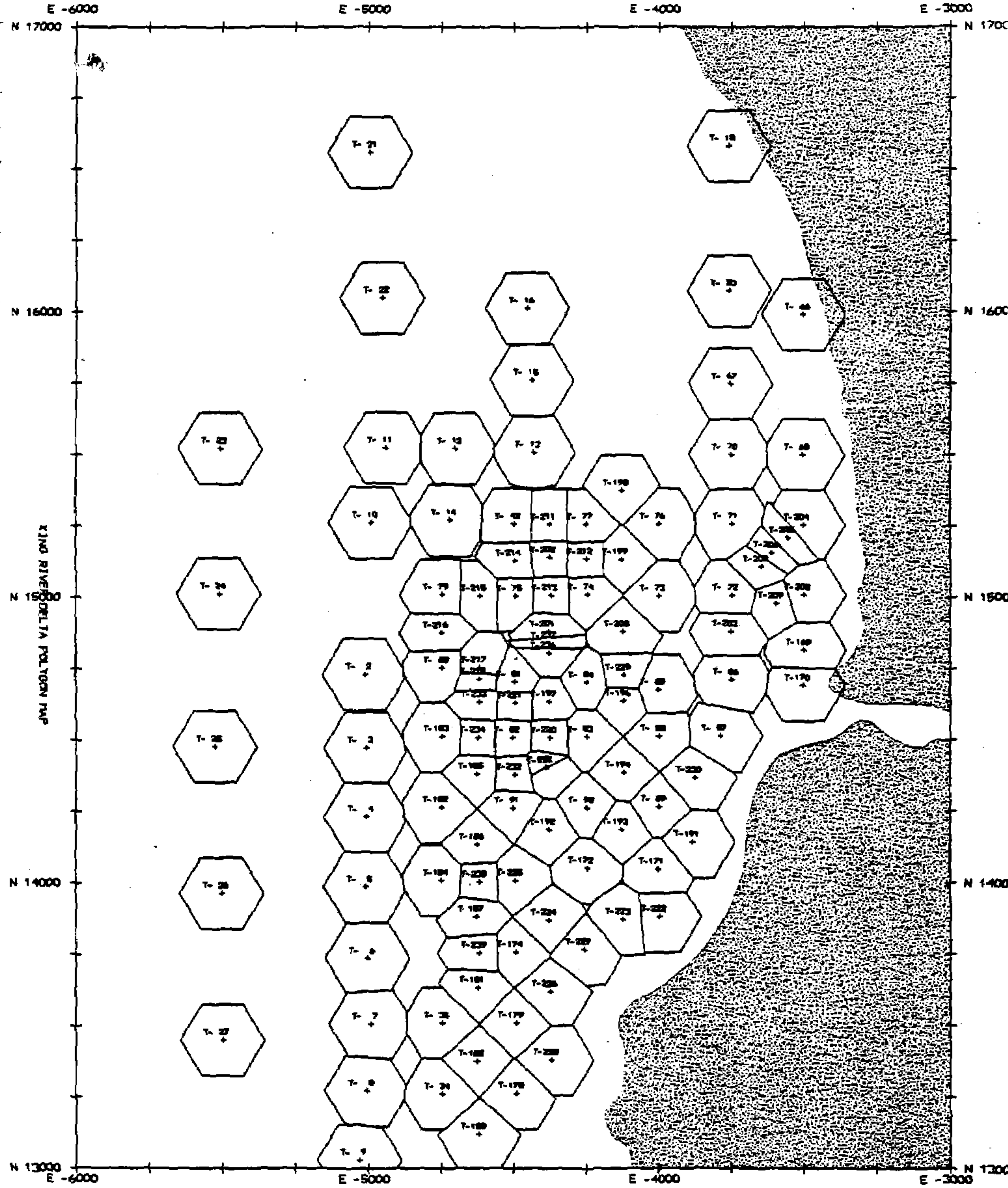

FIGURE B1 POLYGON PLOT (250 m area of influence)

TABLE B-1

COORDINATES - CORE ANALYSIS

HOLE	NORTH M	EAST M	THICK M	CU PPM	CO PPM	ZN PPM	AG PPM	MO PPM
T- 2	14729.	5014.	12.000	920.000	97.000	162.000	0.500	52.500
T- 3	14471.	5008.	9.000	867.000	113.000	157.000	0.000	67.000
T- 4	14230.	5007.	9.000	747.000	113.000	157.000	0.000	67.000
T- 5	13985.	5009.	9.000	1077.000	100.000	190.000	0.000	57.000
T- 6	13735.	5004.	7.000	923.000	112.000	193.000	12.000	40.000
T- 7	13506.	4992.	3.000	680.000	130.000	140.000	0.000	50.000
T- 8	13272.	5006.	3.000	670.000	100.000	180.000	2.000	45.000
T- 9	13028.	5031.	3.000	750.000	125.000	200.000	2.000	35.000
T- 10	15260.	4992.	6.000	872.000	129.000	155.000	5.000	37.000
T- 11	15523.	4940.	3.000	830.000	105.000	170.000	2.000	45.000
T- 12	15521.	4702.	6.000	640.000	151.000	190.000	2.000	47.000
T- 13	15509.	4432.	3.000	823.000	132.000	155.000	2.000	45.000
T- 14	15269.	4721.	9.000	823.000	131.000	157.000	0.600	35.000
T- 15	15761.	4438.	3.000	760.000	30.000	170.000	2.000	45.000
T- 16	16012.	4454.	4.000	790.000	95.000	165.000	2.000	42.000
T- 18	16580.	3760.	3.000	770.000	120.000	170.000	0.000	45.000
T- 20	16072.	3759.	7.000	728.000	94.000	183.000	0.000	38.000
T- 21	16558.	4995.	3.000	800.000	95.000	170.000	2.000	35.000
T- 22	16048.	4954.	3.000	760.000	110.000	160.000	2.000	45.000
T- 23	15522.	5507.	4.000	770.000	113.000	175.000	1.000	45.000
T- 24	15013.	5513.	4.000	770.000	109.000	160.000	2.000	45.000
T- 25	14477.	5525.	3.000	683.000	102.000	145.000	1.000	47.000
T- 26	13964.	5501.	3.000	780.000	120.000	150.000	2.000	50.000
T- 27	13449.	5500.	3.000	740.000	94.000	170.000	2.000	60.000
T- 34	13260.	4746.	3.000	870.000	110.000	160.000	0.000	75.000
T- 35	13509.	4746.	3.000	850.000	110.000	150.000	0.000	70.000
T- 42	15257.	4500.	10.000	1033.000	103.000	197.000	0.000	45.000
T- 66	15989.	3503.	3.000	1200.000	100.000	190.000	0.000	55.000
T- 67	15747.	3754.	14.000	1306.000	70.000	252.000	0.000	45.000
T- 68	15498.	3505.	3.000	820.000	30.000	110.000	0.000	20.000
T- 70	15499.	3752.	3.000	1050.000	50.000	380.000	0.000	30.000
T- 71	15257.	3749.	3.000	930.000	50.000	480.000	0.000	30.000
T- 72	15007.	3750.	3.000	1250.000	40.000	450.000	0.000	30.000
T- 73	15004.	4003.	3.000	1850.000	50.000	110.000	0.000	55.000
T- 74	15009.	4248.	14.000	1376.000	54.000	186.000	0.000	47.000
T- 75	15005.	4496.	15.000	1660.000	73.000	148.000	1.200	47.000
T- 76	15256.	4001.	6.000	1850.000	85.000	265.000	1.500	53.000
T- 77	15255.	4252.	21.000	1414.000	100.000	160.000	1.500	51.000
T- 79	15009.	4751.	32.000	1286.000	89.000	163.000	1.200	53.000
T- 80	14753.	4749.	29.000	1135.000	82.000	140.000	1.100	32.000
T- 81	14705.	4497.	21.000	1443.000	80.000	153.000	1.100	49.000
T- 82	14509.	4505.	14.000	1430.000	80.000	157.000	1.100	48.000
T- 83	14512.	4250.	7.500	1433.000	87.000	187.000	1.400	47.000
T- 84	14702.	4250.	9.000	1400.000	61.000	163.000	1.300	35.000
T- 85	14675.	4002.	6.000	1300.000	60.000	130.000	1.000	30.000
T- 86	14712.	3750.	3.000	2100.000	56.000	230.000	1.800	42.000
T- 87	14514.	3788.	3.000	1450.000	50.000	420.000	1.200	32.000

TABLE B-1 (cont'd)

