CANADIAN LENCOURT MINES LTD.

SANDY K MINES

1000 STPD TAILS REMILLING PROJECT

GOWGANDA, ONTARIO

FEASIBILITY STUDY

APRIL 1987

Prepared By

Kilborn Limited
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KILBORN

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April 27, 1987

Canadian Lencourt Mines Ltd.
Suite 400
8 King Street East
Toronto, Ontario
M5C 1B5

Attention: Mr. Ross Lawrence, President

Reference: Feasibility Study
On The, Sandy K Silver Tailings Project, Gowganda Ontario

Dear Sirs:

Please find enclosed our Feasibility report on the Sandy K Tailings project. This report is based on current available data which in some cases requires additional confirmation. The main area of confirmation is the metallurgical testwork and this testwork is underway and should be completed in 2 - 3 weeks.

Specific areas that are to be confirmed are:

1) Current available used equipment prices have been used. These are subject to availability and some adjustment at the time of commitment.

2) The testwork shows that additional grinding with 450 - 500 HP, instead of 250 HP, will increase leach extractions to 86 - 87%. Capital is included for this additional grinding and a silver recovery of 85% can be reasonably anticipated.

3) The testwork shows that the leach extraction curve tends to flatten out at approximately 35 hrs. retention time. If confirmed the leach tankage could be reduced.

4) The greatest discrepancy exists in the filter area as used in this report. Filtration tests were run without flocculant and with poor success. Filter sizing was based on data from Timmins, Kirkland Lake and Val d'Or areas at 80 - 100 lbs. per square foot per hour. Failure to confirm this rate would have an influence on the capital costs.

KILBORN

The overall accuracy of the estimate is plus or minus 20% and with the exception of the filtration data we would not expect any significant change from the costs and recoveries as stated in this study.

Yours very truly,

KILBORN LIMITED

C.A. Freitag, P.Eng.
Senior Vice-President
and General Manager

Encl.

cc: R.G. Jenkins

CAF/ps

FEASIBILITY STUDY FOR CANADIAN LENCOURT MINES LTD. ON THE SANDY K. MINES 1000 STPD TAILS REMILLING PROJECT GOWGANDA, ONTARIO

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VOLUME II

APPENDICES

1.0 SUMMARY

1.1 THE PROJECT:

Canadian Lencourt Mines Limited has entered into an agreement with Sandy K Mines and Siscoe Metals of Ontario for the treatment of all the tailings on a group of claims near Gowganda, Ontario. There are approximately 1,827,000 tons having an average grade of 1.43 ounces of silver per ton.

In this report the feasibility of doing this has been investigated based on treating 1000 tons per day eight months of the year or a total of 225,000 tons annually.

1.2 CONCLUSIONS:

.1 The silver tailings at Sandy K are ammenable to treatment by a standard cyanide leach process and recovery by a Merrill-Crowe system subject to a satisfactory system for separating the leach solution from the pulp.

.2 Recoveries of better than 85 percent appear to be attainable providing sufficient grinding and leach times are provided.

.3 Given favourable silver prices and economy of design, construction and operation, the project can yield a significant profit over the seven-year life.

1.3 RECOMMENDATIONS:

It is recommended that Canadian Lencourt proceed with the project with the ultimate objective of starting operations on April 1, 1988 subject to giving priority to, and the satisfactory completion of the following key activities:

.1 Additional metallurgical testing which we understand is now underway.

.2 Initiation of the preparation of all applications for environmental permits.

.3 Confirm and where possible obtain commitments for used equipment.

2.0 INTRODUCTION

2.1 PURPOSE OF THE STUDY

Sandy K Mines Ltd. is the registered holder of fourteen contiguous mining claims of approximately 547 acres in Haultain and Nicol Townships in the Larder Lake Mining Division. These claims are about two miles from the town of Govganda, and they are readily accessible from provincial highway No. 560.

From 1909 until 1972 the claims were worked continuously, for silver, as well some cobalt and nickel was produced. The old records and reports show that more than 42 million ounces of silver were recovered from two million tons of ore. The bulk of the residues from these old operations remain on the site.

Canadian Lencourt Mines Ltd. has entered into an option agreement with Sandy K Mines to acquire the old mill tailings, and the purpose of the study reported here is to examine the feasibility of reprocessing these tailings.

The method has been to examine all available historical data, to test the deposit by taking core samples and assaying them, to carry out metallurgical tests, to examine the environmental problems and to determine the potential earnings by developing capital and operating costs for a typical processing facility.

2.2 PROJECT TEAM

This study was undertaken by a multi-disciplined group of specialists. The team members and their respective area of responsibility are:

.1 Kilborn Limited

Kilborn Limited carried out the required studies for developing the metallurgical flow sheet, equipment selection and sizing as well as the development of capital and operating cost data. Kilborn has prepared this report based on their studies together with the studies of others as appended hereto.

.2 Robert L.V. Ekstrom B.A.Sc., P. Eng.

Robert Ekstrom carried out a field sampling program of the insitu tailings, Bell White Laboratories carried out assays on the samples. Based on these investigations Robert Ekstrom performed ore reserve calculations. These studies are appended hereto and presented as Appendix A.

.3 Witteck Development Inc.

Metallurgical testing and head assaying was carried out by Witteck Development Inc. The results of these analysis are presented in Appendix B and have been incorporated into the metallurgical design and equipment sizing studies by Kilborn Limited.

.4 SENES Consultants Limited

Environmental considerations were studied by SENES Consultants Limited and their findings are presented herein as Appendix D.

.5 Markham Data Incorporated

Financial analysis of the project economics has been carried out by Markham Data and is presented herein as Appendix E. This is based on the various agreements and royalties that will govern the future operations of the property.

2.3 GENERAL DESCRIPTION

The fourteen claims that now constitute the Sandy K property were originally staked and held in 1909 by three separate groups, and identified as the Bonsall, Millerett and Miller Lake O'Brien. In 1912 M.J. O'Brien Ltd., the owners and operators of the Miller Lake O'Brien group acquired the single Millerett claim with its surface plant including a 35-40 TPD mill, and in 1922 the Bonsall group of three claims were also acquired. Thereafter the property was operated continuously as a single entity until 1972 when it ceased production.

The small mill on the Millerett claim treated all of the ore, extracted from the properties apart from a small tonnage of high-grade, hand picked ore that was shipped directly to the smelters. It operated from 1911 until 1939 and ceased operation at the outbreak of World War II. It was never started up again.

The Millerett mill was a gravity plant employing jigs, stamps and tables. Over its operating life this mill was modified to increase its capacity from 30-40 tons per day to 80 - 100 tons per day and the old records indicate that it treated over 600,000 tons of ore and produced over 15 million ounces of silver.

In 1945 ownership of the claims transferred to Siscoe Metals of Ontario Limited, a subsidiary of Siscoe Gold Mines Limited. This company built a new mill (No.1 Mill) having a capacity of 60 tons per day. This was located about 2000 feet south of the old Millerett mill at the Siscoe main shaft area. This was a combined gravity flotation mill that treated about 300,000 tons of ore.

In 1950 Siscoe erected a tailings retreatment plant (the No.2 Mill) to process tailings from the old Miller Lake O'Brien operation. There were reported to be 400,000 tons of these tailings averaging 3.8 ounces of silver per ton. This plant never operated at this capacity for a sustained period but between 1950 and 1956 it did treat approximately 235,000 tons of tailings.

In 1956 the smaller No.1 Mill and the buildings around Siscoe main shaft area were destroyed by fire. This mill was re-built the same year but it was used for custom ore and similar applications and the No.2 Mill was modified to treat ore. There was no further treatment of the tailings after 1956.

From 1956 until 1972, when it was closed, the No.2 Mill treated a total of 802,731 tons of ore from underground.

Based on information contained in government reports the tonnage and grade of tailings can be estimated as follows:

Period	Tons	Grade oz/ton	Total Ounces Silver	
1911 - 1939	400,000	3.8	1,520,000	(1)
1946 - 1956	350,000	1.0	350,000	(2)
1950 - 1956	(235,000)	3.8	(893,000)	(3)
1950 - 1956	235,000	1.0	235,000	(3)
1956 - 1972	800,000	1.0	800,000	(4)
Sub-Total	1,550,000	1.30	2,012,000	
1911 - 1939	200,000	3.00	600,000	
	1,750,000	1.49	2,612,000	(5)

.1 Siscoe estimated this tonnage and grade, as reported by the Ontario Department of Mines in 1951. This was probably a minimum recoverable tonnage based on conditions in 1950 when the retreatment plant was built.

.2 This is an estimate of the tons treated in the No.1 Mill.

.3 This is the tons of tailings retreated by Siscoe and returned to the tailings areas.

.4 This is an estimate of the tons of ore milled in the No.2 Mill based on annual reports of the Ontario Department of Mines.

.5 Records from the period prior to 1940 suggest a higher tonnage of ore treated than that indicated in (2) above. This is an estimate based on the recorded production of silver and the method of processing employed.

In 1981, after ownership of the claims was acquired by Sandy K Mines Ltd., Watts, Griffis and McOuat undertook a detailed assessment of the work that they undertook both on surface and underground up to that time. Included in this was an evaluation of the old tailings, and an estimate of the proven, probable and possible tonnage and grade of the tailings was made.

Subsequently Canadian Lencourt Mines commissioned a study of the recovery of silver from the tailings. In 1986 - 87 further sampling of the tailings was carried out to place all of the reserve in the proven category, as well as carry out preliminary metallurgical, environmental and engineering studies. The results of this work are recorded in this report.

2.4 UNITS

All costs are expressed in terms of Current (1987) Canadian dollars and units are imperial.

3.0 RESERVES

3.1 PREVIOUS WORK

In 1981, a sampling program of the tailings was carried out by Watts, Griffis and McOuat. This was done for the most part by hand augering using an agricultural soil sampler which was effective only to a depth of 15 feet or to the water table. Some additional sampling using a drive pipe was also undertaken to sample the deeper and wetter material.

The extent of the 1981 program can be summarized as follows:

Hand Augering - 545 holes total length 2760 feet
Drive Pipe - 27 holes total length 629 feet

Holes were sampled generally at 5 foot intervals, and a total of 1236 samples were sent for fire assays.

From this sampling the following reserves were estimated:

	Tons	Oz Ag/Ton
Proven	1,087,000	1.48
Probable	412,000	1.35
Possible	202,000	1.52
Total	1,701,000	1.45

A more detailed description of this work is contained in Appendix A to this report.

3.2 1986/87 SAMPLING PROGRAM

In 1981, WGM recommended further sampling using the drive pipe method to test the deeper and wetter materials. The 1987 sampling program generally followed this recommendation; however, a Wink Vibra Core Sonic Drill was found to give better samples faster and more economically than the drive pipe method, and it was used throughout.

The aim of this 1987 program was to place all of the reserves in the proven category. To do this all of the tailings included had to be within 100 feet of a drill hole. In addition all of the holes had to penetrate the tailings pile and define the substrata.

To achieve this objective the work consisted of evaluating the previous work by WGM, siting and completing sufficient holes, sampling and assaying all holes, determining a representative tonnage factor and calculating the new reserves.

From a detailed check of the previous work and further work on the tonnage factor, it was decided that all of the proven reserves then determined could be incorporated in the current work. This permitted the work in 1987 to concentrate on areas previously defined as probable and possible.

In 1981, only one sample was taken to determine the tonnage factor of 16.26 cubic feet per ton. This was not adequate for the purposes of this study. Consequently a further eleven samples were taken in different parts of the tailings pile. This additional work did confirm that the factor used previously was realistic, and the decision was taken to use it for all of the calculations of reserves.

In 1986/87 a total of 152 holes having a total length of 2,986.5 feet were drilled. Four hundred and fifty-seven samples were collected and fire-assayed.

3.3 PROVEN RESERVES

The total proven reserves of tailings are now estimated to be as follows:

	Tons	Silver oz/ton
From 1981 program above 15 feet	1,087,000	1.48
From 1987 program above 15 feet	101,000	1.30
Below 15 feet	639,000	1.35
	1,827,000	1.43

In Appendix A to this report, details of the drilling, sampling, and calculation of the reserves are presented. Also, the reserve plans show the locations of all of the holes and the reserve blocks.

3.4 RECOVERABLE RESERVES

An examination of the silver distribution throughout the reserves by grade shows reasonable continuity of grade from hole to hole. This is to be expected especially in those sections containing the tails from the No. 2 Mill which probably discharged a consistent grade of material. There are however, sections containing material with a considerably higher grade than the average of 1.42 ounces per ton. On the other hand there is little material that averages less than 0.90 ounces per ton. Further, much of the higher grade material is accessable from the surface. Consequently it will be possible to maintain a fairly constant head grade going to the retreatment plant.

It is proposed to treat 225,000 tons of tailing per year, which are anticipated by blending, should have the following grade:

Operating Year	Annual Production Tons	Silver Grade oz/Ton
Year 1	225,000	2.0
2	225,000	1.75
3	225,000	1.75
4	225,000	1.5
5	225,000	1.5
6	225,000	1.0
7	225,000	1.0

By the end of seven years of operation at this rate of reclamation of the tailings about 250,000 tons would be left having an estimated grade of less than one ounce per ton. Whether or not this will be treated will depend on accessibility and debris content of the remaining deposits. Therefore it has not been included in the valuation.

4.0 PROCESSING

4.1 PLANT DESCRIPTION AND OPERATION

General Description

The project is planned to be a seasonal operation running approximately eight months per year. The mill and filter areas are sheltered under a roof and the process tanks are set on prepared sand bases outside with a ring dyke. The filter, floors, stairs and handrailing in the grinding and filtering areas will be constructed from locally sawn lumber.

The office and wash house will be made up of used trailer units. The electrical centre and chemical storage will be housed in used cargo containers.

The plant is to be located at the original Siscoe Metals No.2 mill area at the north end of Percy Lake. Suitable foundation conditions are known to exist at this site and it is readily accessible by existing roads.

4.2 FLOWSHEETS

Mining

Feed to the mill is slurried in a portable slurry station located in the former tailing area. A 3 cu.yd. front end loader picks up the tailings and dumps them on a 1" grizzly screen over a large pump box equipped with water sprays and a hosing station. A 5 x 4 slurry pump then transfers the slurry to a permanent pump station located adjacent to the area where mining for the 8 month season is to be done. In addition, two 2 1/2" Galligher vertical slurry pumps are used to drain the area and to recover, by monitoring, feed material which may be inaccessible to the front end loader.

Milling

Slurry from the mining operation will be pumped at approximately 40% solids by a 5 x 4 slurry pump, to a 22′ dia. x 24′ high agitated surge tank, sized to hold approximately 4 hours of feed material. Feed is withdrawn from this tank at a constant rate of 45 t/h of solids and pumped with a 5 x 4 slurry pump to a cluster of 3′ - 6" dia. densifying cyclones. Cyclone underflow passes by gravity to two skid mounted 7′ dia. x 10′ ball mills in parallel powered by 250 H.P. electric motors. Mill discharge slurry is pumped with a 5 x 4 slurry pump to join the cyclone overflow stream as feed to 6 - 31′ dia. x 32′ high leach agitators. Each agitator is powered by a 40 H.P. electric motor.

After 48 hours of leaching the slurry is pumped with a 5 x 4 slurry pump to a pair of first stage 11 1/2′ dia. x 14′ long drum filters operating in parallel to remove pregnant solution from the solids. Filter cake from the first stage is repulped to 50% solids and filtered a second time with the identical sizes of feed pump and filters used in the first stage. Cake from the second stage filter is repulped with water from the reclaim system and pumped to a new tailings area with a 5 x 4 slurry pump. Pregnant solution from all four filters is recovered in 2 receivers and pumped by two - 4 x 3 filtrate pumps to a 22′ dia. x 24′ high unclarified pregnant solution tank. Vacuum for the filters is provided by two 3000 ACFM liquid ring vacuum pumps powered by 200 H.P. electric drive motors.

Silver in the pregnant solution is recovered by zinc dust precipitation in a Merrill-Crowe system. The unclarified pregnant solution is pumped by a 6 x 4 solution pump to two 5′ x 8′ x 30 leaf clarifiers. Leafs for the clarifiers are acid washed and precoated in three 9′ x 6′ x 1 1/2′ tanks prior to use in the clarifier tanks. Clarified pregnant solution is treated in a 4′ dia. x 13′ Merrill-Crowe tank. A wetting cone is used to add zinc dust, lead acetate and precoat slurry to the pregnant solution in the filter feed pump inlet line. Precipitate is recovered in a 42" x 40 leaf

Perrin filter press with an air/hydraulic closing mechanism. Filtrate is collected in a 22' dia. x 24' high barren solution tank while the cake falls by gravity into metal drums for transport to the refinery. Barren solution is used for filter wash and make up water in the feed slurry operation.

Reagent Preparation and Addition

Control of alkalinity is by addition of lime to the head of the leach circuit. Lime is added from a loop system from a 6' diameter by 6' high agitated slurry tank. Quicklime is received in bulk trucks and unloaded pneumatically into a 60 ton silo. A screw feeder on the bottom of the silo feeds the agitated stock tank where recycled barren is added to slurry the quicklime.

Sodium cyanide which is used in the leach circuit is received in one ton bags. It is prepared for use in the circuit by mixing with barren solution in 6' diameter by 6' high agitated tank and then transferred to a 7' diameter by 7' diameter stock tank. The solution strength is 15% NaCN.

Hydrochloric acid which is added to the ball mill discharge pump box is received as 36% acid in 55 gallon drums. It is prepared for use in the circuit by transferring the concentrated acid to a 5' diameter by 5' high tank by barrel pump. It is diluted to a 1% solution by adding barren solution and mixed with a circulation pump which also serves as the addition pump.

Precoat which is used for the Leaf clarifiers and the Perrin press is received in 100 pound bags. It is prepared for use by adding the precoat to a 4' diameter by 4' high tank agitated with a rim mounted agitator. It is slurried with barren and added to the clarifiers with a 2 x 2 SRL pump.

Lead acetate and zinc dust are received in bags and fed to the Merrill-Crowe system using variable speed reagent feeders.

4.3 CONTINUING METALLURGICAL TESTWORK

Witteck Development Inc. have carried out metallurgical testwork and head assays. Witteck's report entitled Metallurgical Testwork on Sandy K Tailings, Phase I Report, April 16, 1987 is appended hereto as Appendix B.

Kilborn Limited have reviewed the metallurgical test results. The test data received to date indicate the filtering tests were inconclusive. It was noted that the tests were run on samples without the use of flocculant. The correct flocculant will improve filtering rates dramatically. Additional filtration testwork is continuing with flocculant addition on a composite sample.

4.4 ENVIRONMENTAL CONTROLS

4.4.1 Plantsite

The leach tank area will be contained within a dyked tank farm. The area will be underlain with a 35 mil PVC impervious membrane sandwiched between sand cushion materials. The mill area will also be founded on an impervious membrane system. The final surface will be paved with HL4 asphalt pavement or concrete to protect the membrane system from damage. Areas around tanks and within the mill will be sloped to sumps to enable recovery of spilled materials by returning to mill circuits.

4.4.2 Tailings Management and Water Reclaim

The tailings reclaim plan is described in detail in Appendix C of this report. The general arrangement of the tailings is shown on Drawing No. 100-30-001. In general, the tailings will be systematically reclaimed and redeposited, working from the south end of the tailings area to the north. A reclaim water pond will be maintained within the tailings area to supply water for tailings recovery and slurry make-up. The reclaim pond will be moved from time to time to maintain the pond in close proximity to the slurry station.

The existing tailings deposit has been placed over the height of land to the north. The plan for remilling and redeposition includes placement of the tails totally within the Sandy K mine property watershed to avoid potential seepages to adjoining properties. The redeposition of tailings would be carried out in such a manner that all runoff from the plant and tailings area would flow to the south. A large polishing pond will be located at the south end of the property where a natural basin exists. Excess runoff would flow south east to Miller Creek thence north via Everett Lake to the Montreal River.

SENES Consultants Limited have carried out an Environmental Overview of the Sandy "K" Tailings Remilling Project which is appended hereto as Appendix D. SENES Consultants discuss existing environmental conditions and environmental aspects of the project. Studies by SENES indicate that seasonal discharges of water from the polishing pond will be required to accomplish natural degradation of cyanide. SENES state that "there is a potential that arsenic removal may be required. This would be accomplished with ferric chloride addition to the recycle pond overflow." Treated water from the recycle pond would age in the final polishing pond prior to discharging.

5.0 CAPITAL COSTS

5.1 BASIS FOR THE CAPITAL COST ESTIMATE

The figures derived for the Capital Cost Estimate were based on the following:

.01 This Capital Cost Estimate is based on the Process Flowsheets and General Arrangement drawings developed by Kilborn Limited following review of available laboratory testwork and subsequent meetings with Canadian Lencourt Mines Ltd.

.02 The concepts covered by this Capital Cost Estimate are:

- The project is to be economical and for summer use only. Service platforms stairs and handrails are to be wood construction from local lumber.
- Based on 8 operating months of the year.
- Supply of portable power supply unit to slurry/reclaim module.

.03 The capital costs are reported in 2nd Quarter 1987 Canadian dollars.

.04 The estimate is based on the use of local non union Labour, on a single shift construction program (five days a week, eight hours a day, forty hours a week) undertaken by Lencourt Mines direct hire work force supervised by the Lencourt operating staff.

.05 An average rate of $20.00 per hour was used for all direct hire workers. Mechanical, Electrical and Instrumentation.

.06 Certain major items of mechanical equipment have been quoted by Minpro Ltd. as reconditioned as of April 13, 1987. These quoted prices have been used for this estimate after allowing for freight cost to site. the remaining items of equipment have been priced as new. These used items included major equipment such as grinding mills agitator mechanisms and filters.

.07 The following items have been excluded from this estimate:

- Mobile equipment
- Initial charges, reagents and fills
- Incoming electrical power line
- Bullion furnace
- Project Insurances
- Working capital
- Spare parts and inventory
- Financing and Interest charges during construction
- Engineering, procurement and construction management charges
- Escalation
- Owners costs
- Contingency

5.2 CAPITAL COST SUMMARY

The capital cost summary sheets are appended hereto as Appendix F. The capital costs are estimated to be $4,150,000 as summarized in Table 5.1.

TABLE 5.1

CAPITAL COST SUMMARY

Area	Capital Cost $ x 1000
Site Work and Tailings	334
Buildings and Structures	409
Building Services	52
Mechanical Equipment	2,215
Process Piping	570
Electrical	250
Instrumentation	50
Total Direct Cost	$3,880
Construction Indirects	270
Total Estimated Cost	$4,150

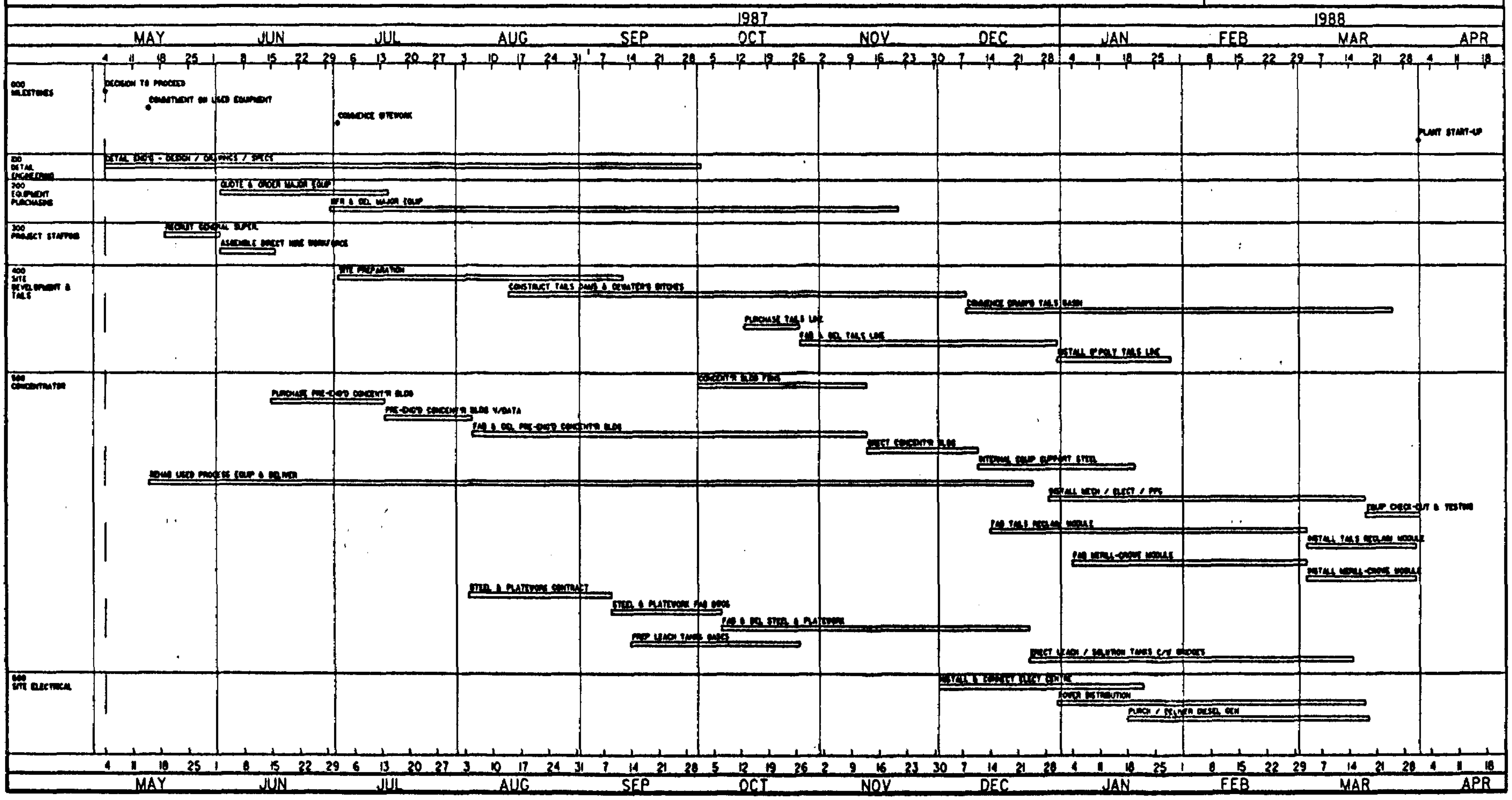
The following two cost items have been identified by the Client, Bullion Refinery and Engineering Design. The Client has carried allowances of $50,000 and $300,000 for refinery and engineering costs, bringing the capital cost to $4.5 million excluding working capital.

5.3 SCHEDULE

The following summary schedule reflects the proposed sequence of events for startup, April 1, 1988.

CANADIAN LENCOURT MINES LIMITED
SANDY K SILVER PROJECT
PROJECT SUMMARY SCHEDULE

LEGEND
EARLIEST START
EARLIEST FINISH
LATEST FINISH
ACTUAL PROGRESS
MILESTONE ◇



6.0 SUMMARY OPERATING COSTS

6.1 BASIS OF COST CALCULATION

Operating costs for the Sandy K operation have been calculated based on 8 months of operations and 4 months of shutdown during the cold winter months. The details of the operating costs are included in Appendix H.

6.2 OPERATING COST SUMMARY

The annual operating costs will be $1,255,000. These costs are summarized in Table 6.1.

TABLE 6.1

ANNUAL OPERATING COSTS

Item	Annual Cost $
Labour	555,000
Power	265,000
Reagents and Supplies	345,000
Office Rental	5,000
Environmental Sampling etc.	30,000
Property Taxes	By Owner
Insurance	50,000
Surface Transport (1 Vehicle)	5,000
TOTAL OPERATING COSTS	1,255,000

7.0 PROJECT EVALUATION

7.1 SUMMARY

A series of financial projections have been made based on the capital and operating costs developed in this report and for a range of silver prices. The significant operation parameters include the annual treatment of 225,000 tons of tailings having an average grade of 2 ounces of silver per ton in the first year, 1.75 ounces in the second and third, 1.50 ounces in the fourth and fifth, and 1.00 ounce thereafter.

The following variables were applied in determining the financial projections:

.1 Silver Price: 6, 8, 10 and 12 dollars U.S. per troy ounce.

.2 Capital Cost: 4.2 million and 4.7 million dollars Cdn. (This brackets the estimate of 4.5 million contained in this report.)

.3 Operating Cost: 1.3 million dollars which is 100,000 more than the estimate in this report and allows for Owner's costs.

.4 Refining Charge: A charge of 22 cents per ounce of silver less one percent of the silver delivered to the refiner.

.5 Working Capital: An allowance of 300,000 dollars to cover initial salaries and inventories. The latter will be minimal since the property is ideally situated to operate on a "just-in-time" supply basis.

.6 Royalties: Provision has been made for Canadian Lencourt to fulfill its obligations to Sandy K Mines and Siscoe Metals of Ontario Limited under the respective royalty agreements between these parties.

.7 Taxes and Duties: The various taxes and mining duties have been calculated in accordance with current provincial and federal regulations.

The results of these projections are summarized in Table 7.1 which shows the rate of return and the present value of the net cash flow to Canadian Lencourt at 15 percent.

TABLE 7.1

SENSITIVITY TO CAPITAL COST AND PRICE OF SILVER

Silver Price $U.S.	CAPITAL COST $4.2 Million Cdn.		CAPITAL COST $4.7 Million Cdn.	
	$\$4.2 \times 10^6$ Cdn Capital Cost		4.7×10^6 Cdn Capital Cost	
	Rate of Return	Present Value	Rate of Return	Present Value
6	10.6	(323,000)	7.8	(618,000)
8	29.3	991,000	24.7	767,000
10	42.9	1,717,000	37.3	1,586,000
12	56.4	2,373,000	49.3	2,257,000

The detailed financial projection based on a silver price of 8 dollars U.S. and a capital cost of 4.7 million Cdn. is presented in Table 7.2. The remainder of the financial projections can be found in Appendix F.