HOLE	NORTH M	EAST M	THICK M	CU PPM	CO PPM	ZN PPM	AG PPM	MO PPM
T- 88	14513.	4000.	3.000	1850.000	56.000	260.000	1.800	28.000
T- 89	14266.	3999.	6.000	1295.000	97.000	275.000	1.400	20.000
T- 90	14263.	4248.	6.000	1525.000	81.000	210.000	1.600	32.000
T- 91	14263.	4502.	14.000	1290.000	128.000	130.000	1.500	42.000
T-169	14815.	3500.	3.000	1850.000	56.000	260.000	18.000	28.000
T-170	14691.	3503.	4.000	1125.000	83.000	377.000	1.000	5.000
T-171	14047.	4003.	3.000	1050.000	32.000	120.000	0.600	2.000
T-172	14052.	4248.	6.000	1375.000	34.000	290.000	1.400	19.000
T-174	13758.	4494.	9.000	1083.000	70.000	227.000	1.100	67.000
T-178	13262.	4490.	3.000	850.000	84.000	160.000	1.200	66.000
T-179	13510.	4491.	3.000	900.000	86.000	280.000	1.000	62.000
T-181	13633.	4624.	9.000	1083.000	70.000	187.000	1.100	55.000
T-182	14264.	4750.	21.000	1050.000	85.000	180.000	2.300	54.000
T-183	14512.	4749.	10.000	1250.000	77.000	129.000	1.100	53.000
T-184	14009.	4750.	6.000	1375.000	62.000	190.000	1.400	45.000
T-185	14383.	4628.	15.000	1180.000	81.000	139.000	1.400	47.000
T-186	14135.	4626.	10.000	1500.000	61.000	149.000	1.200	45.000
T-187	13882.	4630.	18.000	1383.000	64.000	150.000	1.100	48.000
T-188	13375.	4626.	3.000	750.000	44.000	250.000	1.200	36.000
T-189	13119.	4619.	9.000	700.000	40.000	140.000	1.200	36.000
T-191	14143.	3883.	18.000	1467.000	99.000	173.000	1.500	47.000
T-192	14187.	4379.	9.000	1633.000	105.000	160.000	1.700	52.000
T-193	14185.	4129.	3.000	1850.000	150.000	220.000	2.200	48.000
T-194	14386.	4122.	3.000	1950.000	50.000	230.000	1.200	42.000
T-195	14405.	4391.	12.000	1400.000	80.000	205.000	1.400	56.000
T-196	14636.	4123.	6.000	1725.000	142.000	225.000	1.600	39.000
T-197	14635.	4379.	12.000	1462.000	67.000	220.000	1.300	60.000
T-198	15372.	4129.	19.000	1497.000	117.000	164.000	1.700	55.000
T-199	15131.	4130.	10.000	1600.000	68.000	167.000	1.300	56.000
T-200	14878.	4126.	6.000	1900.000	76.000	200.000	1.300	52.000
T-201	14880.	4382.	18.000	1608.000	93.000	172.000	1.300	48.000
T-202	15140.	4378.	21.000	1371.000	121.000	166.000	1.400	59.000
T-203	14880.	3752.	3.000	1550.000	116.000	150.000	1.400	44.000
T-204	15252.	3504.	3.000	1950.000	92.000	330.000	1.800	38.000
T-205	15208.	3556.	3.000	1000.000	36.000	230.000	1.000	13.000
T-206	15157.	3610.	3.000	2150.000	70.000	200.000	1.200	46.000
T-207	15106.	3646.	3.000	1900.000	88.000	170.000	1.200	50.000
T-208	15008.	3502.	6.000	2100.000	157.000	550.000	1.800	48.000
T-209	14979.	3597.	3.000	2400.000	82.000	290.000	1.200	66.000
T-211	15255.	4378.	24.000	1420.000	90.000	154.000	1.000	38.000
T-212	15134.	4251.	24.000	1294.000	86.000	147.000	1.300	38.000
T-213	15007.	4375.	24.000	1425.000	98.000	137.000	2.600	39.000
T-214	15129.	4498.	21.000	1536.000	102.000	123.000	5.500	33.000
T-215	15005.	4620.	30.000	1465.000	72.000	126.000	1.100	33.000
T-216	14875.	4752.	27.000	1328.000	78.000	111.000	1.400	35.000
T-217	14755.	4619.	12.000	1625.000	87.000	122.000	1.300	36.000
T-218	14714.	4621.	12.000	1562.000	75.000	167.000	1.500	40.000

HOLE	NORTH M	EAST M	THICK M	CU PPM	CO PPM	ZN PPM	AG PPM	MO PPM
T-220	14509.	4377.	12.000	1800.000	95.000	197.000	1.300	71.000
T-221	14631.	4496.	21.000	1407.000	81.000	133.000	1.100	43.000
T-222	13882.	3999.	3.000	800.000	38.000	100.000	0.800	17.000
T-223	13872.	4124.	3.000	1400.000	50.000	120.000	1.000	22.000
T-224	13870.	4380.	9.000	1900.000	67.000	167.000	1.700	42.000
T-225	14009.	4493.	3.000	1650.000	82.000	150.000	1.000	60.000
T-226	13621.	4375.	6.000	1425.000	41.000	225.000	1.200	21.000
T-227	13767.	4259.	6.000	1825.000	83.000	170.000	1.700	45.000
T-228	13381.	4370.	6.000	1125.000	102.000	140.000	1.100	62.000
T-229	14726.	4122.	6.000	1450.000	39.000	185.000	1.000	30.000
T-230	14369.	3876.	3.000	2000.000	78.000	400.000	1.200	56.000
T-231	14403.	4390.	12.000	1462.000	71.000	160.000	1.200	50.000
T-232	14380.	4497.	18.000	1392.000	81.000	140.000	1.200	48.000
T-233	14634.	4619.	24.000	1369.000	86.000	106.000	1.300	35.000
T-234	14509.	4623.	24.000	1187.000	84.000	121.000	1.000	34.000
T-236	14805.	4379.	18.000	1250.000	77.000	146.000	1.000	35.000
T-237	14840.	4379.	19.000	1375.000	78.000	145.000	0.800	37.000
T-238	14004.	4617.	18.000	1242.000	91.000	134.000	1.000	42.000
T-239	13755.	4619.	15.000	1190.000	103.000	146.000	0.900	43.000

*

TABLE B-2

CALCULATED AREA - RESERVES
(250 m area of influence)

HOLE	AREA SQ-M	THICK M	COPPER LBS	COBALT LBS	ZINC LBS	SILVER LBS	MOLY LBS
T- 2	53922.1	12.0	3363445.	354624.	592259.	1828.	191936.
T- 3	53214.8	9.0	2346078.	305775.	424838.	0.	181300.
T- 4	52765.0	9.0	2004276.	303190.	421247.	0.	179768.
T- 5	53379.6	9.0	2923358.	271435.	515727.	0.	154718.
T- 6	52537.9	7.0	1917878.	232722.	401030.	24934.	83115.
T- 7	50660.5	3.0	583913.	111630.	120217.	0.	42935.
T- 8	52212.9	3.0	592956.	88501.	159302.	1770.	39825.
T- 9	53696.3	3.0	682614.	113769.	182030.	1820.	31855.
T- 10	53976.0	6.0	1595574.	236042.	283617.	9149.	67702.
T- 11	52995.1	3.0	745562.	94318.	152705.	1797.	40422.
T- 12	52775.9	6.0	1145026.	270155.	339930.	3578.	84088.
T- 13	53629.7	3.0	748126.	119991.	140899.	1818.	40906.
T- 14	51923.0	9.0	2172954.	345877.	414525.	1584.	95050.
T- 15	54013.1	3.0	695797.	27466.	155639.	1831.	41198.
T- 16	54022.2	4.0	964512.	115986.	201449.	2442.	51278.
T- 18	54127.5	3.0	706445.	110095.	155968.	0.	41286.
T- 20	54106.0	7.0	1557842.	201150.	391600.	0.	81316.
T- 21	54127.5	3.0	733969.	87159.	155968.	1835.	32111.
T- 22	54127.5	3.0	697270.	100921.	146794.	1835.	41286.
T- 23	54127.5	4.0	941927.	138231.	214074.	1223.	55048.
T- 24	54127.5	4.0	941927.	133338.	195725.	2447.	55048.
T- 25	54127.5	3.0	626626.	93581.	133032.	917.	43121.
T- 26	54127.5	3.0	715620.	110095.	137619.	1835.	45873.
T- 27	54127.5	3.0	678921.	86241.	155968.	1835.	55048.
T- 34	42462.2	3.0	626169.	79171.	115157.	0.	53980.
T- 35	41902.6	3.0	603712.	78127.	106537.	0.	49717.
T- 42	31222.3	10.0	1822274.	181698.	347520.	0.	79383.
T- 66	54106.0	3.0	1100516.	91710.	174248.	0.	50440.
T- 67	53983.0	14.0	5576692.	298904.	1076054.	0.	192152.
T- 68	53083.5	3.0	737808.	26993.	98974.	0.	17995.
T- 70	52646.8	3.0	936981.	44618.	339098.	0.	26771.
T- 71	45035.1	3.0	709911.	38167.	366406.	0.	22900.
T- 72	30355.6	3.0	643159.	20581.	231537.	0.	15436.
T- 73	42311.5	3.0	1326783.	35859.	78890.	0.	39445.
T- 74	23940.1	14.0	2605679.	102258.	352221.	0.	89002.
T- 75	20540.4	15.0	2889726.	127078.	257638.	2089.	81818.
T- 76	42015.2	6.0	2634983.	121067.	377444.	2136.	75489.
T- 77	24524.1	21.0	4114435.	290978.	465566.	4365.	148399.
T- 79	34631.3	32.0	8052082.	557259.	1020598.	7514.	331851.
T- 80	30275.9	29.0	5630402.	406778.	694499.	5457.	158743.
T- 81	16351.9	21.0	2799641.	155212.	296843.	2134.	95067.
T- 82	15318.5	14.0	1732721.	96935.	190236.	1333.	58161.
T- 83	27396.8	7.5	1663626.	101002.	217096.	1625.	54564.
T- 84	23721.5	9.0	1688734.	73581.	196617.	1568.	42218.
T- 85	32704.6	6.0	1441292.	66521.	144129.	1109.	33261.
T- 86	42413.1	3.0	1509694.	40259.	165347.	1294.	30194.
T- 87	42217.1	3.0	1037591.	35779.	300544.	859.	22899.