MP - 11:25 AM MON., 4 MAY , 1987

Silver at $US 6.00/oz
$4,200,000 Capital Costs

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
REVENUE									
Ton of Ore Milled	-	225,000	225,000	225,000	225,000	225,000	225,000	225,000	1,575,000
Grade (oz/ton)	-	2.0	1.8	1.8	1.5	1.5	1.0	1.0	1.5
Recovery (%)	-	87.0	87.0	87.0	87.0	87.0	87.0	87.0	87.0
Production (ozs)	-	391,500	342,563	342,563	293,625	293,625	195,750	195,750	2,055,375
CDN. Revenue @ 8.75	-	3,132,001	2,740,501	2,740,501	2,349,001	2,349,001	1,566,000	1,566,000	16,443,002
OPERATING COSTS									
Mining & Milling	-	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	9,100,000
Refining Charges	-	116,589	102,015	102,015	87,442	87,442	58,294	58,294	612,091
	-	1,416,589	1,402,015	1,402,015	1,387,442	1,387,442	1,358,294	1,358,294	9,712,092
NET OPERATING PROFIT	-	1,715,412	1,338,486	1,338,486	961,559	961,559	207,706	207,706	6,730,914

CASH FLOW TO CLE

	1987	1988	1989	1990	1991	1992	1993	1994	Total
Net Operating Profit	-	1,715,412	1,338,486	1,338,486	961,559	961,559	207,706	207,706	6,730,914
Less: Capital Investment	2,800,000	1,400,000	-	-	-	-	-	-	4,200,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
NPI to Siscoe	-	-	-	-	-	-	-	-	-
Royalty to Sandy K	-	-	-	-	316,191	359,238	83,082	403,082	1,161,593
Federal Income Tax	-	-	-	38,957	130,687	121,970	25,236	100,000	416,851
Provincial Income Tax	-	-	-	18,597	62,386	58,224	12,047	50,000	201,254
Provincial Mining Duty	-	-	-	-	63,465	63,465	-	-	126,930
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Net Cash Flow to CLE	(2,800,000)	15,412	1,338,486	1,280,931	388,831	358,662	87,340	454,624	1,124,285
Accumulated Net Cash Flow	(2,800,000)	(2,784,589)	(1,446,103)	(165,172)	223,659	582,321	669,661	1,124,285	1,124,285
Rate of Return to CLE	-	-	-	-	2.9	6.8	7.5	10.6	10.6
Present Value of NCF at 10%	(2,800,000)	(2,785,990)	(1,679,803)	(717,421)	(451,844)	(229,143)	(179,842)	53,452	53,452
Present Value of NCF at 15%	(2,800,000)	(2,786,599)	(1,774,512)	(932,279)	(709,964)	(531,646)	(493,886)	(322,976)	(322,976)
Present Value of NCF at 20%	(2,800,000)	(2,787,157)	(1,857,653)	(1,116,374)	(928,860)	(784,721)	(755,471)	(628,594)	(628,594)

MP - 11:25 AM MON., 4 MAY , 1987

Silver at $US 6.00/oz
$4,200,000 Capital Costs

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
SISCOE ROYALTY CALCULATION									
Opening Unrecovered Costs	2,000,000	4,800,001	5,597,091	5,071,106	4,545,121	4,459,528	4,373,935	4,978,730	-
Plus: New Capital Investment	2,800,000	1,700,000	-	-	-	-	-	-	4,500,001
Operating Costs	-	1,416,589	1,402,015	1,402,015	1,387,442	1,387,442	1,358,294	1,358,294	9,712,092
Depreciation	-	812,500	812,500	812,500	812,500	812,500	812,500	812,500	5,687,501
Prov. Mining Duties	-	-	-	-	63,465	63,465	-	-	126,930
Less: Total Revenue	-	3,132,001	2,740,501	2,740,501	2,349,001	2,349,001	1,566,000	1,566,000	16,443,002
Closing Unrecovered Costs	4,800,001	5,597,091	5,071,106	4,545,121	4,459,528	4,373,935	4,978,730	5,583,525	-
Siscoe NPI	-	-	-	-	-	-	-	-	-
CALCULATION OF CASH DISTRIBUTION									
Operating Profit	-	1,715,412	1,338,486	1,338,486	961,559	961,559	207,706	207,706	6,730,914
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Less: NPI to Siscoe	-	-	-	-	-	-	-	-	-
Prov. Mining Duties	-	-	-	-	63,465	63,465	-	-	126,930
Net Cash for Distribution	-	1,715,412	1,338,486	1,338,486	898,094	898,094	207,706	1,007,706	7,403,984
CLE's Recovery of Capital	-	1,715,412	1,338,486	1,338,486	107,618	-	-	-	4,500,002
CLE's Premium Account	-	-	-	-	158,095	179,619	41,541	201,541	580,797
CLE's Payout	-	-	-	-	316,191	359,238	83,082	403,082	1,161,593
Total Payout to CLE	-	1,715,412	1,338,486	1,338,486	581,904	538,856	124,624	604,624	6,242,392
Payout to Sandy K	-	-	-	-	316,191	359,238	83,082	403,082	1,161,593
CLE'S NET CASH FLOW									
Total Payout to CLE	-	1,715,412	1,338,486	1,338,486	581,904	538,856	124,624	604,624	6,242,392
Less: Capital Investment	2,800,000	1,400,000	-	-	-	-	-	-	4,200,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
Federal Income Tax	-	-	-	38,957	130,687	121,970	25,236	100,000	416,851
Prov. Income Tax	-	-	-	18,597	62,386	58,224	12,047	50,000	201,254
Net Cash Flow to CLE	(2,800,000)	15,412	1,338,486	1,280,931	388,831	358,662	87,340	454,624	1,124,286

TABLE 7.2

FINANCIAL ANALYSIS

CANADIAN LENCOURT MINES LTD.
SANDY K PROJECT
BY
MARKHAM DATA INC.

BASIS: Silver at $U.S. 8.00/oz.
CAPITAL COSTS $4,700,000

Summary Sheets follow: Page 1
Page 2

MP - 11:27 AM MON., 4 MAY , 1987

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 8.00/oz
$4,700,000 Capital Costs

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
REVENUE									
Ton of Ore Milled	-	225,000	225,000	225,000	225,000	225,000	225,000	225,000	1,575,000
Grade (oz/ton)	-	2.0	1.8	1.8	1.5	1.5	1.0	1.0	1.5
Recovery (%)	-	87.0	87.0	87.0	87.0	87.0	87.0	87.0	87.0
Production (ozs)	-	391,500	342,563	342,563	293,625	293,625	195,750	195,750	2,055,375
Revenue	-	4,176,001	3,654,001	3,654,001	3,132,001	3,132,001	2,088,000	2,088,000	21,924,004
OPERATING COSTS									
Mining & Milling	-	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	9,100,000
Refining Charges	-	127,029	111,150	111,150	95,272	95,272	63,514	63,514	666,901
	-	1,427,029	1,411,150	1,411,150	1,395,272	1,395,272	1,363,514	1,363,514	9,766,902
NET OPERATING PROFIT	-	2,748,972	2,242,851	2,242,851	1,736,729	1,736,729	724,486	724,486	12,157,106

CASH FLOW TO CLE

	1987	1988	1989	1990	1991	1992	1993	1994	Total
Net Operating Profit	-	2,748,972	2,242,851	2,242,851	1,736,729	1,736,729	724,486	724,486	12,157,106
Less: Capital Investment	3,200,000	1,500,000	-	-	-	-	-	-	4,700,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
NPI to Siscoe	-	-	-	-	-	-	-	-	-
Royalty to Sandy K	-	-	-	781,355	616,594	616,594	280,529	600,529	2,895,603
Federal Income Tax	-	-	59,094	295,953	226,827	226,827	89,901	125,101	1,023,704
Provincial Income Tax	-	-	28,210	141,278	108,280	108,280	42,916	61,982	490,945
Provincial Mining Duty	-	-	-	281,285	195,244	195,244	23,163	23,163	718,098
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Net Cash Flow to CLE	(3,200,000)	948,972	2,155,547	742,980	589,784	589,784	287,977	713,710	2,828,755
Accumulated Net Cash Flow	(3,200,000)	(2,251,028)	(95,481)	647,499	1,237,283	1,827,067	2,115,044	2,828,755	2,828,755
Rate of Return to CLE	-	-	-	10.0	16.4	20.6	22.1	24.7	24.7
Present Value of NCF at 10%	(3,200,000)	(2,337,298)	(555,854)	2,358	405,188	771,398	933,953	1,300,200	1,300,200
Present Value of NCF at 15%	(3,200,000)	(2,374,807)	(744,905)	(256,383)	80,828	374,054	498,555	766,865	766,865
Present Value of NCF at 20%	(3,200,000)	(2,409,190)	(912,283)	(482,318)	(197,892)	39,129	135,571	334,755	334,755

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 8.00/oz
$4,700,000 Capital Costs

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
SISCOE ROYALTY CALCULATION									
Opening Unrecovered Costs	2,000,000	5,200,001	5,126,031	3,758,181	2,671,616	2,005,131	1,338,646	1,512,322	-
Plus: New Capital Investment	3,200,000	1,800,000	-	-	-	-	-	-	5,000,001
Operating Costs	-	1,427,029	1,411,150	1,411,150	1,395,272	1,395,272	1,363,514	1,363,514	9,766,902
Depreciation	-	875,000	875,000	875,000	875,000	875,000	875,000	875,000	6,125,001
Prov. Mining Duties	-	-	-	281,285	195,244	195,244	23,163	23,163	718,098
Less: Total Revenue	-	4,176,001	3,654,001	3,654,001	3,132,001	3,132,001	2,088,000	2,088,000	21,924,004
Closing Unrecovered Costs	5,200,001	5,126,031	3,758,181	2,671,616	2,005,131	1,338,646	1,512,322	1,685,999	-
Siscoe NPI	-	-	-	-	-	-	-	-	-
CALCULATION OF CASH DISTRIBUTION									
Operating Profit	-	2,748,972	2,242,851	2,242,851	1,736,729	1,736,729	724,486	724,486	12,157,106
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Less: NPI to Siscoe	-	-	-	-	-	-	-	-	-
Prov. Mining Duties	-	-	-	281,285	195,244	195,244	23,163	23,163	718,098
Net Cash for Distribution	-	2,748,972	2,242,851	1,961,566	1,541,485	1,541,485	701,323	1,501,324	12,239,008
CLE's Recovery of Capital	-	2,748,972	2,242,851	8,178	-	-	-	-	5,000,002
CLE's Premium Account	-	-	-	390,678	308,297	308,297	140,265	300,265	1,447,801
CLE's Payout	-	-	-	781,355	616,594	616,594	280,529	600,529	2,895,603
Total Payout to CLE	-	2,748,972	2,242,851	1,180,211	924,891	924,891	420,794	900,794	9,343,404
Payout to Sandy K	-	-	-	781,355	616,594	616,594	280,529	600,529	2,895,603
CLE'S NET CASH FLOW									
Total Payout to CLE	-	2,748,972	2,242,851	1,180,211	924,891	924,891	420,794	900,794	9,343,404
Less: Capital Investment	3,200,000	1,500,000	-	-	-	-	-	-	4,700,001
Working Capital	-	300,000			-	-	-	-	300,000
Federal Income Tax	-	-	59,094	295,953	226,827	226,827	89,901	125,101	1,023,704
Prov. Income Tax	-	-	28,210	141,278	108,280	108,280	42,916	61,982	490,945
Net Cash Flow to CLE	(3,200,000)	948,972	2,155,547	742,980	589,784	589,784	287,977	713,710	2,828,755

TABLE 7.3

ADDENDUM
TO
FINANCIAL ANALYSIS
SANDY K PROJECT
TO REFLECT

1. Purchase of Sandy K Royalty
2. Three-Year Exemption from Ontario Mining Tax

Prepared by:
Watts, Griffis and McOuat Limited

July 30, 1987

Case 'C' - 100% Lencourt
Canadian Dollars

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver Price: $US 8.00/oz
Capital Costs $4,700,000
Property Payment: $900,000
Ontario Tax Holiday: 3 Years

	1987	1988	1989	1990	1991	1992	1993	1994	Total
REVENUE									
Ton of Ore Milled	-	225,000	225,000	225,000	225,000	225,000	225,000	225,000	1,575,000
Grade (oz/ton)	-	2.0	1.8	1.8	1.5	1.5	1.0	1.0	1.5
Recovery (%)	-	87.0	87.0	87.0	87.0	87.0	87.0	87.0	87.0
Production (ozs)	-	391,500	342,562	342,562	293,625	293,625	195,750	195,750	2,055,375
Less: Sandy K Share of Silver (ozs	-	-	-	-	-	-	-	-	-
Net Silver to CLE (ozs)	-	391,500	342,562	342,562	293,625	293,625	195,750	195,750	2,055,375
Revenue	-	4,176,000	3,654,000	3,654,000	3,132,000	3,132,000	2,088,000	2,088,000	21,924,004
OPERATING COSTS									
Mining & Milling	-	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	9,100,000
Refining Charges	-	127,029	111,150	111,150	95,272	95,272	63,514	63,514	666,901
	-	1,427,029	1,411,150	1,411,150	1,395,271	1,395,271	1,363,514	1,363,514	9,766,902
NET OPERATING PROFIT	-	2,748,972	2,242,850	2,242,850	1,736,729	1,736,729	724,486	724,486	12,157,106

CASH FLOW TO CLE

	1987	1988	1989	1990	1991	1992	1993	1994	Total
Net Operating Profit	-	2,748,972	2,242,850	2,242,850	1,736,729	1,736,729	724,486	724,486	12,157,106
Less: Capital Investment	3,200,000	1,500,000	-	-	-	-	-	-	4,700,000
Working Capital	-	300,000	-	-	-	-	-	-	300,000
Payment to Sandy K	900,000	-	-	-	-	-	-	-	900,000
NPI to Siscoe	-	-	-	-	-	-	-	-	-
Federal Income Tax	-	-	-	393,533	332,934	338,560	137,519	240,276	1,442,823
Provincial Income Tax	-	-	-	187,859	158,932	161,617	65,647	116,963	691,018
Provincial Mining Duty	-	-	-	-	172,294	172,294	213	7,863	352,663
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Net Cash Flow to CLE	(4,100,000)	948,972	2,242,850	1,661,458	1,072,569	1,064,257	521,107	1,159,384	4,570,599
Accumulated Net Cash Flow	(4,100,000)	(3,151,028)	(908,177)	753,280	1,825,850	2,890,107	3,411,214	4,570,599	4,570,599
Rate of Return to CLE	-	-	-	8.3	16.5	21.7	23.5	26.2	26.2
Present Value of NCF at 10%	(4,100,000)	(3,237,298)	(1,383,702)	(135,424)	597,155	1,257,976	1,552,127	2,147,075	2,147,075
Present Value of NCF at 15%	(4,100,000)	(3,274,806)	(1,578,890)	(486,455)	126,790	655,913	881,202	1,317,057	1,317,057
Present Value of NCF at 20%	(4,100,000)	(3,309,190)	(1,751,655)	(790,164)	(272,914)	154,787	329,305	652,868	652,868

July 30, 1987

Silver Price: $US 8.00/oz
Capital Costs $4,700,000
Property Payment: $900,000
Ontario Tax Holiday: 3 Years