TABLE B-2 (cont'd)

HOLE	AREA SQ-M	THICK M	COPPER LBS	COBALT LBS	ZINC LBS	SILVER LBS	MOLY LBS
T- 88	31454.8	3.0	986344.	29857.	138621.	960.	14928.
T- 89	26828.3	6.0	1177776.	88220.	250107.	1273.	18190.
T- 90	27555.3	6.0	1424540.	75664.	196166.	1495.	29892.
T- 91	24448.6	14.0	2494711.	247537.	251405.	2901.	81223.
T-169	36111.1	3.0	1132354.	34277.	159142.	11017.	17138.
T-170	42041.8	4.0	1068913.	78862.	358205.	950.	4751.
T-171	32656.5	3.0	581204.	17713.	66423.	332.	1107.
T-172	38673.6	6.0	1802673.	44575.	380200.	1835.	24910.
T-174	26857.6	9.0	1479063.	95600.	310016.	1502.	91503.
T-178	37420.3	3.0	539133.	53279.	101484.	761.	41862.
T-179	31375.1	3.0	478627.	45735.	148906.	532.	32972.
T-181	29363.1	9.0	1617042.	104518.	279212.	1642.	82121.
T-182	41911.1	21.0	5221390.	422684.	895095.	11437.	268529.
T-183	38222.0	10.0	2699429.	166285.	278581.	2375.	114456.
T-184	38699.0	6.0	1803857.	81338.	249260.	1837.	59035.
T-185	23402.6	15.0	2340377.	160653.	275688.	2777.	93218.
T-186	28259.6	10.0	2395001.	97397.	237903.	1916.	71850.
T-187	26151.2	18.0	3678195.	170213.	398937.	2926.	127660.
T-188	31765.1	3.0	403814.	23690.	134605.	646.	19383.
T-189	45418.6	9.0	1616675.	92381.	323335.	2771.	83143.
T-191	39104.6	18.0	5834168.	393717.	688010.	5965.	186916.
T-192	29264.0	9.0	2430026.	156248.	238092.	2530.	77380.
T-193	26572.9	3.0	833260.	67562.	99090.	991.	21620.
T-194	30375.6	3.0	1003990.	25743.	118419.	618.	21624.
T-195	6483.5	12.0	615414.	35167.	90114.	615.	24617.
T-196	19400.2	6.0	1134475.	93389.	147975.	1052.	25649.
T-197	18268.6	12.0	1810849.	82987.	272494.	1610.	74317.
T-198	41238.2	19.0	6627100.	517950.	726015.	7526.	243481.
T-199	27030.1	10.0	2443521.	103850.	255043.	1985.	85523.
T-200	33691.9	6.0	2170095.	86804.	228431.	1485.	59392.
T-201	17260.7	18.0	2822704.	163253.	301931.	2282.	84260.
T-202	15299.2	21.0	2488708.	219645.	301332.	2541.	107100.
T-203	33363.7	3.0	876548.	65600.	84827.	792.	24883.
T-204	35729.7	3.0	1180956.	55717.	199854.	1090.	23013.
T-205	16611.4	3.0	281563.	10136.	64760.	282.	3660.
T-206	12297.9	3.0	448166.	14591.	41690.	250.	9589.
T-207	16553.8	3.0	533115.	24692.	47700.	337.	14029.
T-208	33239.1	6.0	2366292.	176908.	619743.	2028.	54087.
T-209	21216.3	3.0	863079.	29489.	104289.	432.	23735.
T-211	22478.2	24.0	4328222.	274324.	469399.	3048.	115826.
T-212	15281.3	24.0	2681355.	178204.	304605.	2694.	78741.
T-213	16148.7	24.0	3120413.	214597.	299998.	5693.	85401.
T-214	21194.6	21.0	3862640.	256503.	309313.	13831.	82986.
T-215	27880.4	30.0	6923191.	340252.	595442.	5198.	155949.
T-216	28385.8	27.0	5750577.	337760.	480658.	6062.	151559.
T-217	19833.6	12.0	2185167.	116990.	164056.	1748.	48410.
T-218	8188.4	12.0	867181.	41638.	92714.	833.	22207.

TABLE B-2 (cont'd)

HOLE	AREA SQ-M	THICK M	COPPER LBS	COBALT LBS	ZINC LBS	SILVER LBS	MOLY LBS
T-220	14765.4	12.0	1801969.	95104.	197216.	1301.	71078.
T-221	11851.3	21.0	1978463.	113899.	187019.	1547.	60465.
T-222	36235.6	3.0	491355.	23339.	61419.	491.	10441.
T-223	33320.6	3.0	790698.	28239.	67774.	565.	12425.
T-224	31914.7	9.0	3083439.	108732.	271018.	2759.	68160.
T-225	31156.8	3.0	871378.	43305.	79216.	528.	31686.
T-226	36591.1	6.0	1767625.	50858.	279099.	1489.	26049.
T-227	35287.4	6.0	2183143.	99288.	203361.	2034.	53831.
T-228	41695.5	6.0	1590162.	144175.	197887.	1555.	87636.
T-229	17784.6	6.0	874202.	23513.	111536.	603.	18087.
T-230	35616.1	3.0	1207386.	47088.	241477.	724.	33807.
T-231	15377.8	12.0	1524303.	74026.	166813.	1251.	52131.
T-232	14781.5	18.0	2092564.	121766.	210459.	1804.	72157.
T-233	16062.1	24.0	2981710.	187310.	230870.	2831.	76231.
T-234	15327.9	24.0	2467136.	174591.	251494.	2078.	70668.
T-236	15522.6	18.0	1973311.	121556.	230483.	1579.	55253.
T-237	9644.7	19.0	1423618.	80758.	150127.	828.	38308.
T-238	16338.9	18.0	2063789.	151212.	222663.	1662.	69790.
T-239	23080.5	15.0	2327726.	201475.	295587.	1760.	84111.
TOTALS	3849372.		216885102.	15227254.	30230169.	248756.	7722255.

TOTAL TONNAGE = 86485167.

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TABLE B-3

CALCULATED AREA - RESERVES
(500 m area of influence)