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Case 'C' - 100% Lencourt
Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
SISCOE ROYALTY CALCULATION									
Opening Unrecovered Costs	2,000,000	5,200,000	5,126,030	3,758,180	2,390,330	1,700,895	1,011,460	1,162,187	-
Plus: New Capital Investment	3,200,000	1,800,000	-	-	-	-	-	-	5,000,000
Operating Costs	-	1,427,029	1,411,150	1,411,150	1,395,271	1,395,271	1,363,514	1,363,514	9,766,902
Depreciation	-	875,000	875,000	875,000	875,000	875,000	875,000	875,000	6,125,000
Prov. Mining Duties	-	-	-	-	172,294	172,294	213	7,863	352,663
Less: Total Revenue	-	4,176,000	3,654,000	3,654,000	3,132,000	3,132,000	2,088,000	2,088,000	21,924,004
Closing Unrecovered Costs	5,200,000	5,126,030	3,758,180	2,390,330	1,700,895	1,011,460	1,162,187	1,320,564	-
Siscoe NPI	-	-	-	-	-	-	-	-	-
CALCULATION OF CASH DISTRIBUTION									
Operating Profit	-	2,748,972	2,242,850	2,242,850	1,736,729	1,736,729	724,486	724,486	12,157,106
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Less: NPI to Siscoe	-	-	-	-	-	-	-	-	-
Prov. Mining Duties	-	-	-	-	172,294	172,294	213	7,863	352,663
Net Cash for Distribution	-	2,748,972	2,242,850	2,242,850	1,564,435	1,564,435	724,273	1,516,623	12,604,444
CLE's Recovery of Capital	-	2,748,972	2,242,850	8,177	-	-	-	-	5,000,001
CLE's Payout	-	-	-	2,234,673	1,564,435	1,564,435	724,273	1,516,624	7,604,442
Total Payout to CLE	-	2,748,972	2,242,850	2,242,850	1,564,435	1,564,435	724,273	1,516,624	12,604,444
Payout to Sandy K	-	-	-	-	-	-	-	-	-
CLE'S NET CASH FLOW									
Total Payout to CLE	-	2,748,972	2,242,850	2,242,850	1,564,435	1,564,435	724,273	1,516,624	12,604,444
Less: Capital Investment	3,200,000	1,500,000	-	-	-	-	-	-	4,700,000
Working Capital	-	300,000	-	-	-	-	-	-	300,000
Federal Income Tax	-	-	-	393,533	332,934	338,560	137,519	240,276	1,442,823
Prov. Income Tax	-	-	-	187,859	158,932	161,617	65,647	116,963	691,018
Net Cash Flow to CLE	(3,200,000)	948,972	2,242,850	1,661,458	1,072,569	1,064,257	521,107	1,159,385	5,470,603

July 30, 1987

LENCOURT LIMITED - SANDY K PROJECT

Analysis of Effect of
Purchase of Sandy K Royalty
and New 3-Year Ontario Mining Tax Holiday

Silver Price US $	Situation As Per Feasibility Study		Pro Forma After Buy-Out of Sandy K Royalty		Pro Forma After Royalty Buy-Out and Tax Holiday	
$4.2 million capital costs	Present value @ 15%	10%	Present value @ 15%	10%	Present value @ 15%	10%
$ 6	(0.3)	0.05	(0.6)	(0.1)	(0.6)	(0.1)
8	1.0	1.5	1.5	2.3	1.7	2.5
10	1.7	2.3	2.8	3.7	3.2	4.2
12	2.4	3.0	4.2	5.2	4.8	5.9
	ROR		ROR		ROR	
$ 6	10.6%		9.1%		9.1%	
8	29.3		29.3		30.8	
10	42.9		43.8		48.0	
12	56.4		58.7		65.0	
$4.7 million capital costs	Present value @ 15%	10%	Present value @ 15%	10%	Present value @ 15%	10%
$ 6	(0.6)	(0.2)	(1.0)	(0.5)	(1.0)	(0.5)
8	0.8	1.3	1.1	2.0	1.3	2.1
10	1.6	2.2	2.6	3.6	3.0	4.0
12	2.3	2.9	3.9	5.0	4.6	5.8
	ROR		ROR		ROR	
$ 6	7.8%		6.3%		6.3%	
8	24.7		24.8		26.2	
10	37.3		39.2		42.4	
12	49.3		52.3		58.6	

NOTE: Present values are in $C millions

July 30, 1987

APPENDIX A

Field Investigation
And
Reserve Calculations

REPORT ON SAMPLING

SANDY K TAILINGS

FOR

CANADIAN LENCOURT MINES LTD.

By

Robert L.V. Ekstrom

BA.Sc., P.Eng.

REPORT ON SAMPLING

SANDY K TAILINGS

For

CANADIAN LENCOURT MINES LTD.

By

Robert L.V. Ekstrom

BA.Sc., P. Eng.

REPORT ON SAMPLING

SANDY K TAILINGS

1. INTRODUCTION

During 1986 and 1987 Canadian Lencourt Mines Ltd. by an agreement with Sandy K Mines Ltd. undertook a program of further sampling of the silver tailings on the old Siscoe Metals property near Gowganda, Ontario.

Some previous sampling of these tailings had been carried out by Watts Griffis and McOuat Ltd. in 1980-1981. The results of that work demonstrated a proven, probable and possible reserve of 1.7 million tons having an average grade of 1.45 ounces of silver per ton of tailings.

The aim of this latest sampling program has been to put all of the reserve in the proven category. To do this, the previous work was reviewed and its recommendations for further sampling where initiated where possible and modified where necessary.

In the earlier program by WGM, detailed sampling was restricted, because of water conditions and the method employed, to the upper 15 feet of the tailings pile. The current program, by the use of a different method of sampling has concentrated on the deeper zones. In addition some infill drilling was done on the fringe areas of the pile and in areas under water.

Combined with the work done in 1980-1981, all of the old tailings have been sampled and all of the reserves can now be placed in the proven category.

2. PREVIOUS WORK

The work by WGM in 1980-81 was part of a larger study of this property carried out for Sandy K. The following description of the work and methods used are extracted from the report on that work.

A sampling program was carried out in four stages:

1. Hand sampling to a maximum depth of 15 feet of the main tailings pile located on claim RSC84 and of the tailings filling what was known as Percy Lake. This first phase consisted of holes 100 feet apart along the length of the pile.

2. Hand sampling to 15-foot depth on a 100-foot grid of the main tailings pile. This grid was later in-filled to 50 feet.

3. Hand sampling of small tailings zones that represent washout material from the main pile. These washouts are as far as 2,000 feet away and downstream from the main pile. These small zones were sampled to the bottom of the tailings on a 50-foot grid.

4. Drive-pipe sampling of the main tailings pile to reach the glacial till or bedrock underlying the tailings.

Hand sampling or hand augering was mainly carried out with a manually operated agricultural soil sampler. The sampler is an open 1-1/4" diameter, 18" long tube with a cutting edge that permits the collection of compacted soil as a core 1 inch in diameter. The sampler is lowered by pusing on a handle attached to a 30" long extension rod. Sampling was carried down to a depth of 15 feet, each sample representing a 5-foot interval. Cave-in was minimal above the water table. Poor recovery and contamination by caving of the walls occurred below the water table. A 1-1/4" spiral hand auger with a 2-1/2" pitch was used occasionally where tailings were somewhat clayey and dry.

The drive-pipe technique was used for sampling of tailings in excess of 15-foot thickness and for obtaining large-sized samples for silver recovery testing. We used a 3-1/2" diameter 5-foot long barrel sampler equipped with a flap retainer for material above the water table; a 2-1/2" diameter 2-foot long split spoon was used for sampling inside NW casing for material below the water table. The sampler and casing were driven by a 140 lb. hammer dropped 3 feet and operated by a motorized hoist. Drive pipe sampling was conducted to the bottom of the tailings pile, that is until glacial till material was retrieved from the hole or refusal on bedrock obtained. A maximum tailings depth of 54 feet was reached in one of 27 holes.

Manually 545 holes were sunk for cumulative length of 5,160 feet, and 27 drive-pipe holes for 629 feet. A total of 1,236 samples were collected and sent for fire assay, most samples taken at 5-foot drilling intervals.

3. TAILINGS SAMPLINGS 1986-87

Although effective, the drive-pipe technique was considered to be time-consuming and expensive. Consequently, alternatives were considered including a vibrating type of drill. Test drilling was carried out on the Sandy K silver tailing during August 1986 using the Wink Vibra Corer Sonic Drill. This proved successful and a full scale program was undertaken during January and February 1987 to complete the sampling.

The Vibra Corer Drill is a simple, efficient, lightweight tool which recovers excellent samples in unconsolidated material. In certain types of ground the drill recovers a core of undisturbed material in which bedding structures, etc. can be seen. The drill rods are vibrated into the ground by the Vibra Head which weighs 12 kg. The vibration is created by an eccentric bar in the head rotated by a flex cable powered by a four to eight horsepower gas engine. In the present program, the rods used were five foot BQ diamond drill rods (inside diameter 1.91 inches, outside diameter 2.20 inches). Although various drive shoes and core catchers are available for the Sonic drill, it was found in the present program that the drill rods with no shoe was the most effective method of sample recovery.

In the dryer, hard packed tailings above the water table, it was necessary to pull the rods every few inches and the hole remained open with no caving.

On reaching the water table, it was necessary to make certain that the rods were not plugged with the hard packed tailings. It was found that a short hard plug would not allow the wet flowable tailings into the string and the rods could be driven to the bottom without recovering any sample. To overcome this when the water table was encountered and recognized, the rods were cleaned and the hole driven to the bottom of the tailings with measurements made inside the rods each time a new rod was added to make sure a sample was being recovered. In most cases, in order to recover the sample, the rods could be plugged in the hard packed silty sand, humus and/or clay immediately beneath the tailings. Samples below the water table were often lost if this base plug was not made. This usually occurred at times if rock was encountered at the bottom of the hole.

Although recovery of sample is estimated to have been generally between 90 and 100%, poor or no recovery was noted in a few holes below the water table. In these cases it is thought that debris (sticks, twigs, small muck fragments or pebbles) was mixed in the tailings and was driven ahead of the bit and did not allow the sample to enter the rod string. These holes were redrilled up to four times with a move of two to five feet to pass through undisturbed material. Only four holes were drilled from which almost no sample was recovered (110, 117, 142 and 059). Hole 142 was thought to be totally in wet humus.

A total of 248 feet in seven holes was drilled with the Vibra Corer in August 1986. A further 2738.5 feet were drilled in 145 holes during January and February of 1987. The results of this drilling are tabulated in Appendix A to this report.

Forty-four samples were taken in the August program and analyzed by Witteck Development Inc. Four hundred and thirteen samples were taken and sent for analysis to Bell White Analytical Laboratories Ltd. in Haileybury. To check the assaying a composite sample was prepared by Witteck and assayed then and by Bell White and also by Bordar Clegg & Company Ltd. The correlation between these results were acceptable and the assays by Bell White are considered to by within generally accepted limits for silver.

4. COMPARISON WITH 1980-81 PROGRAM

To eliminate duplication of work it was decided to use as much as possible of the work done in 1980-81. In particular if the reserves in the proven category could be accepted a significant effort could be eliminated.

A comparison of the 1981 drilling and the 1986-87 drilling in the 0-15 feet range was made over the main tailings area. Using 61 of the 1986-87 holes and 146 of the 1981 holes, it was calculated that the 1986-87 values were 2.717% lower than the 1981 analyses. By inspection it was noted that two anomalous high grade patches were sampled in the 1981 program (577 holes) which averaged over 5.0 oz Ag/ton over 15.0 feet. No similar high grade patches were encountered in the 1986-87 drilling, probably because of statistical probabilities with fewer holes. If the high grade holes are disregarded, the 1986-87 average was seen to be only 0.247% low.

Because of the excellent correlation of values, it has been decided to accept the 1981 calculation of proven reserve in the 0 to minus 15 feet portion of the tailings except where new data has changed the tons to a material extent (Block 10). Drilling in untested areas has also expanded the reserve in the surface to the minus 15-foot horizon.

The remainder of the drilling on 100-foot centres has allowed the calculation of proven reserves below the minus 15-foot horizon.

5. TONNAGE FACTOR

The reserves calculated in 1980-81 were based on a tonnage factor derived from one sample. This was not considered adequate for the purposes of the current study and further sampling was carried out.

Eleven samples were taken in an attempt to define the specific gravity of the deposit. Samples were taken using the Vibra Corer Drill in drier areas and with an ice auger in low wet areas.

The average of five samples in the hard packed sand was 13.326 cu ft per ton and in the low wet slimey areas was 21.304 when the volumes were adjusted for ten percent expansion because of frost. The average of these two samples ranges from 15.95 to 17.32 cu ft per ton when the ratio of sand to wet slimes is varied from 2:1 to 1:1. Present studies do not allow a more accurate calculation of specific gravity and the 16.26 cu ft per ton factor which was used in the 1981 is accepted as realistic.

To be consistent this factor was used for the new calculation of reserves.

6. RESERVES

It was decided to use the same general method of calculating reserves as was used in 1980-81. The following is extracted from the WGM report on that work.

10.3.2 METHOD OF ESTIMATION

Reserves were calculated manually: drilled blocks were outlined on plans, areas of influence measured, areas were combined with intervals along drilled holes to obtain blocks of mineable size.

Based on the above, the reserves were divided into three categories:

a) Proven Reserves include those parts of the tailings that are within 100 feet of a drilled hole and lie between the surface and a depth of 15 feet.

b) Probable Reserves include: 1) those parts of the tailings that can be observed at surface, that are located more than 100 feet from a drilled hole and that could be extrapolated to a depth of 15 feet with confidence; and 2) those parts of the tailings deeper than 15 feet from surface which are no more from 150 feet than a drive-pipe hole.

c) Possible Reserves include: 1) those parts of the tailings extrapolated down from the 15-foot depth and that were sampled from surface to 15 feet; and 2) those parts of the tailings that are submerged and could not be sampled due to thin ice conditions during the winter.

Overall, the reserve category is determined by the number and relative spacing of drill holes.

No cutoff grade was used. Within the tailings, no samples were barren of silver, and very few had a silver content of less than 0.5 oz/ton. Low-grade tailings were included in the reserve estimate.

Table 10-1 summarizes the reserves estimate by category and by range of grade. The total reserves are based on tonnage and grade from 656 sub-blocks grouped into 91 blocks of mineable size.

Additional drilling of deep tailings and of those that are water-saturated is recommended to enhance the probable and possible reserve estimates.

(By Category)

Category	Reserves (tons)	Grade (oz Ag/ton)
Proven	1,087,000	1.48
Probable	412,000	1.35
Possible	202,000	1.52
Total	1,701,000	1.45

(In Order of Decreasing Grade)

Range of Grade (oz/ton)	No. of Blocks	Reserves (tons)	Grade (oz Ag/ton)	Cumulative Tonnage	Cumulative Grade (oz/ton)
3.00	4	80,500	3.75	80,500	3.75
2.00-2.99	3	59,400	2.44	139,900	3.19
1.75-1.99	8	137,600	1.82	277,500	2.52
1.50-1.74	12	214,300	1.60	491,800	2.12
1.25-1.49	22	399,100	1.37	890,900	1.78
1.00-1.24	33	577,800	1.15	1,468,700	1.53
0.81-0.99	9	232,600	0.93	1,701,300	1.45
Totals	91	1,701,300	1.45		

The proven reserves calculated from the 1981 program were rechecked by recalculating the area of influence for every fifth block. It was found that there was no more than 2.64 percentage difference for the combined area of the blocks averaged. A check was then made upon the reserve estimate for 1981 on the proven ore blocks and the calculations for 1981 were confirmed by these checks.

The 1987 drill holes were plotted and tonnage calculations were made using 10,000 square-foot blocks in areas of uniformly spread holes and polygons when hole spacing was irregular. Tabulated below are the total tonnages and grades from the 1987 sonic drill program added to the 1981 proven reserves.

TAILINGS RESERVE SUMMARY

	Depth (ft)	Tonnage	Average Grade (oz/t)
1981 Proven Reserves Sheets 1A, 2A, 3, 4 & 5	0-15	1,086,620	1.48
Drive-Pipe Holes Sheets 1B & 2B	below 15	140,511	1.37
1987 Proven Reserves			
Tailings Reserve Sheet 1A	0-15	62,028	1.15
Tailings Reserve Sheet 2A	0-15	38,913	1.67
Tailings Plan Sheet 1B	below 15	252,928	1.30
Tailings Plan Sheet 2B	below 15	246,023	1.41
Total (rounded tons)		1,827,000	1.43

The seven reserve sheets noted in the above tabulation accompany this report. They show the locations of all the 1981 and 1987 drilling. Reserve blocks for the 1981 and 1987 proven tonnages are tabulated in Appendix B.

Robert L.V. Ekstrom
B.A.Sc.,P.Eng.

CERTIFICATE

I, Robert L.V. Ekstrom, do hereby certify that:

(1) I am president of the private mineral exploration service firm, Canadian Oresearch Inc. of Toronto.

(2) I reside at 1 Rolph Road, Toronto, Ontario.

(3) I am a graduate of the University of Toronto with a B.A.Sc. degree in Applied Geology, 1956.

(4) I am a registered member of the Association of Professional Engineers of Ontario.

(5) I have worked in my profession in mining and exploration in geology since 1956, and have had experience with precious, base-metal and iron deposits in Canada, the USA, the United Kingdom and South America.

(6) I do not own, directly or indirectly, nor do I expect to receive any interest, directly or indirectly, in the property described in this report, or in any associated or affiliated company nor do I beneficially own, directly or indirectly, any securities of Canadian Lencourt Mines Ltd. or any associated or affiliated company.

(7) The accompanying report is based on personal supervision of the drilling and subsequent processing and reporting of results.

Robert L. V. Ekstrom
B.A.Sc., P. Eng.

APPENDIX A

TAILINGS SAMPLING TABULATION SHEETS
FOR 1987 DRILLING

TAILINGS SAMPLING TABULATION SHEET
(January - February, 1987)

Hole No.	Northing (ft)	Easting (ft)	Collar Elevation (ft)	Sampling: Sample No.	Sampling: From (ft)	Sampling: To (ft)	Sampling: Length (ft)	Assay Ag (oz/t)	Hole Average Depth (ft)	Hole Average Ag (oz/t)	Water Table Depth (ft)	Observations
004-87	12,230	12,024	4,923.4	-	0.0	1.0	1.0	-	1.0	-	0.0	Rock (muck)
007-87	13,968	11,485	4,944.4	87251	0.0	4.0	4.0	1.31				Sandy
				52	4.0	10.0	6.0	1.49			4.0	Sandy
				53	10.0	16.0	6.0	1.25	16.0	1.36		Sandy
008-87	13,644	11,515	4,937.5	72782	0.0	4.0	4.0	0.97				Dry, sandy
				83	4.0	9.0	5.0	0.81				Dry, sandy
				84	9.0	15.0	6.0	0.66				Wet, slimey
				85	15.0	21.5	6.5	1.13	21.5	0.89	3.0	Wet, slimey
009-87	13,267	11,652	4,925.7	72795	0.0	4.0	4.0	1.25				Dry, sandy
				96	4.0	10.0	6.0	1.40				Wet, slimey
				97	10.0	18.0	8.0	1.65				Wet, slimey
				98	18.0	26.0	8.0	1.66				Wet, slimey
				99	26.0	31.0	5.0	1.95				Wet, slimey
				72800	31.0	35.5	4.5	0.71	35.5	1.49	4.0	Wet, slimey
010-87	12,891	11,788	4,923.1	-	0.0	14.0	14.0	-			0.0	Water and floating humus, no core
				72905	14.0	22.0	8.0	1.24	8.0	1.24		Slimey
011-87	12,516	11,922	4,925.2	72918	0.0	4.0	4.0	1.35				Sandy
				19	4.0	10.0	6.0	0.93			4.0	Mixed
				20	10.0	15.0	5.0	0.92				Sandy
				21	15.0	20.0	5.0	1.04				Sandy
				22	20.0	25.0	5.0	0.86	25.0	1.00		Sandy
012-87	12,522	12,239	4,923.1	-	0.0	15.0					0.0	Water
				-	15.0	20.0						Soft humus, cored
				72931	20.0	25.0	5.0	0.02				Wet, slimey clay
				32	25.0	30.0	5.0	0.02	10.0	0.02		Wet, slimey clay
013-87	12,429	12,272	4,923.1	-	0.0	8.0					0.0	Water
				-	8.0	10.0						Soft humus, cored
				72930	10.0	13.0	3.0	0.02	3.0	0.02		Wet, slimey clay
014-87	12,463	12,366	4,923.1	-	0.0	3.0					0.0	Water, rock at 3.0
015-87	12,557	12,334	4,923.1	-	0.0	3.0		-		-	0.0	Water, rock at 3.0
016-87	12,614	12,206	4,923.1	-	0.0	7.0		-		-	0.0	Water, wet humus, rock at 7.0
018-87	12,581	12,112	4,923.1	-	0.0	14.0		-		-	0.0	Water, floating humus
				72939	14.0	17.0	3.0	1.41	3.0	1.41		Slimey 14-22, clay 17-22
				40	17.0	22.0	5.0	0.02				Gravel at 22.0
019-87	12,489	12,146	4,923.1	-	0.0	19.0		-				Water and floating humus
				72933	19.0	25.0	6.0	0.70	6.0	0.70	0.0	
020-87	12,362	12,084	4,923.1	-	0.0	10.0		0.0		0.0	0.0	Water and floating humus
				72929	10.0	16.0	6.0	0.66	6.0	0.66		Slimey
022-87	12,198	11,929	4,923.2	-	0.0	1.0		-		-	0.0	Water, rock bottom
023-87	12,283	11,911	4,923.1	-	0.0	2.0		-		-	0.0	Water, rock bottom
024-87	12,327	11,990	4,923.5	-	0.0	2.0		-			0.0	Water
				72928	2.0	7.0	5.0	1.66	5.0	1.66		Slimey
025-87	12,421	11,956	4,923.5	72923	0.0	5.0	5.0	0.56			0.0	Sandy
				24	5.0	13.0	8.0	0.64				Sandy

TAILINGS SAMPLING TABULATION SHEET
(January - February, 1987)

Hole No.	Northing (ft)	Easting (ft)	Collar Elevation (ft)	Sampling: Sample No.	Sampling: From (ft)	Sampling: To (ft)	Sampling: Length (ft)	Assay: Ag (oz/t)	Hole Average: Depth (ft)	Hole Average: Ag (oz/t)	Water Table Depth (ft)	Observations
				25	13.0	18.0	5.0	0.60				Mixed
				26	18.0	24.0	6.0	0.54				Mixed
				27	24.0	31.0	7.0	1.10	31.0	0.70		Slimey
026-87	12,455	12,050	4,923.2	-	0.0	10.0		-			0.0	Water and floating humus
				72934	10.0	15.0	5.0	0.76				Slimey
				35	15.0	20.0	5.0	0.85	10.0	0.80		Slimey, humus at 20.0
					20.0	38.0						Hole to 38 but no sample
027-87	12,548	12,016	4,923.3	-	0.0	20.0		-		-	0.0	Wet and floating humus
				72936	20.0	25.0	5.0	1.85				Slimey
				37	25.0	30.0	5.0	0.30	10.0	1.08		Slimey
				38	30.0	34.0	4.0	0.03				Clay, gravel at bottom
028-87	12,643	11,983	4,923.3	-	0.0	10.0		-		-	0.0	Water and humus
				72941	10.0	15.0	5.0	0.25	5.0	0.25		Probably mostly clay
029-87	12,609	11,888	4,925.1	72910	0.0	4.0	4.0	0.97				Sandy
				11	4.0	10.0	6.0	0.71			4.0	Slimey
				12	10.0	15.0	5.0	0.81				Sandy
				13	15.0	21.0	6.0	0.84				Sandy
				14	21.0	27.0	6.0	1.29				Sandy
				15	27.0	32.0	5.0	1.11				Sandy
				16	32.0	38.0	6.0	1.76				Sandy
				17	38.0	44.0	6.0	0.56	44.0	1.01		Slimey
030-87	12,702	11,854	4,924.1	-	0.0	17.0					0.0	Water, wet humus and light fines
				72907	17.0	20.0	3.0	1.69				Slimey
				908	20.0	28.0	8.0					Slimey
				909	28.0	33.0	5.0	0.20				Slimey
030A-87	12,702	11,854	4,924.1	-	0.0	20.0		-			0.0	Water, floating humus and slime
				72822	20.0	24.0	4.0	1.47				
				23	24.0	28.0	4.0	1.31				
				24	28.0	36.0	8.0	0.78				
									19.0	0.98		Average of combined 030 and 030A
031-87	12,796	11,820	4,923.1	-	0.0	19.0					0.0	Water and floating humus
				72906	19.0	24.0	5.0	1.12	5.0	1.12		Slimey
				-	24.0	25.0						Fine muck
032-87	12,985	11,753	4,923.1	-	0.0	14.0		-		-	0.0	Water 0.0 - 3.5, Floating slimes 3.5 - 14.0
					14.0	16.5	2.5	0.40	2.5	0.40		Mixed. To be redrilled 032A
032A-87	12,985	11,753	4,923.1	-	0.0	3.5		-			0.0	
				72820	3.5	14.0	10.5	1.19	10.5	1.19		Floating mud, poor recovery
				21	14.0	17.0	3.0	0.14				Sandy, gravel bottom
033-87	13,079	11,722	4,923.1	-	0.0	15.0		-			0.0	Water and floating humus
				72903	15.0	18.0	3.0	Tr	3.0	Tr		Slimey clay
034-87	13,173	11,686	4,923.4	72901	0.0	4.0	4.0	0.92			0.0	Slimey
				02	4.0	8.0	4.0	0.91	8.0	0.92		Slimey

TAILINGS SAMPLING TABULATION SHEET

(January - February, 1987)

Hole No.	Northing (ft)	Easting (ft)	Collar Elevation (ft)	Sampling: Sample No.	Sampling: From (ft)	Sampling: To (ft)	Sampling: Length (ft)	Assay Ag (oz/t)	Hole Average Depth (ft)	Hole Average Ag (oz/t)	Water Table Depth (ft)	Observations
036-87	13,388	11,396	4,929.7	72814	0.0	4.0	4.0	1.23				Sandy
				15	4.0	9.0	5.0	0.91				Sandy
				16	9.0	14.0	5.0	1.09				Sandy
				17	14.0	19.0	5.0	0.92				Sandy
				18	19.0	24.0	5.0	1.97				Sandy
				19	24.0	29.0	5.0	1.70	29.0	1.31		Sandy
037-87	13,576	11,327	4,932.0	72848	0.0	4.0	4.0	0.84				Sandy
				49	4.0	9.0	5.0	1.28	9.0	1.08		Sandy, rock bottom
038-87	13,876	11,447	4,941.3	87247	0.0	4.0	4.0	1.29				Sandy
				48	4.0	10.0	6.0	0.86			4.0	Sandy
				49	10.0	17.0	7.0	0.66	17.0	0.88		Fine, sandy
039-87	13,973	11,272	4,946.5	87254	0.0	2.0	2.0	1.08				Sandy
				55	2.0	4.0	2.0	1.29				Sandy
				56	4.0	12.5	8.5	1.17	12.5	1.17		Sandy, mud bottom
	14,347	11,862	4,966.1	56.40	4.0	12.5	8.5	1.17	12.5	1.17		Sandy, mud bottom
041-87	19,066	11,528	4,947.8	72718	0.0	2.5	2.5	1.05	2.5	1.05		Sandy, rock bottom
042-87	14,230	11,659	4,959.6		0.0	5.0	5.0	2.14				Sandy
					5.0	10.0	5.0	2.08				Sandy
					10.0	15.0	5.0	1.49				Sandy
					15.0	20.0	5.0	2.13				Sandy
					20.0	25.0	5.0	1.02				Sandy
					25.0	30.0	5.0	0.75				Sandy
					30.0	33.0	3.0	1.21	33.0	1.57		Sandy
043-87	14,347	11,862	4,966.1	87226	0.0	4.0	4.0	0.75				Sandy
				27	4.0	9.0	5.0	0.78				Sandy
				28	9.0	14.0	5.0	1.19				Sandy
				29	14.0	18.0	4.0	1.09	1.80	0.96		Sandy and fine sandy
045-87	14,582	11,311	4,952.1	72778	0.0	4.0	4.0	1.44			4.0	Sandy
				79	4.0	11.0	7.0	0.83	11.0	1.05		Slimey
046-87	14,842	11,203	4,939.1	72750	0.0	5.0	5.0	1.21			2.0	Sandy 0 - 2, Slimey 2 - 5
				51	5.0	11.0	6.0	0.71	11.0	0.94		Slimey
050-87	14,950	11,464	4,943.7	72725	0.0	4.0	4.0	1.22				Sandy
				26	4.0	9.0	5.0	1.17			4.0	Sandy
				27	9.0	14.0	5.0	0.69				Slimey
				28	14.0	19.0	5.0	2.21	19.0	1.33		Slimey
051-87	15,135	11,542	4,941.0	72722	0.0	4.0	4.0	1.32				Sandy
				23	4.0	6.5	2.5	0.83	6.5	1.13	4.0	Sandy
052-87	15,174	11,449	4,939.2	72746	0.0	3.5	3.5	1.29	3.5	1.29	0.0	Slimey
053-87	15,212	11,358	4,936.8	72770	0.0	4.0	4.0	1.14			0.0	Slimey
				71	4.0	10.0	6.0	0.88	10.0	0.98		Slimey
054-87	15,305	11,396	4,937.2	72772	0.0	4.0	4.0	1.28			0.0	Slimey
				73	4.0	6.5	2.5	1.37	6.5	1.31		Slimey

TAILINGS SAMPLING TABULATION SHEET

(January - February, 1987)