HOLE	AREA SQ-M	THICK M	COPPER LBS	COBALT LBS	ZINC LBS	SILVER LBS	MOLY LBS
T- 2	28091.4	12.0	1752229.	184746.	308545.	952.	99991.
T- 3	93415.0	9.0	4118382.	536767.	745774.	0.	318260.
T- 4	92805.5	9.0	3525212.	533265.	740908.	0.	316184.
T- 5	92271.8	9.0	5053307.	469202.	891484.	0.	267445.
T- 6	3826.6	7.0	139689.	16950.	29209.	1816.	6054.
T- 7	84041.3	3.0	968660.	185185.	199430.	0.	71225.
T- 8	91237.4	3.0	1036138.	154647.	278365.	3093.	69591.
T- 9	60052.4	3.0	763416.	127236.	203578.	2036.	35626.
T- 10	32645.9	6.0	965039.	142764.	171538.	5533.	40948.
T- 11	30033.0	3.0	422519.	53451.	86540.	1018.	22908.
T- 12	87082.7	6.0	1889346.	445768.	560900.	5904.	138749.
T- 13	77760.0	3.0	1084740.	173980.	204295.	2636.	59311.
T- 14	61331.9	9.0	2566712.	408553.	489640.	1871.	112274.
T- 15	17314.5	3.0	223045.	8804.	49892.	587.	13207.
T- 16	69272.1	4.0	1236784.	148727.	258316.	3131.	65753.
T- 18	16510.1	3.0	215482.	33582.	47574.	0.	12593.
T- 20	54467.7	7.0	1568256.	202495.	394218.	0.	81860.
T- 21	16349.3	3.0	221697.	26326.	47111.	554.	9699.
T- 22	13889.6	3.0	178926.	25897.	37669.	471.	10594.
T- 23	16463.1	4.0	286491.	42043.	6511[illegible].	372.	16743.
T- 24	16510.0	4.0	287307.	40671.	59700.	746.	16791.
T- 25	14935.8	3.0	172910.	25823.	36708.	253.	11899.
T- 26	13404.3	3.0	177218.	27264.	34080.	454.	11360.
T- 27	14724.4	3.0	184688.	23460.	42428.	499.	14975.
T- 34	53241.6	3.0	785127.	99269.	144391.	0.	67683.
T- 35	49858.2	3.0	718332.	92961.	126764.	0.	59157.
T- 42	34409.0	10.0	2008264.	200243.	382989.	0.	87485.
T- 66	59335.9	3.0	1206892.	100574.	191091.	0.	55316.
T- 67	20340.1	14.0	2101226.	112623.	405443.	0.	72401.
T- 68	22800.8	3.0	316908.	11594.	42512.	0.	7729.
T- 70	79508.7	3.0	1415056.	67384.	512116.	0.	40430.
T- 71	51748.0	3.0	815730.	43856.	421022.	0.	26314.
T- 72	33283.4	3.0	705192.	22566.	253869.	0.	16925.
T- 73	48833.7	3.0	1531303.	41387.	91050.	0.	45525.
T- 74	23948.5	14.0	2606593.	102294.	352345.	0.	89033.
T- 75	20939.3	15.0	2945845.	129546.	262642.	2130.	83406.
T- 76	51319.7	6.0	3218515.	147878.	461031.	2610.	92206.
T- 77	25965.0	21.0	4356177.	308075.	492920.	4621.	157118.
T- 79	53726.0	32.0	12491768.	864516.	1583327.	11656.	514824.
T- 80	33217.7	29.0	6177487.	446303.	761981.	5987.	174167.
T- 81	16408.4	21.0	2809314.	155749.	297869.	2142.	95396.
T- 82	15318.5	14.0	1732721.	96935.	190236.	1333.	58161.
T- 83	27396.8	7.5	1663626.	101002.	217096.	1625.	54564.
T- 84	23761.5	9.0	1691581.	73705.	196948.	1571.	42290.
T- 85	38225.5	6.0	1684598.	77751.	168460.	1296.	38875.
T- 86	46386.1	3.0	1651113.	44030.	180836.	1415.	33022.
T- 87	67928.6	3.0	1669515.	57569.	483584.	1382.	36844.

TABLE B-3 (cont'd)

HOLE	AREA SQ-M	THICK M	COPPER LBS	COBALT LBS	ZINC LBS	SILVER LBS	MOLY LBS
T- 88	31521.7	3.0	988442.	29920.	138916.	962.	14960.
T- 89	26828.3	6.0	1177776.	88220.	250107.	1273.	18190.
T- 90	27555.3	6.0	1424540.	75664.	196166.	1495.	29892.
T- 91	24459.4	14.0	2495813.	247647.	251516.	2902.	81259.
T-169	60220.9	3.0	1888377.	57162.	265394.	18373.	28581.
T-170	4040.8	4.0	102737.	7580.	34428.	91.	457.
T-171	34145.9	3.0	607712.	18521.	69453.	347.	1158.
T-172	38905.7	6.0	1813492.	44843.	382482.	1846.	25059.
T-174	26857.6	9.0	1479063.	95600.	310016.	1502.	91503.
T-178	61034.8	3.0	879359.	86901.	165526.	1241.	68280.
T-179	31375.1	3.0	478627.	45735.	148906.	532.	32972.
T-181	31702.3	9.0	1745863.	112844.	301456.	1773.	88663.
T-182	47452.1	21.0	5911701.	478566.	1013435.	12949.	304030.
T-183	43752.5	10.0	3090020.	190345.	318890.	2719.	131017.
T-184	48578.0	6.0	2264342.	102101.	312891.	2306.	74106.
T-185	23402.6	15.0	2340377.	160653.	275688.	2777.	93218.
T-186	28259.6	10.0	2395001.	97397.	237903.	1916.	71850.
T-187	29533.2	18.0	4153877.	192226.	450529.	3304.	144169.
T-188	31782.8	3.0	404039.	23704.	134680.	646.	19394.
T-189	10926.3	9.0	388922.	22224.	77784.	667.	20002.
T-191	85552.8	18.0	12763956.	861371.	1505225.	13051.	408934.
T-192	29275.5	9.0	2430980.	156309.	238185.	2531.	77410.
T-193	26572.9	3.0	833260.	67562.	99090.	991.	21620.
T-194	30375.6	3.0	1003990.	25743.	118419.	618.	21624.
T-195	6483.5	12.0	615414.	35167.	90114.	615.	24617.
T-196	19400.2	6.0	1134475.	93389.	147975.	1052.	25649.
T-197	18268.6	12.0	1810849.	82987.	272494.	1610.	74317.
T-198	83085.4	19.0	13352069.	1043548.	1462752.	15163.	490557.
T-199	27030.1	10.0	2443521.	103850.	255043.	1985.	85523.
T-200	35168.7	6.0	2265216.	90609.	238444.	1550.	61995.
T-201	17395.3	18.0	2844716.	164526.	304286.	2300.	84917.
T-202	15299.2	21.0	2488708.	219645.	301332.	2541.	107100.
T-203	38314.4	3.0	1006615.	75334.	97414.	909.	28575.
T-204	72507.5	3.0	2396554.	113068.	405571.	2212.	46702.
T-205	16713.1	3.0	283287.	10198.	65156.	283.	3683.
T-206	12297.9	3.0	448166.	14591.	41690.	250.	9589.
T-207	16553.8	3.0	533115.	24692.	47700.	337.	14029.
T-208	64709.7	6.0	4606684.	344404.	1206512.	3949.	105296.
T-209	21221.8	3.0	863303.	29496.	104316.	432.	23741.
T-211	23528.6	24.0	4530479.	287143.	491334.	3190.	121238.
T-212	15281.3	24.0	2681355.	178204.	304605.	2694.	78741.
T-213	16148.7	24.0	3120413.	214597.	299998.	5693.	85401.
T-214	21617.9	21.0	3939784.	261626.	315491.	14107.	84644.
T-215	28467.0	30.0	7068854.	347411.	607970.	5308.	159230.
T-216	34679.4	27.0	7025575.	412647.	587228.	7406.	185162.
T-217	20233.3	12.0	2229204.	119348.	167362.	1783.	49385.
T-218	8188.4	12.0	867181.	41638.	92714.	833.	22207.

HOLE	AREA SQ-M	THICK M	COPPER LBS	COBALT LBS	ZINC LBS	SILVER LBS	MOLY LBS
T-220	14765.4	12.0	1801969.	95104.	197216.	1301.	71078.
T-221	11851.3	21.0	1978463.	113899.	187019.	1547.	60465.
T-222	93250.8	3.0	1264481.	60063.	158060.	1264.	26870.
T-223	38561.7	3.0	915069.	32681.	78434.	654.	14380.
T-224	32077.0	9.0	3099119.	109285.	272396.	2773.	68507.
T-225	31177.6	3.0	871960.	43334.	79269.	528.	31708.
T-226	48248.4	6.0	2330760.	67060.	368015.	1963.	34348.
T-227	53735.4	6.0	3324475.	151195.	309677.	3097.	81973.
T-228	91690.1	6.0	3496831.	317046.	435161.	3419.	192714.
T-229	17799.4	6.0	874930.	23533.	111629.	603.	18102.
T-230	49254.7	3.0	1669734.	65120.	333947.	1002.	46753.
T-231	15377.8	12.0	1524303.	74026.	166818.	1251.	52131.
T-232	14781.5	18.0	2092564.	121766.	210459.	1804.	72157.
T-233	16062.1	24.0	2981710.	187310.	230870.	2831.	76231.
T-234	15327.9	24.0	2467136.	174591.	251494.	2078.	70668.
T-236	15522.6	18.0	1973311.	121556.	230483.	1579.	55253.
T-237	9644.7	19.0	1423618.	80758.	150127.	828.	38308.
T-238	16338.9	18.0	2063789.	151212.	222663.	1662.	69790.
T-239	29813.5	15.0	3006766.	260249.	368897.	2274.	108648.
TOTALS	4078323.		246119876.	17282160.	33770754.	255172.	8803640.