Hole No.	Northing (ft)	Easting (ft)	Collar Elevation (ft)	Sampling: Sample No.	Sampling: From (ft)	Sampling: To (ft)	Sampling: Length (ft)	Assay Ag (oz/t)	Hole Average Depth (ft)	Hole Average Ag (oz/t)	Water Table Depth (ft)	Observations
055-87	15,229	11,580	4,939.7	72724	0.0	5.0	5.0	1.47	5.0	1.47		Sandy
056-87	15,115	11,854	4,941.8	72686	0.0	4.0	4.0	1.04				Sandy
				87	4.0	9.0	5.0	0.67			4.0	Sandy
				88	9.0	15.0	6.0	0.65	15.0	0.76		Slimey
058-87	15,191	11,672	4,940.7	72690	0.0	4.0	4.0	1.48			0.0	Slimey
				72691	4.0	8.0	4.0	1.11				Slimey
				92	8.0	12.0	4.0	1.00	12.0	1.20		Slimey
059-87	15,097	11,633	4,943.0	72696	0.0	1.0	1.0	1.34	1.0	1.34	6.0	Slimey - Drilled to 11.0 four times, no sample
					1.0	11.0		-				No sample
063-87	14,930	11,777	4,949.5	72681	0.0	4.0	4.0	1.58				Sandy
				82	4.0	9.0	5.0	0.97				Sandy
				83	9.0	14.0	5.0	1.88			9.0	Sandy
				84	14.0	19.0	5.0	0.47				Slimey
				85	19.0	23.5	4.5	1.05	23.5	1.18		Slimey
065-87	15,005	11,595	4,945.2	72693	0.0	4.0	4.0	1.59				Sandy
				94	4.0	9.0	5.0	1.02			4.0	Mixed
				95	9.0	14.0	5.0	0.67	14.0	1.06		Slimey
066-87	15,043	11,504	4,942.3	72719	0.0	4.0	4.0	1.24				Sandy
				20	4.0	9.0	5.0	0.90			4.0	Mixed
				21	9.0	13.0	4.0	1.05	13.0	1.05		Mixed
067-87	15,082	11,411	4,938.8	72744	0.0	5.0	5.0	1.29			4.0	Sandy to 4.0
				45	5.0	10.0	5.0	0.95	10.0	1.12		Slimey
068-87	14,989	11,373	4,940.0	72743	0.0	5.0	5.0	1.34	5.0	1.34		Sandy
069-87	14,912	11,557	4,947.2	72697	0.0	4.0	4.0	1.43			0.0	Slimey - mixed
				98	4.0	10.0	6.0	1.21	10.0	1.30		Slimey - mixed
070-87	14,837	11,739	4.953.5	72677	0.0	4.0	4.0	1.48				Sandy
				78	4.0	9.0	5.0	0.97				Sandy
				79	9.0	14.0	5.0	0.84			9.0	Mixed
					14.0	20.0	-	-				No sample recovered
				22680	20.0	25.5	5.5	1.49	25.5	1.15		Slimey
072-87	14,745	11,701	4,959.9	72672	0.0	4.0	4.0	1.72				Sandy
				73	4.0	9.0	5.0	1.07				Sandy
				74	9.0	14.0	5.0	1.23				Sandy
				75	14.0	19.0	5.0	1.17			14.0	Slimey
				76	19.0	25.0	6.0	1.45	25.0	1.32		Sandy
074-87	14,820	11,518	4,951.7	72699	0.0	4.0	4.0	1.45				Sandy
				700	4.0	9.0	5.0	1.13				Sandy
				01	9.0	17.0	8.0	1.14	17.0	1.21	9.0	Slimey
075-87	14,858	11,426	4,946.8	72729	0.0	4.0	4.0	1.50				Sandy
				30	4.0	9.0	5.0	0.95			4.0	Mixed
				31	9.0	14.0	5.0	1.02				Mixed
				32	14.0	21.0	7.0	2.26	21.0	1.51		Mixed
076-87	14,805	11,295	4,944.1	72740	0.0	4.0	4.0	1.31				Sandy
					4.0	8.0	4.0	1.06	8.0	1.18	4.0	Sandy, rock bottom

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(January - February, 1987)

Hole No.	Northing (ft)	Easting (ft)	Collar Elevation (ft)	Sampling				Assay	Hole Average		Water Table Depth (ft)	Observations
				Sample No.	From (ft)	To (ft)	Length (ft)	Ag (oz/t)	Depth (ft)	Ag (oz/t)		
077-87	14,882	11,110	4,936.7	72747	0.0	4.0	4.0	2.29			0.0	Slimey
				48	4.0	10.0	6.0	1.22				Slimey
				49	10.0	15.0	5.0	0.64	15.0	1.31		Slimey
079-87	14,789	11,073	4,937.2	72733	0.0	4.0	4.0	1.40			0.0	Slimey
				34	4.0	9.0	5.0	1.08				Slimey
				35	9.0	16.5	7.5	0.51	16.5	0.90		Slimey
080-87	14,751	11,165	4,940.1	72736	0.0	4.0	4.0	1.35			0.0	Slimey
				37	4.0	11.5	7.5	0.63	11.5	0.88		Slimey
081-87	14,712	11,258	4,946.0	72738	0.0	4.0	4.0	1.30				Sandy
				39	4.0	9.0	5.0	0.85	9.0	1.32	4.0	Sandy, rock bottom
083-87	14,636	11,442	4,956.1	72705	0.0	4.0	4.0	1.01				Sandy
				06	4.0	9.0	5.0	1.16				Sandy, rock bottom
				07	8.0	13.5	4.5	1.20	13.5	1.13		Sandy, rock bottom
084-87	14,728	11,481	4,954.4	72702	0.0	4.0	4.0	1.28				Sandy
				03	4.0	9.0	5.0	1.57				2' missing
				04	9.0	15.0	6.0	1.26	15.0	1.37	9.0	Mixed
				-	15.0	20.0	-					Rock bottom, lost 5'
086-87	14,561	11,625	4,970.3	72651	0.0	5.0	5.0	1.73				Sandy
				52	5.0	10.0	5.0	1.42				Sandy
				53	10.0	15.0	5.0	2.23				Sandy
				54	15.0	24.0	9.0	1.54			20.0	Mixed, possibly lost 21 - 24
				55	24.0	29.0	5.0	1.37				Mixed
				56	29.0	33.0	4.0	0.85				Mixed
				57	33.0	32.5	4.5	0.26	37.5	1.29		Slimey, 4" clay at bottom
087-87	14,653	11,663	4,963.7	72667	0.0	4.0	4.0	1.50				Sand
				68	4.0	9.0	5.0	0.94				Sand, slimes 4 - 5
				69	9.0	14.0	5.0	1.29				Sand
				70	14.0	19.0	5.0	1.01			14.0	Slimey
				71	19.0	22.0	3.0	1.48	22.0	1.21		Slimey, humus and clay bottom
088-87	14,572	11,849	4,968.5	87218	0.0	4.0	4.0	1.36				Sandy
				19	4.0	9.0	5.0	0.98				Sandy
				20	9.0	14.0	5.0	1.05				Sandy
				21	14.0	19.0	5.0	0.94				Sandy
				22	19.0	21.0	2.0	1.07	21.0	1.07		Sandy
089-87	14,520	11,718	4,973.1	87230	0.0	4.0	4.0	2.25				Sandy
				31	4.0	9.0	5.0	1.97				Sandy
				32	9.0	14.0	5.0	1.52				Sandy
				33	14.0	19.0	5.0	1.27				Sandy
				34	19.0	24.0	5.0	2.43				Sandy
				35	24.0	29.0	5.0	1.59				Sandy
				36	29.0	34.0	5.0	0.94			29.0	Sandy fine
				37	34.0	39.0	5.0	0.66				Mixed
				38	39.0	46.5	7.5	0.81	46.5	1.44		Mixed
090-87	14,481	11,810	4,972.3	87202	0.0	4.0	4.0	1.07				Sandy
				03	4.0	8.0	4.0	1.21				Sandy
				04	8.0	11.0	3.0	1.39				Sandy

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Hole No.	Northing (ft)	Easting (ft)	Collar Elevation (ft)	Sampling: Sample No.	Sampling: From (ft)	Sampling: To (ft)	Sampling: Length (ft)	Assay Ag (oz/t)	Hole Average Depth (ft)	Hole Average Ag (oz/t)	Water Table Depth (ft)	Observations
				05	11.0	14.0	3.0	2.07				Sandy
				06	14.0	12.0	3.0	1.90				Sandy
				07	17.0	22.0	5.0	2.21				Sandy
				08	22.0	29.0	7.0	2.46				Sandy
				09	29.0	34.0	5.0	1.21			29.0	Sandy
				10	34.0	39.0	5.0	0.81				Sandy
				11	39.0	43.0	4.0	0.69				Sandy
				12	43.0	47.0	4.0	0.80				Mixed
				13	47.0	51.5	4.5	1.73	51.5	1.50		Mixed
091-87	14,387	11,772	4,972.2	87192	0.0	4.0	4.0	1.76				Sandy
				93	4.0	9.0	5.0	2.23				Sandy
				94	9.0	14.0	5.0	3.17				Sandy
				95	14.0	19.0	5.0	1.77				Sandy
				96	19.0	24.0	5.0	1.80				Sandy
				97	24.0	29.0	5.0	1.54				Sandy
				98	29.0	34.0	5.0	1.24			29.0	Sandy
				99	34.0	39.0	5.0	0.75				Sandy
				87200	39.0	44.0	5.0	0.65				Mixed
				01	44.0	48.5	4.5	2.28	48.5	1.71		Mixed
092-87	14,468	11,586	4,968.8	72658	0.0	5.0	5.0	2.93				Sandy
				59	5.0	10.0	5.0	1.88				Sandy
				60	10.0	15.0	5.0	2.50				Sandy
				61	15.0	20.0	5.0	3.47				Sandy
				62	20.0	25.0	5.0	1.36			22.0	Sandy to 22, slimey 22-45
				63	25.0	30.0	5.0	1.59				Slimey
				64	30.0	35.0	5.0	0.97				Slimey
				65	35.0	40.0	5.0	1.12				Slimey
				66	40.0	45.0	5.0	1.10	45.0	1.88		Slimey, 4" clay at bottom
094-87	14,620	11,218	4,946.2	72759	0.0	4.0	4.0	1.67	4.0	1.67		Sandy
095-87	14,490	11,272	4,950.1	72780	0.0	4.0	4.0	1.04				Sandy
				81	4.0	10.0	6.0	0.86	10.0	0.93	4.0	Mixed
096-87	14,451	11,365	4,953.6	72708	0.0	4.0	4.0	0.90				Sandy
				09	4.0	9.0	5.0	1.15				Sandy
				10	9.0	14.0	5.0	1.16				Sandy
				11	14.0	18.0	4.0	1.00	18.0	1.06	14.0	Mixed
098-87	14,356	11,598.5	4,966.8		0.0	5.0	5.0	1.63				Sandy
					5.0	10.0	5.0	3.21				Sandy
					10.0	15.0	5.0	2.77				Sandy
					15.0	20.0	5.0	1.93				Sandy
					20.0	25.0	5.0	1.09			20.0	Mixed
					25.0	30.0	5.0	1.28				Mixed
					30.0	35.0	5.0	1.02				Mixed
					35.0	40.0	5.0	0.92				Mixed
					40.0	45.0	5.0	1.44	45.0	1.70		Humus in bottom

TAILINGS SAMPLING TABULATION SHEET

(January - February, 1987)

Hole No.	Northing (ft)	Easting (ft)	Collar Elevation (ft)	Sampling Sample No.	Sampling From (ft)	Sampling To (ft)	Sampling Length (ft)	Assay Ag (oz/t)	Hole Average Depth (ft)	Hole Average Ag (oz/t)	Water Table Depth (ft)	Observations
099-87	14,318	11,689	4,967.7		0.0	5.0	5.0	1.22				Sandy
					5.0	10.0	5.0	2.59				Sandy
					10.0	15.0	5.0	2.25				Sandy
					15.0	20.0	5.0	2.64			15.0	Mixed
					20.0	25.0	5.0	1.32				Mixed
					25.0	30.0	5.0	1.53				Mixed
					30.0	35.0	5.0	0.75				Mixed
					35.0	40.0	5.0	1.10	40.0	1.68		Mixed, bottom gravelly till
100-87A	14,286	11,779	4,962.6		0.0	5.0	5.0	2.06				Sandy
					5.0	10.0	5.0	1.20				Sandy
					10.0	15.0	5.0	1.14				Sandy
					15.0	20.0	5.0	0.70				Mixed
					20.0	25.0	5.0	1.01			18.0	Mixed
					25.0	30.0	5.0	1.45				Mixed
					30.0	32.0	2.0	3.24	32.0	1.38		Mixed
100-87	14,297	11,733	4,963.1	72752	0.0	5.0	5.0	1.34				Sandy
				53	5.0	10.0	5.0	1.75				Sandy
				54	10.0	15.0	5.0	1.76				Sandy
				55	15.0	20.0	5.0	1.38			15.0	Mixed
				56	20.0	25.0	5.0	0.67				Mixed
				57	25.0	30.0	5.0	0.57				Mixed
				58	30.0	36.5	6.5	3.60	36.5	1.66		Mixed
101-87	14,206	11,695	4,957.5	87186	0.0	4.0	4.0	1.01				Sandy
				87	4.0	9.0	5.0	0.90				Sandy
				88	9.0	14.0	5.0	0.97				Sandy
				89	14.0	19.0	5.0	1.60			14.0	Sandy
				90	19.0	24.0	5.0	1.47				Sandy
				91	24.0	30.0	6.0	0.99	30.0	1.16		Sandy
102-87	14,281	11,565	4,962.5		0.0	5.0	5.0	2.31				Sandy
					5.0	10.0	5.0	1.97				Sandy
					10.0	15.0	5.0	1.37				Sandy
					15.0	20.0	5.0	1.62			15.0	Sandy
					20.0	25.0	5.0	1.47				Mixed
					25.0	30.0	5.0	0.55				Mixed
					30.0	35.0	5.0	0.90				Mixed
					35.0	40.0	5.0	1.88	40.0	1.51		Mixed, hard packed and coarse at 39.0
103-87	14,358	11,327	4,952.1	72712	0.0	4.0	4.0	0.91				Sandy
				13	4.0	9.0	5.0	1.39				Sandy
				14	9.0	12.0	3.0	1.29	12.0	1.20	9.0	Mixed
104-87	14,266	11,287	4,949.0	72715	0.0	4.0	4.0	0.88				Sandy
				16	4.0	9.0	5.0	1.45				Sandy
				17	9.0	13.0	4.0	1.51	13.0	1.29	9.0	Mixed, rock bottom
106-87	14,177	11,519	4,954.8		0.0	5.0	5.0	1.32				Sandy
					5.0	10.0	5.0	1.26				Sandy

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					10.0	15.0	5.0	1.21			10.0	Mixed
					15.0	20.0	5.0	0.55				Mixed
					20.0	25.0	5.0	0.57				Mixed
					25.0	30.0	5.0	0.46				Mixed
					30.0	35.0	5.0	2.18	35.0	1.07		Sandy, coarse, hard packed
108-87	14,122	11,659	4,951.6	87183	0.0	4.0	4.0	0.50				Sandy
				84	4.0	9.0	5.0	1.65				Sandy
				85	9.0	13.0	4.0	1.47	13.0	1.24	9.0	Sandy, gravel bottom
109-87	14,026	11,620	4,947.2	87178	0.0	4.0	4.0	1.10				Sandy
				79	4.0	9.0	5.0	1.34				Sandy
				80	9.0	14.0	5.0	1.01				Sandy
				81	14.0	19.0	5.0	0.89			14.0	Fine sand
				82	19.0	26.5	7.5	0.76	26.5	0.99		Fine sand
110-87	14,078	11,482	4,947.8	No samples				20.0				Rock bottom
112-87	14,125	11,119	4,944.8	87262	0.0	4.0	4.0	0.91				Sandy
				63	4.0	9.0	5.0	1.18				Sandy
				64	9.0	14.0	5.0	0.76				Fine sand
115-87	13,927	11,579	4,944.3	87175	0.0	5.0	5.0	1.15				Sandy
				76	5.0	11.0	6.0	1.00				Sandy
				77	11.0	17.0	6.0	1.03	17.0	1.05		Fine sand
116-87	13,839	11,541	4,939.6	87242	0.0	4.0	4.0	0.78				Sand
				43	4.0	12.0	8.0	0.73			4.0	Mixed
				44	12.0	20.0	8.0	0.68				Mixed
				45	20.0	25.0	5.0	1.87				Mixed
				46	25.0	30.0	5.0	4.10	30.0	1.48		Mixed, coarse sand bottom
117-87	13,906	11,407	4,941.8	No Sample								
118-87	13,994	11,174	4,946.5	87257	0.0	4.0	4.0	2.21				Sandy
				58	4.0	9.0	5.0	1.01				Sandy
				59	9.0	14.0	5.0	1.29				Sandy
				60	14.0	19.0	5.0	1.81				Sandy
				61	19.0	25.0	6.0	1.19	25.0	1.46		Sandy
119-87	13,767	11,567	4,939.5	87239	0.0	4.0	4.0	1.18				Sandy
				40	4.0	10.0	6.0	0.98				Sandy
				41	10.0	18.0	8.0	0.97	18.0	1.02	10.0	Fine sand, humus bottom
120-87	13,746	11,481	4,937.2	No sample						0.5		Rock
121-87	13,671	11,371	4,934.4	87154	0.0	4.0	4.0	0.73				Sandy
				55	4.0	9.0	5.0	4.45				Sandy
				56	9.0	16.5	7.5	2.26	16.5	2.55	10.0	Sandy, mud bottom
122-87	13,621	11,308	4,933.6	72839	0.0	4.0	4.0	1.78				Sandy
				40	4.0	6.5	2.5	2.93	6.5	2.22		Sandy, mud bottom
123-87	13,610	11,420	4,936.7	72841	0.0	4.0	4.0	1.10				Sandy, dry
				42	4.0	9.0	5.0	0.92			4.0	Sandy, wet
				43	9.0	14.0	5.0	1.63				Sandy, wet
				44		14.0	19.0	5.0	1.88			Sandy, wet

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				65	14.0	20.0	6.0	1.79	20.0	1.20		Fine sand, rock bottom
				45	19.0	24.0	5.0	1.87				Sandy, dry
				46	24.0	29.0	5.0	1.58				Sandy, dry
				47	29.0	34.0	5.0	1.26	34.0	1.47		Sandy, fine gravel bottom
124-87	13,678	11,610	4,938.5	87157	0.0	4.0	4.0	0.94				Sandy
				58	4.0	9.0	5.0	0.77				Sandy
				59	9.0	14.0	5.0	0.99				Sandy
				60	14.0	17.0	3.0	1.10	17.0	0.93		Mixed, mud bottom
125-87	13,546	11,550	4,935.2	72786	0.0	4.0	4.0	0.79				Sandy
				87	4.0	9.0	5.0	0.78			4.0	Sandy
				88	9.0	14.0	5.0	1.21				Mixed
				89	14.0	20.0	6.0	1.26	20.0	1.03		Mixed
126-87	13,503	11,420	4,933.6	87161	0.0	4.0	4.0	1.94				Sandy
				62	4.0	9.0	5.0	0.88				Sandy
				63	9.0	14.0	5.0	1.34			9.0	Sandy
				64	14.0	19.0	5.0	1.28				Sandy
				65	19.0	25.0	6.0	1.29				Sandy
				66	25.0	31.0	6.0	1.43	31.0	1.34		Sandy, some mud & gravel bottom
127-87	13,481	11,359	4,929.9	87151	0.0	4.0	4.0	1.59				Sandy
				52	4.0	9.5	5.5	1.38	9.5	1.47	5.0	Sandy
128-87	13,352	11,300	4,924.2	72807	0.0	4.0	4.0	0.90				Sandy
				08	4.0	9.0	5.0	0.80				Sandy
				09	9.0	14.0	5.0	0.92			9.0	Sandy
				10	14.0	19.0	5.0	0.54				Sandy, fine
				11	19.0	24.0	5.0	0.49				Sandy, 2" mud
				12	24.0	27.5	3.5	1.30	27.5	0.80		Gravel bottom
129-87	13,421	11,488	4,931.5	72832	0.0	4.0	4.0	1.78				Sandy
				33	4.0	9.0	5.0	3.15				Sandy
				34	9.0	14.0	5.0	2.79				Sandy
				35	14.0	19.0	5.0	2.60				Sandy
				36	19.0	24.0	5.0	1.73				Sandy
				37	24.0	29.0	5.0	1.50				Sandy
				38	29.0	34.0	5.0	2.61	34.0	2.32		Slimey, some mud bottom
130-87	13,293	11,428	4,928.1	72873	0.0	4.0	4.0	1.19				Sandy
				74	4.0	9.0	5.0	1.01				Sandy
				75	9.0	14.0	5.0	0.99				Sandy
				76	14.0	19.0	5.0	0.98				Sandy
				77	19.0	24.0	5.0	1.29				Sandy
				78	24.0	29.0	5.0	1.57				Sandy
				79	29.0	34.0	5.0	1.69				Sandy
				80	34.0	39.0	5.0	1.26				Sandy
				81	39.0	43.0	4.0	1.47	43.0	1.27		Sandy
131-87	13,327	11,523	4,928.2	72825	0.0	4.0	4.0	1.80				Sandy
				26	4.0	9.0	5.0	1.83				Sandy

TAILINGS SAMPLING TABULATION SHEET

(January - February, 1987)

Hole No.	Northing (ft)	Easting (ft)	Collar Elevation (ft)	Sampling				Assay	Hole Average		Water Table Depth (ft)	Observations
				Sample No.	From (ft)	To (ft)	Length (ft)	Ag (oz/t)	Depth (ft)	Ag (oz/t)		
				27	9.0	14.0	5.0	1.80				Sandy
				28	14.0	19.0	5.0	0.45				Sandy
				29	19.0	23.0	4.0	0.25				Sandy, some mud
				30	23.0	27.0	4.0	1.14	27.0	1.23		Slimey, some mud
				31	27.0	31.0	4.0	0.45	31.0	1.13		Slimey, some mud
132-87	13,358	11,619	4,928.8	72790	0.0	4.0	4.0	3.50				Sandy
				91	4.0	9.0	5.0	2.13			4.0	Mixed
				92	9.0	14.0	5.0	1.88				Mixed
				93	14.0	19.0	5.0	1.41				Mixed
				94	19.0	23.0	4.0	1.02	23.0	2.43		Mixed
133-87	13,289	11,740	4,924.3	72865	0.0	6.5	6.5	1.47	6.0	1.47	0.0	Sandy, some slime
134-87	13,232	11,557	4,925.0	72856	0.0	4.0	4.0	2.26			1.0	Sandy
				57	4.0	10.0	6.0	1.93				Sandy, some mud
				58	10.0	15.0	5.0	2.09				Sandy
				59	15.0	20.0	5.0	1.82				Mixed
				60	20.0	25.0	5.0	1.85				Mixed
				61	25.0	30.0	5.0	1.67				Mixed
				62	30.0	35.0	5.0	2.10	35.0	1.95		Mixed
				-	35.0	42.0		-				No sample in three tries
135-87	13,192	11.441	4,925.1	87167	0.0	4.0	4.0	0.69				Mud (humus and tails)
				68	4.0	9.0	5.0	1.91			4.0	Mud
				69	9.0	14.0	5.0	1.61				Slimey mud
				70	14.0	19.0	5.0	1.43				Fine sand, some mud
				71	19.0	24.0	5.0	1.94				Mixed
				72	24.0	29.0	5.0	1.55				Mixed
				73	29.0	34.0	5.0	2.36				Mixed
				74	34.0	40.0	6.0	1.53	40.0	1.65		Mixed, humus bottom
136-87	15,342	11,303	4,936.6	72774	0.0	4.0	4.0	1.56			0.0	Mixed
				75	4.0	7.0	3.0	1.24	7.0	1.42		Mixed
				-	7.0	8.0						Clay and gravel
137-87	15,366	11,243	4,936.0	72776	0.0	5.0	5.0	1.71	5.0	1.71	0.0	Mixed
139-87	15,120	11,317	4,937.1	72767	0.0	4.0	4.0	1.20			0.0	Mixed
				68	4.0	9.0	5.0	0.93				Mixed, slimier
				69	9.0	16.0	7.0	0.71	16.0	0.90		Mixed, slimier, fibrous humus bottom
140-87	15,027	11,281	4,936.2	72760	0.0	4.0	4.0	1.50				Sandy
				61	4.0	9.0	5.0	1.13			4.0	Mixed
				62	9.0	14.0	5.0	0.74				Mixed
141-87	15,065	11,187	4,934.4	72766	0.0	2.0	2.0	1.74	2.0	1.74	0.0	Mixed, wet humus bottom
142-87	14,973	11,150	4,934.6	-	0.0	10.0		-		-	0.0	Wet humus, no sample
143-87	14,935	11,241	4,937.6	72764	0.0	4.0	4.0	1.44			0.0	Slimey
				65	4.0	9.0	5.0	1.05	9.0	1.22		Slimey fibrous humus bottom
144-87	14,896	11,334	4,942.1	72742	0.0	3.0	3.0	1.46	3.0	1.46	0.0	Mixed
				63	14.0	17.0	3.0	0.75	17.0	1.04		Mixed
200-87	14,301	11,492	4,959.7		0.0	5.0	5.0	1.02				Sandy, hard packed

TAILINGS SAMPLING TABULATION SHEET
(January - February, 1987)

Hole No.	Northing (ft)	Easting (ft)	Collar Elevation (ft)	Sampling				Assay	Hole Average		Water Table Depth (ft)	Observations
				Sample No.	From (ft)	To (ft)	Length (ft)	Ag (oz/t)	Depth (ft)	Ag (oz/t)		
					5.0	10.0	5.0	1.00				Sandy, hard packed
					10.0	12.0	2.0	1.63	12.0	1.11		Sandy, hard packed
												Hole stopped at 12 in 3 attempts
201-87	14,192	11,449	4,954.5		0.0	5.0	5.0	1.74				Sandy
					5.0	10.0	5.0	1.36				Sandy
					10.0	15.0	5.0	1.78				Sandy
					15.0	20.0	5.0	0.70			15.0	Mixed
					20.0	25.0	5.0	0.58				Mixed
					25.0	30.0						Missing?
					30.0	33.0	3.0	0.93	33.0	1.19		Mixed
202-87	14,103	11,414	4,952.6		0.0	5.0	5.0	4.45				Sandy, hard packed
					5.0	10.0	5.0	2.10				Sandy, hard packed
					10.0	15.0	5.0	1.17				Sandy, hard packed
					15.0	20.0	5.0	0.71			15.0	Mixed
					20.0	25.0	5.0	0.76				Mixed
					25.0	30.0	5.0	1.09	30.0	1.71		Mixed, muskeg bottom
203-87	14,167	11,244	4,948.2		0.0	5.0	5.0	1.18				Sandy, hard packed
					5.0	10.0	5.0	0.95				Sandy, hard packed
					10.0	15.0	5.0	1.83			10.0	Mixed
					15.0	20.0	5.0	0.67				Mixed
					20.0	25.0	5.0	0.74	25.0	1.07		Mixed, clay bottom
204-87	14,396	11,533	4,964.9		0.0	5.0	5.0	1.54				Sandy, hard packed
					5.0	10.0	5.0	2.79				Sandy, hard packed
					10.0	15.0	5.0	2.21				Sandy, hard packed
					15.0	20.0	5.0	2.11				Sandy, hard packed
					20.0	25.0	5.0	1.15			20.0	Mixed
					25.0	30.0	5.0	1.13				Mixed
					30.0	35.0	5.0	0.73	35.0	1.67		Mixed, muskeg bottom
205-87	15,207	11,891	4,940.8	72689	0.0	1.0	1.0	1.30	1.0	1.30		Sandy, dry, rock bottom
206-87	15,355	11,416	4,937.2	72777	0.0	4.5	4.5	1.30	4.5	1.30	0.0	Mixed, bottom clay, some humus
547-87	13,320	11,206	4,922.3	72801	0.0	4.0	4.0	0.41			0.0	Mud (humus and slime)
				02	4.0	9.0	5.0	0.42				Mud
				03	9.0	14.0	5.0	0.42	14.0	0.42		Mud
				04	14.0	19.0	5.0	1.18				Slimey
				05	19.0	23.0	4.0	1.31				Slimey
				06	23.0	26.0	3.0	1.09	26.0	0.78		Slimey, gravel bottom
548-87	13,259	11,332	4,924.0	72892	0.0	5.0	5.0	2.08			0.0	Mud
				93	5.0	12.0	7.0	1.92				Mud
				94	12.0	20.0	8.0	1.32				Mud
				95	20.0	30.0	10.0	1.42				Slimey
				96	30.0	40.0	10.0	1.62	40.0	16.2		Slimey
				97	40.0	48.0	8.0	0.26				Slimey
549-87	13,225	11,237	4,922.3	72888	0.0	4.0	4.0	3.26			0.0	Mud
				89	4.0	9.0	5.0	2.71				Mud

TAILINGS SAMPLING TABULATION SHEET

(January - February, 1987)