TOTAL TONNAGE = 95944229.
*

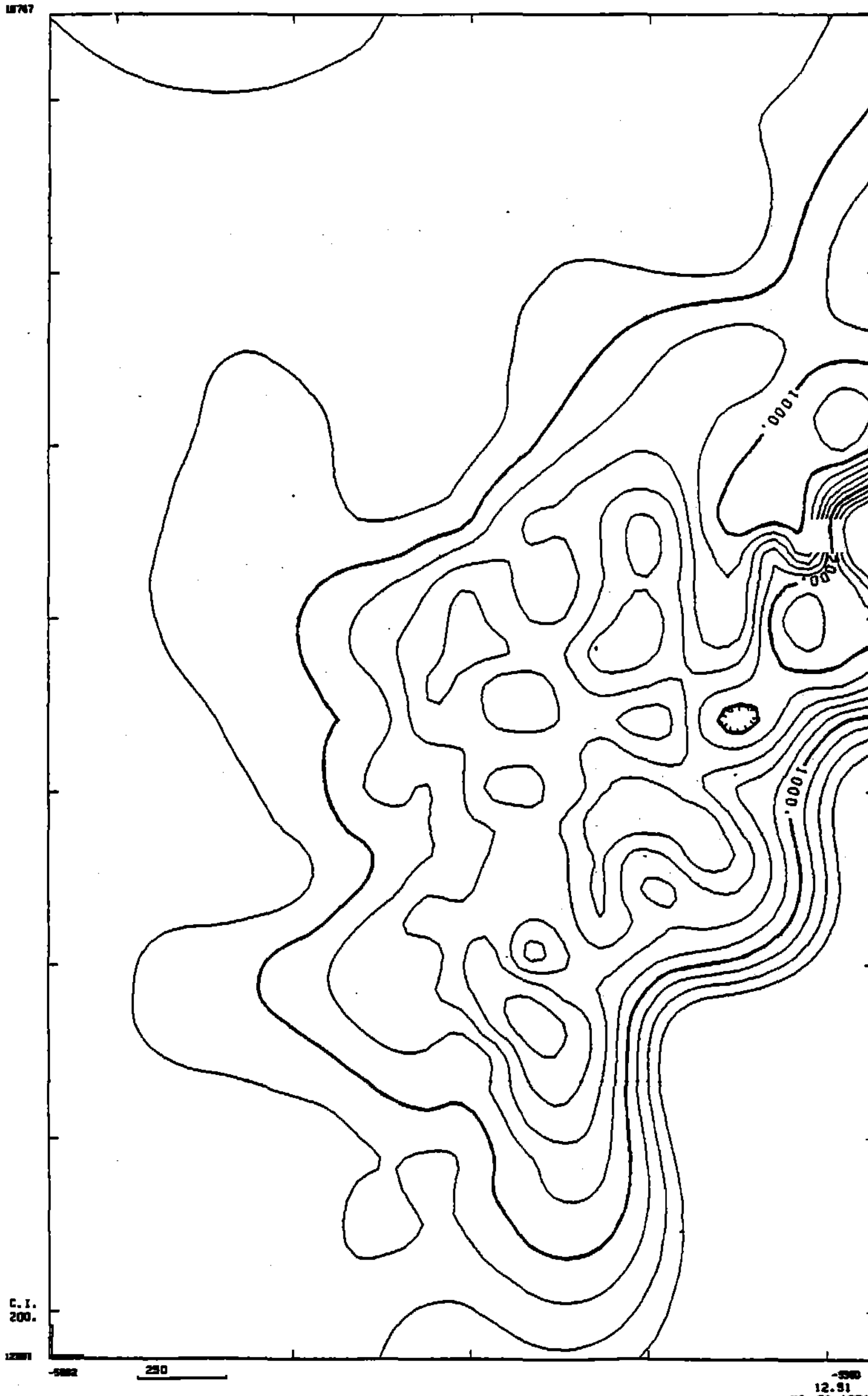


FIGURE B2 COPPER VALUE CONTOUR

5 cm

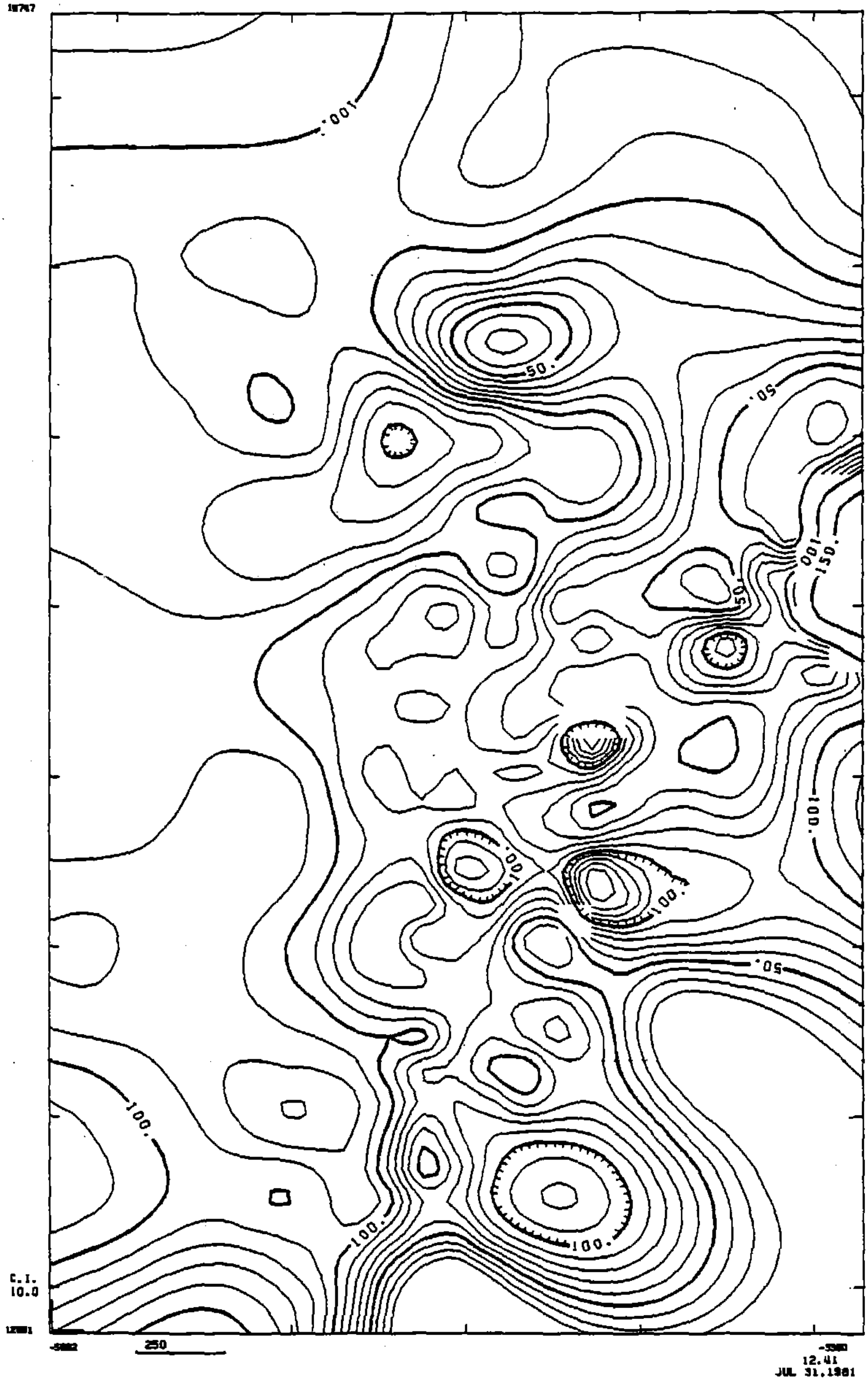


FIGURE B3 COBALT VALUE CONTOUR

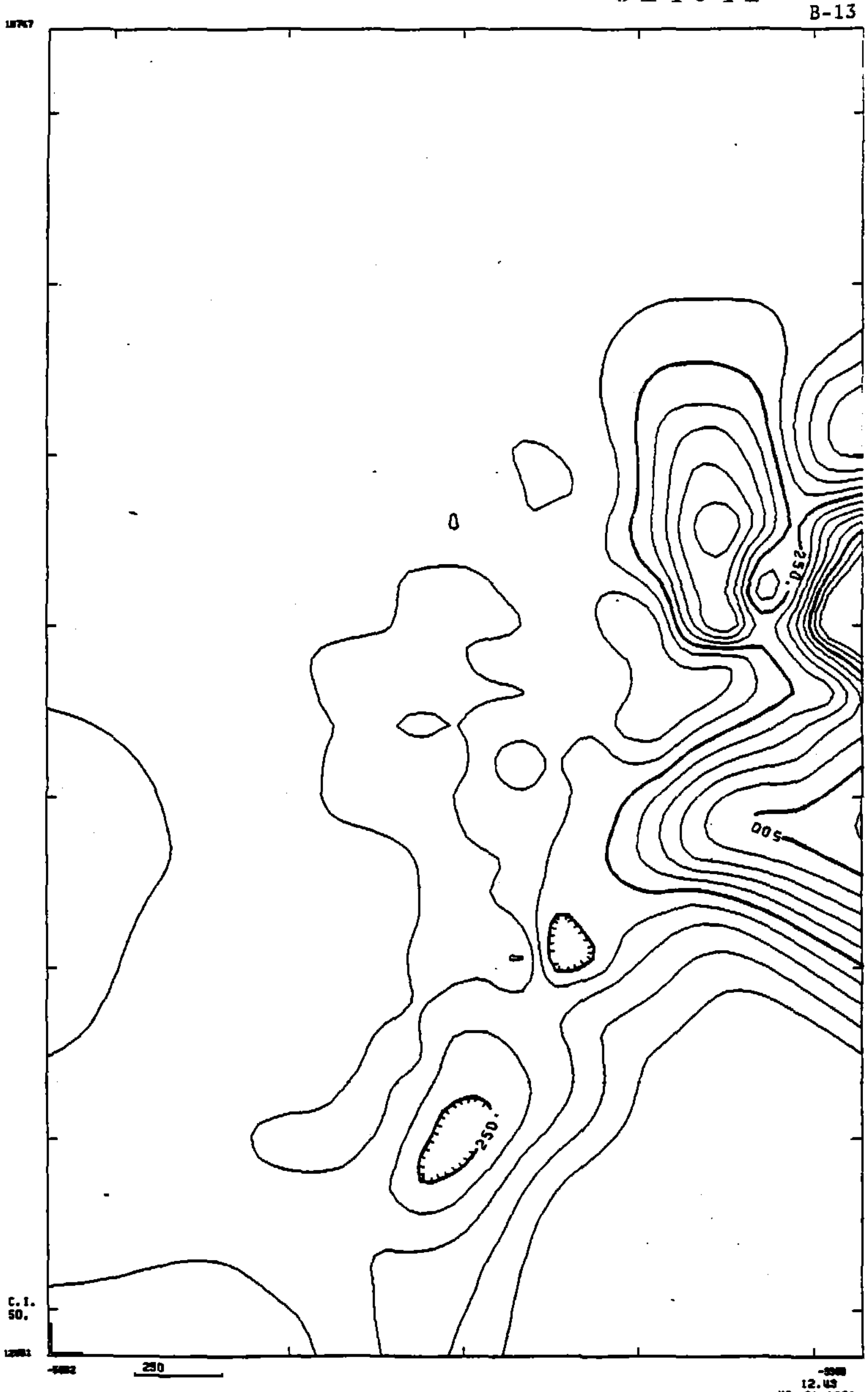


FIGURE B4 ZINC VALUE CONTOUR

5 cm

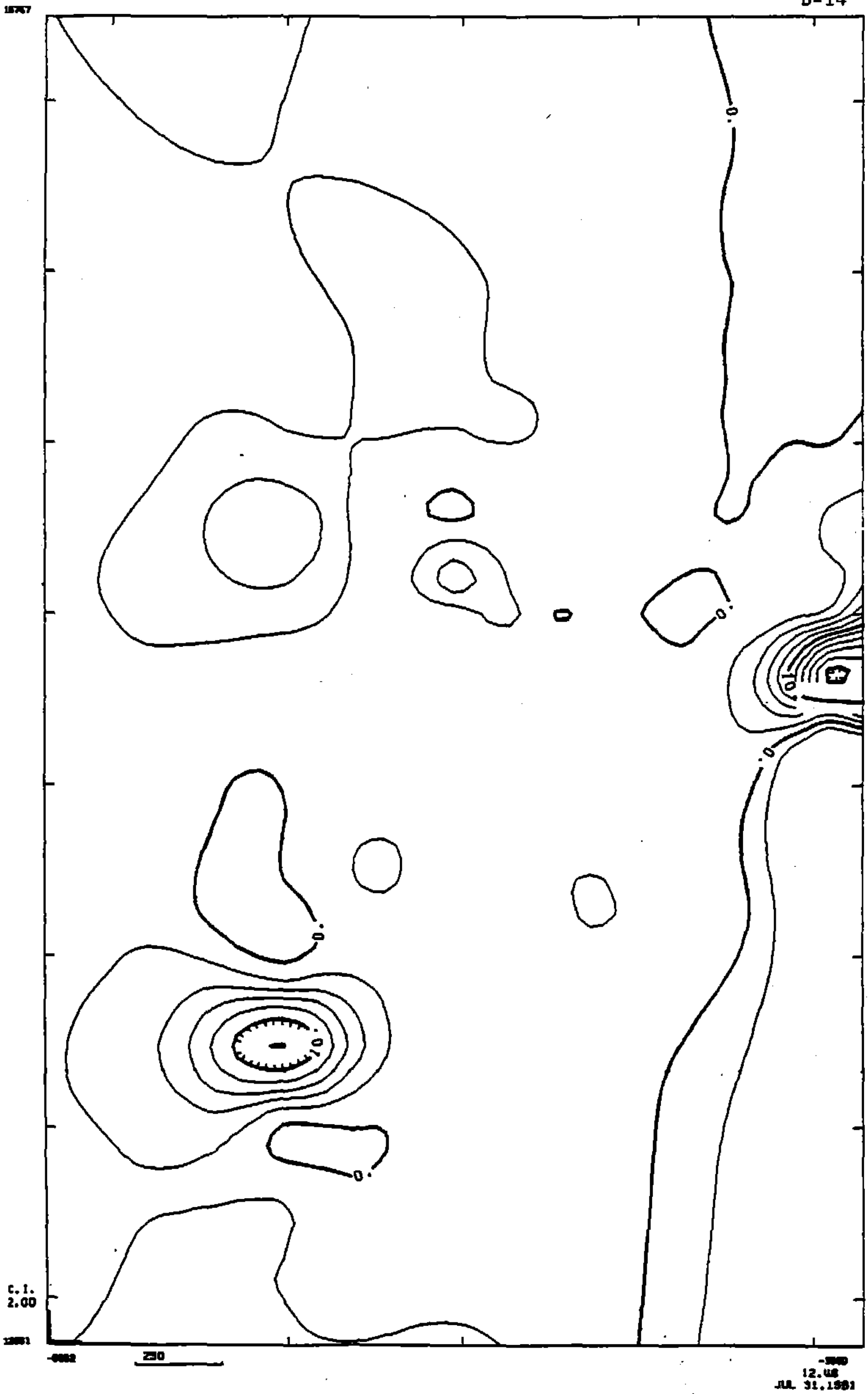


FIGURE B5 SILVER VALUE CONTOUR

5 cm

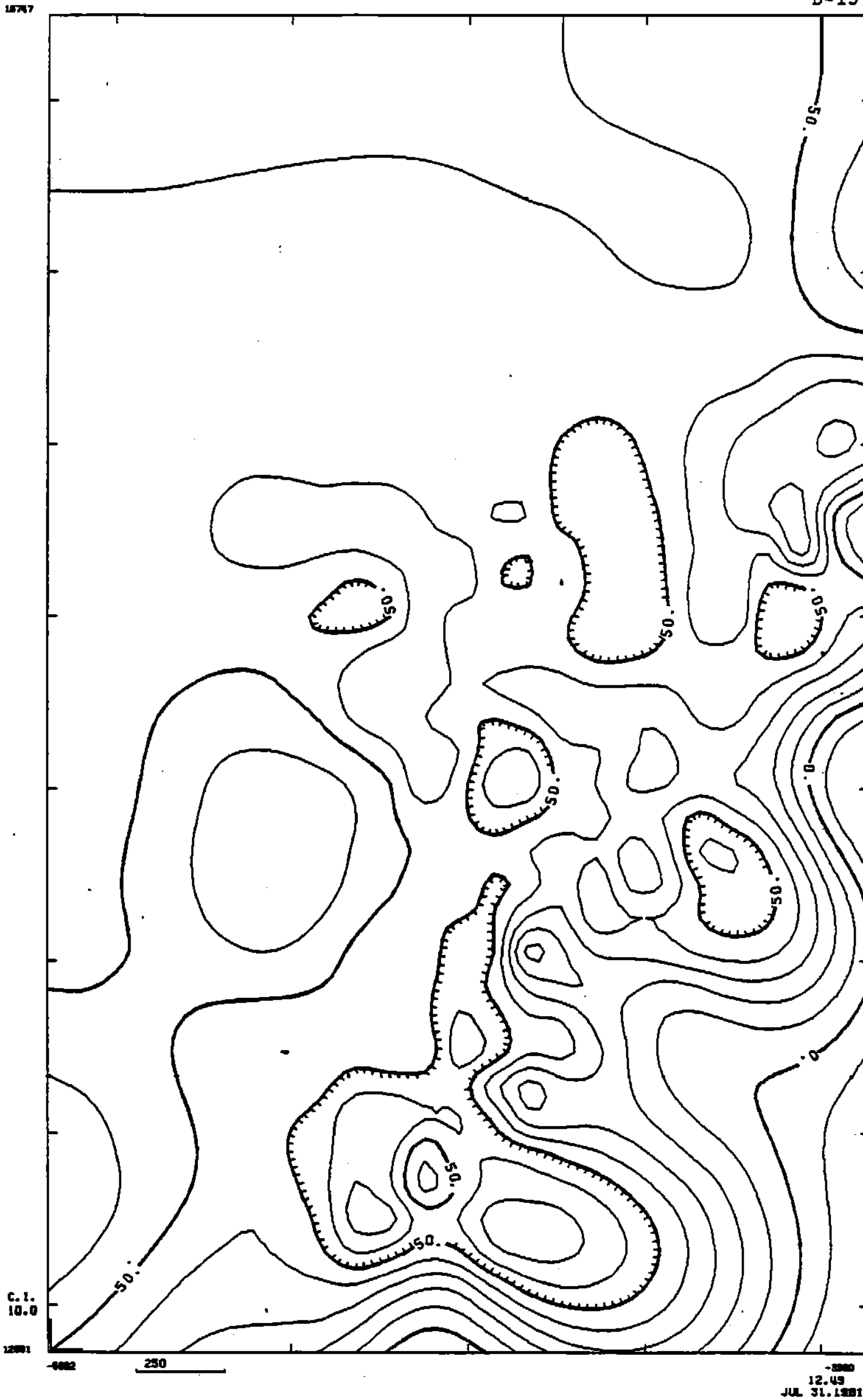
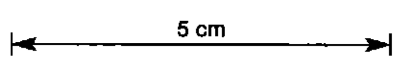


FIGURE B6 MOLY VALUE CONTOUR



APPENDIX C

REPORTS ON THE RESULTS OF PRELIMINARY STUDIES

by

COLORADO SCHOOL OF MINES RESEARCH INSTITUTE
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814045

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CSMRI

CSMRI Project N10326

July 16, 1981

Mr. Sham Zaman
Union Energy Mining Division
Union Oil Company of California
1846 W. Grant Road, Suite 105
Tucson, Arizona 85705

Dear Mr. Zaman:

The purpose of this letter is to report the results of preliminary studies with a Tasmanian tailings sample. The work was designed to explore the potential for the recovery of cobalt from this material.

Studies were authorized under Contract No. 81-2 in two parts. An initial part of the work was defined in a letter dated May 1, 1981, encompassing preliminary analysis, a heavy liquid separation, XRD and chemical analysis of the products as well as petrographic studies on polished thin sections. The objectives of the part of the studies were to identify the gangue minerals and to characterize the occurrence of cobalt. A second part of the work was defined in a letter dated June 8, 1981, encompassing limited flotation testing. The objective of the second part of the work was primarily to determine if cobalt had a tendency to follow and concentrate with a pyrite flotation product.

The details and results of this work are discussed as follows.

Results and Discussion

As you know, the sample was a disappointment in terms of cobalt content, showing a head analysis of 0.01% Co as opposed to approximately 0.20% Co as initially believed. The only other metal of interest was copper at 0.166%. A fire assay of the head showed Au as a possible trace ($\leq$0.005 oz/ton) and Ag as essentially nil.

Work done by our minerology group (part 1 of the work program) showed the tailings to be composed of, by a decreasing order of abundance, quartz, chlorite, mica, pyrite, a possible trace chalcopyrite and hematite. A polished grain mount had been prepared for ore microscopy and examined only briefly. No specific occurrences of cobalt minerals were observed optically and it was concluded that the chance of finding discrete cobalt minerals at this low of a level of cobalt content was very poor at best. The heavy liquid test (specific gravity 3.3) resulted in a heavy fraction of 6.6% of the initial weight containing approximately 63% of the input cobalt. The cobalt analysis of the heavy fraction was still very low, at 0.0725% Co. Further details of this initial work are listed in Exhibit 2, attached.

The flotation work explored the use of standard pyrite flotation procedures on unground feed as Test 2, with Test 1 conducted in a similar manner after acid scrubbing of the unground feed. Details of these tests are listed in Exhibit 3 and summarized as follows.

	Weight and Metal Distribution, %							
	Test 1				Test 2			
Item	Weight	Cu	Co	S	Weight	Cu	Co	S
Rougher Concentrate	6.8	20.9	58.6	93.1	7.5	12.9	74.2	96.2
Scavenger Concentrate	5.3	26.2	5.4	6.6	1.3	11.1	3.5	3.3
Tailings	87.9	52.9	36.0	0.3	91.2	76.0	22.3	0.5
Head (calculated)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Combined Concentrate (calc.)	12.1	47.1	64.0	97.7	8.8	24.0	77.7	99.5

From this data, it can be seen that cobalt did tend to concentrate with the pyrite, although an appreciable amount was lost to the tailings. However, the cobalt content in the rougher concentrate never exceeded 0.085%, which is a discouragingly low level for initiation of chemical processing.

Please let me know if you have any question on the work covered in this report.

Very truly yours,

Hal D Peterson

Hal D. Peterson
Projects Manager
Process Division

HDP/klk

EXHIBIT 1

SAMPLE DESCRIPTION AND PREPARATION

CSMRI Sample No. 1

Sponsor's Designation of Sample:	None.
Dated Received at Institute:	Approximately April 10, 1981.
Sample Weight:	Approximately 6 lb.
Sample Description:	Finely ground material of approximately -65 +400M.
Method of Preparation:	The sample was blended and portions split out for mineralogical and metallurgical investigations, chemical analysis, and detailed analysis by x-ray diffraction and x-ray fluorescence.

Chemical Analysis

Cu	Co	S
0.166	0.010	3.55

EXHIBIT 2

MINERALOGICAL EXAMINATION

MINERALOGICAL EXAMINATION

INTRODUCTION

A few kilograms of fine sand-sized (nominal 65-200 mesh) bulk tailings were received for detailed analytical and mineralogical studies. These analyses were to precede later metallurgical test work.

OBJECTIVES

The purposes of the analytical/mineralogical work were:

1. To obtain certain information about the chemical makeup of the bulk tailings

2. To determine the overall bulk mineralogy of the tailings

3. To perform heavy liquid separations for the purpose of upgrading cobalt and various other heavy metals which might be present

4. To determine the mineralogical occurrence(s) of cobalt, with the more specific purpose of determining whether the cobalt occurs with and in the pyrite.

SUMMARY

About 6.6 weight percent of the -20, +200 mesh fraction of the tailings sinks in a heavy liquid of specific gravity 3.3 (methylene iodide). This heavy mineral fraction is rich in pyrite (est. $\geq$ 60% pyrite), but it only contains a calculated 63 percent (approx.) of the available cobalt and only assays a low 0.073 percent (average of two determinations: 0.070 and 0.075 percent) cobalt by atomic absorption.

As determined by x-ray diffraction, the main minerals in the bulk sample, in approximate order of decreasing abundance, are: quartz, chlorite, mica, pyrite, siderite, chalcopyrite(?), and hematite.

Routine semiquantitative x-ray fluorescence (XRF) analyses of the bulk tailings and heavy (>3.3 sp. gr.) minerals show no detectable cobalt at a detection limit of about 20-30 parts per million. These XRF analyses do show 0.25 percent copper in the bulk tailings being upgraded to 0.43 percent copper in the heavy (>3.3) minerals, but no concentrations of other elements (with atomic numbers greater than 21) which are particularly interesting.

Fire assay on the bulk tailing shows only a trace (<.005 oz/T) gold and no detectable (nil) silver.

A polished grain mount for ore microscopy was prepared from the heavy mineral fraction, but this was only examined rather briefly before the work was halted. No specific occurrences of cobalt (minerals) were observed optically, and the chances of finding discrete cobalt minerals at a level of 0.07 percent cobalt (700 ppm) are fairly small, anyway.

RECOMMENDATIONS

The chances for successful determination of the cobalt occurrence(s) would be increased if a sample with higher cobalt content were available.

DETAILS OF THE INVESTIGATION

Initial Screening and Preparation

An approximate 100 gram split of the bulk tailings sample was wet-screened at 200-mesh to remove the fines prior to heavy liquid separations. The plus 200-mesh was then screened at 20-mesh to scalp off coarse mineral fragments, pebbles, organic (plant) material, and so on. The size distribution in the three screen fractions was as follows:

Screen Fraction	Wt. %
+20m	1.15
-20, +200m	89.65
-200m	9.20

The washed and dried 20 x 200m material was later used as feed to the heavy liquid separation(s).

A portion of the bulk tailings split was also pulverized to a fine (approx. 325-mesh) powder at this stage for subsequent chemical analyses, fire assays, and x-ray diffraction.

Heavy Liquid Separations

A 47-gram (approximate) charge of the cleaned 20 x 200-mesh fraction was first separated in a heavy liquid of specific gravity 2.8. This separation was not really selective enough, in that it yielded too much "heavy" fraction. So the resulting >2.8 material was then rerun through a 3.3 specific gravity liquid, to float out more of the light minerals. The yields of low, intermediate, and high gravity products for these two separations were:

Product (of 20 x 200 mesh)	Wt. %
<2.8	83.85
>2.8, <3.3	9.56
>3.3	6.59

A second separation, at specific gravity 3.3 on a 43-gram (approx.) charge of 20 x 200-mesh material, was also run to provide enough heavy minerals for all subsequent procedures. The reproducibility of >3.3 yield was very good, being 6.57 weight percent as compared to 6.59 percent for the first separation.

The >3.3 material from both separations was combined, ferromagnetic material (about 4.3 percent by weight, mostly tramp iron?) was removed, and a split of the remaining nonmagnetic material was pulverized for cobalt analysis, XRF analysis, and bulk heavy minerals x-ray diffraction analysis. Polished grain mounts were also prepared.

A split of the light (sp. gr. <3.3) minerals was also pulverized for Au/Ag fire assays and cobalt analysis. A chemical pulp was also prepared from the intermediate gravity (>2.8, <3.3) material.

X-ray Diffraction (XRD) Analyses

Powders of both the bulk tailings and the heavy (sp. gr. >3.3) minerals were run on the x-ray diffractometer over the range $2\theta = 3\text{-}61^{o}$ (CuKα radiation) for mineral identification. The patterns were interpreted, and semiquantitative estimates (based on peak heights) were made of the amounts of the minerals identified as follows:

	Est. Amount* in Sample	
Mineral	Bulk	>3.3
Quartz	Major	Minor-Trace
Chlorite**	Moderate	Moderate-Minor
Mica	Moderate-Minor	Trace
Pyrite	Minor	Major
Siderite	Minor	Moderate
Hematite	Trace	Minor-Trace
Chalcopyrite(?)	Trace(?)	Trace(?)
Total Unidentified	Trace(s)	Minor-Trace(s)

*Major >30%
Moderate 10-30%
Minor 3-10%
Trace <3%

**Could include some "hidden" kaolinite, due to peak overlaps.

XRF Analyses

Samples of the bulk tailings and the nonmagnetic heavy (>3.3) minerals were submitted to an outside laboratory for qualitative, semiquantitative x-ray fluorescence (XRF) analyses for all elements heavier than scandium (at. no. 21). The XRF report is attached.

These analyses failed to detect cobalt at estimated detection limits of 22 ppm for the bulk sample and 30 ppm for the (high-iron matrix) heavy minerals. They did show 0.25 percent copper in the bulk tailings and 0.43 percent copper in the heavies (>3.3), but no other metal concentrations that seemed particularly interesting.

Cobalt Analyses (AA)

The bulk sample, minus 200-mesh fines, light (<3.3) minerals, heavy (>3.3) minerals, and intermediate gravity (>2.8, <3.3) minerals were submitted to the CSMRI Analytical Laboratory for cobalt analyses by atomic absorption. The analytical report is attached, and the results are repeated here as follows:

Sample	Cobalt (Co), %
Bulk Tailings	0.0099, 0.0100*
<3.3	0.003
>3.3 non-mags	0.070, 0.075*
-200 mesh	0.030
>2.8, <3.3	0.0090

*Duplicate determinations

Although the cobalt concentration in the 20 x 200-mesh feed to the heavy liquid separations was not determined by experiment, it was calculated to be approximately 0.008 percent, using the screening weight data and the above analytical results.

Figuring 0.008 percent cobalt in the feed to heavy liquid, one can calculate that about 63 percent of the available cobalt reports to the >3.3 heavy minerals in only about 6.6 percent of the weight. However, the cobalt assay is still so low (average of two determinations = 0.0725 percent) in the heavy mineral concentrate as to be rather discouraging. It seems likely that a pyrite concentrate from just about anywhere might well contain that much cobalt as background level.

Fire Assays

Pulps of the bulk tailings and the light (<3.3) minerals were submitted to an outside laboratory for silver and gold fire assays in ounces/ton (oz/T).

The assay report is attached, and the results are repeated here as follows:

Sample	Au(oz/T)	Ag(oz/T)
Bulk tailings	trace*	nil
<3.3	trace*	0.28

*trace is <.005 oz/T

The above result of 0.28 oz/T of silver in the light minerals does not seem reasonable with respect to the other results, and no real explanation can be offered. It could be from contamination, from coarse particles of an Ag phase (and thus lack of homogeneity), or simply a mistake. In any event, the prospects for precious metals do not seem very encouraging based on these results.

Ore Microscopy

Polished grain mounts of the intermediate gravity (>2.8, <3.3) and heavy (>3.3) minerals were examined rather briefly in reflected light under the ore microscope, looking for discrete grains or inclusions of a possible cobalt mineral included in pyrite grains or elsewhere. Nothing promising was observed.

814054 C-10

FLUORESCENT X RAY SPECTROGRAPHIC Analytical Laboratory

718 Sherman Street (rear)
Denver, Colorado 80203
Phone (303) 837-1396
Merlyn L. Salmon, Manager

XXXX QUALITATIVE
XXXX SEMI-QUANTITATIVE
_____ QUANTITATIVE

ANALYTICAL REPORT

Job Number 28553
Page 1 of 2 Pages
Date 12 May 1981

TO: Colorado School of Mines Research Institute
P.O. 34388 N10326 R. Corbett

SAMPLE: Bulk Tailings

Copper	0.25	Iron	5.2	Lanthanum	
Silver		Cobalt	*	Cerium	
Gold		Nickel	0.011	Praseodymium	
Zinc	0.025	Cesium		Neodymium	
Cadmium		Rubidium	0.011	Samarium	
Mercury		Barium	0.46	Europium	
Gallium		Strontium	0.013	Gadolinium	
Indium		Titanium	0.068	Terbium	
Thallium		Zirconium	0.040	Dysprosium	
Germanium		Hafnium		Holmium	
Tin		Thorium		Erbium	
Lead	0.029	Vanadium		Thulium	
Arsenic	0.004	Columbium	0.008	Ytterbium	
Antimony		Tantalum		Lutetium	
Bismuth		Chromium		Yttrium	
Selenium		Molybdenum	0.007		
Tellurium		Tungsten			
Bromine		Uranium			
Iodine		Manganese	0.060		

The values above are estimated elemental concentrations in:
XXXX per cent _____ parts per million _____ grams per liter

No check was made for elements with atomic numbers less than 22.

*NONE DETECTED @ EST. DET. LIMIT OF 22ppm

By [signature]

EXHIBIT 3

FLOTATION TESTS

Flotation Test 1

Purpose: To produce a bulk sulfide concentrate for mineralogical evaluation.

Sample: Approximately 500g of CSMRI Sample 1.

Test Conditions: Frother mix is a 2:1 mix of MIBC;Aerofroth 65.

	Time min	Solids %	pH Start	pH Finish	H_2SO_4	AP 25	Aero 3477	Aero 350	$CuSO_4$	Frother Mix	Na_2CO_3
					Reagents, lb/ton of test feed						
Scrubbing @ 1,200 rpm	30	~60	2.5	2.5	1.5	--	--	--	--	--	--
Filter and re-pulp	--	--	--	--	--	--	--	--	--	--	--
Conditioning @ 400 rpm	10	50	7.0	6.6	--	0.1	0.04	--	--	--	1.4
Rougher Conditioning	1	30	--	--	--	--	--	0.1	--	--	--
Rougher Flotation	1	--	--	6.8	--	--	--	--	--	--	--
Scavenger Conditioning	10	~25	6.2	6.4	--	--	--	--	1.0	--	--
Scavenger Conditioning	1	--	--	6.4	--	--	--	0.1	--	0.1	--
Scavenger Flotation	10	--	--	6.8	--	--	--	--	--	--	--

Results:

	Weight %	Chemical Analysis Cu %	Co %	Stot %	Percent Distribution Cu	Co	Stot
Head (analyzed)	--	0.166	0.010	3.55	--	--	--
Head (calculated)	100.0	0.159	0.010	2.68	100.0	100.0	100.0
Rougher Concentrate	6.8	0.490	0.084	36.7	20.9	58.6	93.1
Scavenger Concentrate	5.3	0.787	0.010	3.35	26.2	5.4	6.6
Tailing	87.9	0.096	0.004	0.010	52.9	36.0	0.3
Combined concentrate (calculated)	12.1	0.620	0.052	22.1	47.1	64.0	97.7

Test Notes and Observations: Initial scrubbing and conditioning performed in a Wemco octagonal cell with a 2 3/4 in. diameter triple propeller.

COLORADO SCHOOL OF MINES RESEARCH INSTITUTE

EXHIBIT 3

Flotation Test 2

Purpose: To produce a bulk sulfide concentrate for mineralogical evaluation.

Sample: Approximately 500g of CSMRI Sample 1.

Test Conditions: Frother mix is a 2:1 mix of MIBC:Aerofroth 65.

	Time min	Solids %	pH Start	pH Finish	Reagents, lb/ton of test feed: Aero 350	Frother Mix	$CuSO_4$	Na_2CO_3
Conditioning	3	30	7.7	6.8	.1	.1	--	1.12
Flotation	2	30	--	6.8	--	--	--	--
Conditioning	10	~25	6.2	6.2	--	--	1.0	--
Conditioning	1	--	--	6.3	.1	.1	--	--
Flotation	10	--	--	6.4	--	--	--	--

Results:

	Weight %	Chemical Analysis Cu %	Co %	Stot %	Percent Distribution Cu	Co	Stot
Head (analyzed)	--	0.166	0.010	3.55	--	--	--
Head (calculated)	100.0	0.174	0.008	3.01	100.0	100.0	100.0
Rougher Concentrate	7.5	0.300	0.081	38.6	12.9	74.2	96.2
Scavenger Concentrate	1.3	1.49	0.022	7.66	11.1	3.5	3.3
Tailing	91.2	0.145	0.002	0.015	76.0	22.3	0.5
Combined concentrate (calculated)	8.8	0.476	0.072	34.0	24.0	77.7	99.5