Hole No.	Northing (ft)	Easting (ft)	Collar Elevation (ft)	Sampling Sample No.	From (ft)	To (ft)	Length (ft)	Assay Ag (oz/t)	Hole Average Depth (ft)	Ag (oz/t)	Water Table Depth (ft)	Observations
				90	9.0	14.0	5.0	3.05				Mud
				91	14.0	18.5	4.5	1.67	18.5	2.67		Slimey, gravel bottom
550-87	13,229	11,130	4,922.8	72813	0.0	6.0	6.0	0.74	6.0	0.74	0.0	Mud
551-87	13,166	11,368	4,923.6	72882	0.0	4.0	4.0	0.44			0.0	Mud
				83	4.0	9.0	5.0	1.60				Slimey
				84	9.0	15.0	6.0	1.91				Slimey
				85	15.0	20.0	5.0	1.13				Slimey
				86	20.0	25.0	5.0	0.85	25.0	1.24		Slimey
552-87	13,138	11,292	4,922.8	72887	0.0	6.0	6.0	0.80	6.0	0.80	0.0	Mud, rocks at bottom
553-87	13,138	11,590	4,923.5	72854	0.0	4.0	4.0	0.85			0.0	Wet slimey
554-87	13,198	11,758	4,923.4	72866	0.0	5.0	5.0	1.72	5.0	1.72	0.0	Sandy, fine
555-87	13,105	11,495	4,923.6	72872	0.0	9.0	9.0	0.08	9.0	0.08	0.0	0-2 Mud, slimey, clay
556-87	12,898	11,686	4,923.1		0.0	6.0		-			0.0	Water and humus
				72863	6.0	12.0	6.0	0.13	6.0	0.13		Mostly Mud, 1.0' slimes at bottom
557-87	12,736	11,740	4,923.3	72853	0.0	5.0	5.0	1.78	5.0	1.78	0.0	Sandy
558-87	12,668	11,758	4,923.3		0.0	5.0		-			0.0	Water and floating humus
				72949	5.0	10.0	5.0	0.32				Mud
				72950	10.0	15.0	5.0	0.53				Slimes
				72850	15.0	20.0	5.0	0.84				Slimes
				51	20.0	25.0	5.0	0.26	20.0	0.49		Slimes
559-87	12,599	11,784	4,924.1	72945	0.0	4.0	4.0	2.16			2.0	Sandy
				46	4.0	10.0	6.0	1.46				Sandy
				47	10.0	15.0	5.0	0.99				Sandy
				48	15.0	20.0	20.0	1.01	20.0	1.37		Sandy
560-87	12,480	11,827	4,923.6	72944	0.0	3.0	3.0	0.93	3.0	0.93	3.0	Sandy, rock bottom
561-87	12,387	11,859	4,924.4	72942	0.0	4.0	4.0	0.77			4.0	Sandy
				43	4.0	9.0	5.0	1.01	9.0	0.90		Sand and slime, rock bottom
562-87	12,823	11,895	4,923.1	-	0.0	3.0		-			0.0	Water
				72864	3.0	7.0	4.0	Tr	4.0	Tr		Mud and 6" tailings at bottom
563-87	12,915	11,858	4,923.1						12.0		0.0	Rock bottom, no sample
564-87	13,009	11,825	4,923.1	-	0.0	6.0		-			0.0	Water and humus
				72870	6.0	10.0	4.0	1.39	4.0	1.39		Sandy
				71	10.0	18.0	8.0	0.02				Sandy, gravel bottom
				-	18.0	22.0		-				Soil sample, possible clay below gravel
565-87	13,103	11,789	4,923.1	-	0.0	7.0		-				Water and humus
				72867	7.0	11.0	4.0	1.37	4.0	1.37		Fine, sandy
				68	11.0	16.0	5.0	0.02				Clay
				69	16.0	23.0	7.0	0.02				Course, sand
566-87	13,542	11,232	4,928.0	87153	0.0	3.0	3.0	4.43	3.0	4.43	0.0	Sandy, some Mud at bottom
567-87	14,678	11,887	4,961.2	87214	0.0	4.0	4.0	1.33				Sandy
				15	4.0	9.0	5.0	1.07				Sandy
				16	9.0	14.0	5.0	1.06				Sandy
				17	14.0	18.5	4.5	1.24	18.5	1.16		Sandy
568-87	14,442	11,901	4,970.6	87223	0.0	4.0	4.0	0.66				Sandy

TAILINGS SAMPLING TABULATION SHEET
(January - February, 1987)

Hole No.	Northing (ft)	Easting (ft)	Collar Elevation (ft)	Sampling: Sample No.	Sampling: From (ft)	Sampling: To (ft)	Length (ft)	Assay Ag (oz/t)	Hole Average Depth (ft)	Hole Average Ag (oz/t)	Water Table Depth (ft)	Observations
				90	9.0	14.0	5.0	3.05				Mud
				91	14.0	18.5	4.5	1.67	18.5	2.67		Slimey, gravel bottom
550-87	13,229	11,130	4,922.8	72813	0.0	6.0	6.0	0.74	6.0	0.74	0.0	Mud
551-87	13,166	11,368	4,923.6	72882	0.0	4.0	4.0	0.44			0.0	Mud
				83	4.0	9.0	5.0	1.60				Slimey
				84	9.0	15.0	6.0	1.91				Slimey
				85	15.0	20.0	5.0	1.13				Slimey
				86	20.0	25.0	5.0	0.85	25.0	1.24		Slimey
552-87	13,138	11,292	4,922.8	72887	0.0	6.0	6.0	0.80	6.0	0.80	0.0	Mud, rocks at bottom
553-87	13,138	11,590	4,923.5	72854	0.0	4.0	4.0	0.85			0.0	Wet slimey
554-87	13,198	11,758	4,923.4	72866	0.0	5.0	5.0	1.72	5.0	1.72	0.0	Sandy, fine
555-87	13,105	11,495	4,923.6	72872	0.0	9.0	9.0	0.08	9.0	0.08	0.0	0-2 Mud, slimey, clay
556-87	12,898	11,686	4,923.1		0.0	6.0		-			0.0	Water and humus
				72863	6.0	12.0	6.0	0.13	6.0	0.13		Mostly Mud, 1.0' slimes at bottom
557-87	12,736	11,740	4,923.3	72853	0.0	5.0	5.0	1.78	5.0	1.78	0.0	Sandy
558-87	12,668	11,758	4,923.3		0.0	5.0		-			0.0	Water and floating humus
				72949	5.0	10.0	5.0	0.32				Mud
				72950	10.0	15.0	5.0	0.53				Slimes
				72850	15.0	20.0	5.0	0.84				Slimes
				51	20.0	25.0	5.0	0.26	20.0	0.49		Slimes
559-87	12,599	11,784	4,924.1	72945	0.0	4.0	4.0	2.16			2.0	Sandy
				46	4.0	10.0	6.0	1.46				Sandy
				47	10.0	15.0	5.0	0.99				Sandy
				48	15.0	20.0	20.0	1.01	20.0	1.37		Sandy
560-87	12,480	11,827	4,923.6	72944	0.0	3.0	3.0	0.93	3.0	0.93	3.0	Sandy, rock bottom
561-87	12,387	11,859	4,924.4	72942	0.0	4.0	4.0	0.77			4.0	Sandy
				43	4.0	9.0	5.0	1.01	9.0	0.90		Sand and slime, rock bottom
562-87	12,823	11,895	4,923.1	-	0.0	3.0		-			0.0	Water
				72864	3.0	7.0	4.0	Tr	4.0	Tr		Mud and 6" tailings at bottom
563-87	12,915	11,858	4,923.1						12.0		0.0	Rock bottom, no sample
564-87	13,009	11,825	4,923.1	-	0.0	6.0		-			0.0	Water and humus
				72870	6.0	10.0	4.0	1.39	4.0	1.39		Sandy
				71	10.0	18.0	8.0	0.02				Sandy, gravel bottom
				-	18.0	22.0		-				No sample, possible clay below gravel
565-87	13,103	11,789	4,923.1	-	0.0	7.0		-				Water and humus
				72867	7.0	11.0	4.0	1.37	4.0	1.37		Fine, sandy
				68	11.0	16.0	5.0	0.02				Clay
				69	16.0	23.0	7.0	0.02				Course, sand
566-87	13,542	11,232	4,928.0	87153	0.0	3.0	3.0	4.43	3.0	4.43	0.0	Sandy, some Mud at bottom
567-87	14,678	11,887	4,961.2	87214	0.0	4.0	4.0	1.33				Sandy
				15	4.0	9.0	5.0	1.07				Sandy
				16	9.0	14.0	5.0	1.06				Sandy
				17	14.0	18.5	4.5	1.24	18.5	1.16		Sandy
568-87	14,442	11,901	4,970.6	87223	0.0	4.0	4.0	0.66				Sandy

TAILINGS SAMPLING TABULATION SHEET
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Hole No.	Northing (ft)	Easting (ft)	Collar Elevation (ft)	Sampling: Sample No.	Sampling: From (ft)	Sampling: To (ft)	Sampling: Length (ft)	Assay Ag (oz/t)	Hole Average Depth (ft)	Hole Average Ag (oz/t)	Water Table Depth (ft)	Observations
				24	4.0	9.0	5.0	0.78				Sandy
				25	9.0	13.5	4.5	0.93	13.5	0.79		Sandy
642-87	13,287	11,110	4,922.3	72898	0.0	5.0	5.0	1.20			0.0	Slimey, some mud
				99	5.0	10.0	5.0	1.02	10.0	1.11		Slimey
				900	10.0	16.0	6.0	0.06				Slimey, some Mud, clay
SDK-S1	14,540	11,727	4,972.7		0.0	5.0	5.0	2.08				Sandy
					5.0	10.0	5.0	1.79				Sandy
					10.0	15.0	5.0	1.99				Sandy
					15.0	20.0	5.0	1.46				Sandy
					20.0	25.0	5.0	1.65				Sandy
					25.0	30.0	5.0	0.92			25.0	Sandy
					30.0	35.0	5.0	0.86				Mixed
					35.0	40.0	5.0	3.30				Mixed
					40.0	45.0	5.0	3.91				Mixed
					45.0	47.0	2.0	3.75	47.0	2.08		Mixed
					47.0	51.0	4.0	-				Grey gravel
SDK-S2	14,438	11,853	4,971.5		0.0	5.0	5.0	1.44				Sandy
					5.0	10.0	5.0	1.91				Sandy
					10.0	15.0	5.0	1.74				Sandy
					15.0	20.0	5.0	1.27				Sandy
					20.0	25.0	5.0	1.45			20.0	Sandy
					25.0	30.0	5.0	1.00				Sandy
					30.0	35.0	5.0	0.57	35.0	1.34		Mixed
					35.0	36.0	1.0	-				Humus
SDK-S3	14,185	11,592	4,953.4		0.0	5.0	5.0	2.28				Sandy
					5.0	10.0	5.0	2.03				Sandy
					10.0	15.0	5.0	1.27				Sandy
					15.0	20.0	5.0	0.69				Sandy
					20.0	25.0	5.0	0.84				Sandy
					25.0	26.0	1.0	0.50	26.0	1.39		Sandy
					26.0	27.0	1.0	-				Coarse brown sand
SDK-S4	13,560	11,497	4,935.2		0.0	5.0	5.0	0.96				Sandy
					5.0	10.0	5.0	1.41			5.0	Sandy
					10.0	15.0	5.0	1.40				Sandy
					15.0	20.0	5.0	1.54				Mixed, minor humus layer
					20.0	25.0	5.0	1.35				Mixed
					25.0	30.0	5.0	1.42				Mixed

TAILINGS SAMPLING TABULATION SHEET
(January - February, 1987)

Hole No.	Northing (ft)	Easting (ft)	Collar Elevation (ft)	Sampling: Sample No.	Sampling: From (ft)	Sampling: To (ft)	Sampling: Length (ft)	Assay Ag (oz/t)	Hole Average Depth (ft)	Hole Average Ag (oz/t)	Water Table Depth (ft)	Observations
					30.0	35.0	5.0	1.42				Mixed
					35.0	40.0	5.0	1.34				Mixed
					40.0	42.0	2.0	1.08	42.0	1.34		Mixed
					42.0	43.0	1.0					Clayey gravel
SDK-S5	13,392	11,576	4,931.0		0.0	5.0	5.0	4.90				Sandy
					5.0	10.0	5.0	1.19				Sandy
					10.0	15.0	5.0	1.99				Sandy
					15.0	20.0	5.0	1.27				Mixed, minor organics
					20.0	25.0	5.0	1.61				Mixed
					25.0	30.0	5.0	1.87				Mixed
					30.0	35.0	5.0	2.23				Mixed
					35.0	36.0	1.0	0.49	36.0	2.11		Mixed
					36.0	38.0	2.0	-				Clayey ground
SDK-S6	12,820	11,796	4,923.1		0.0	15.0					0.0	Water, floating humus and silt
	(Not surveyed)				15.0	20.0	5.0	0.24	0.24	5.0		Slimey
					20.0	24.0	4.0	0.02				Clay
SDK-S7	12,987	11,757			20.0	24.0	4.0	0.96	4.0	0.96		Slimey
					24.0	29.0	5.0	0.04				Clay

APPENDIX B
PROVEN RESERVE SUMMARY TABLE

TABLE I - Sheets 1A, 2A, 3, 4 and 5

1981 Proven Reserves from report by

Watts, Griffis and McOuat

Block No.	Tonnage	Grade (Oz Ag/ton)
1	17,580	5.17
2	23,740	3.46
3	23,430	3.34
4	15,740	3.21
5	14,700	2.30
6	25,950	2.24
7	20,790	1.92
8	19,620	1.75
9	18,820	1.68
10	18,940	1.63
11	20,600	1.62
12	16,300	1.60
13	17,100	1.58
14	18,200	1.48
15	18,270	1.47
16	15,440	1.46
17	17,400	1.43
18	21,220	1.43
19	19,000	1.38
20	19,800	1.36
21	16,360	1.35
22	16,300	1.35
23	17,400	1.30
24	16,050	1.28
25	19,310	1.26
26	17,160	1.26
27	17,590	1.26
28	18,330	1.25
29	16,110	1.25
30	19,740	1.23
31	18,200	1.23
32	24,050	1.23
33	17,960	1.20
34	17,100	1.20
35	18,270	1.19
36	19,930	1.19
37	16,170	1.19
38	14,820	1.18
39	16,730	1.18
40	16,240	1.17
41	19,000	1.17
42	17,530	1.16
43	18,880	1.16
44	16,730	1.15
45	16,240	1.14
46	16,730	1.13

Block No.	Tonnage	Grade (oz Ag/ton)
47	21,090	1.09
48	16,970	1.09
49	21,160	1.08
50	19,740	1.07
51	16,670	1.06
52	19,070	1.02
53	18,940	1.00
54	20,790	0.99
55	21,030	0.98
56	20,970	0.94
57	18,200	0.94
58	15,070	0.92
59	15,310	0.82
Total	1,086,620	1.48

TABLE 2 - Sheet 1A

1987 Sonic Drilling above 15 feet

Hole No.	Tonnage	Grade (oz Ag/ton)
052-87	2,119	1.29
053-87	6,625	0.98
054-87	3,137	1.31
055-87	2,658	1.47
067-87	3,926	1.12
068-87	1,718	1.34
136-87	4,638	1.42
137-87	2,127	1.71
139-87	12,171	0.90
140-87	9,038	1.04
141-87	1,112	1.74
142-87	--	--
205-87	474	1.30
206-87	1,178	1.30
HA-029	1,840	0.98
HA-030	654	1.74
051-87	1,223	1.13
HA-057	957	1.35
058-87	2,846	1.20
HA-070	1,350	0.77
HA-096A	184	1.07
HA-097	368	1.29
HA-098	213	1.29
143-87	957	1.22
144-87	147	1.46
HA-473	110	0.76
HA-475	258	2.06
Total	62,028	1.15

TABLE 3 - Sheet 2A

1987 Sonic Drilling above 15 feet

Hole No.	Tonnage	Grade (oz Ag/ton)
135-87	5,644	1.45
549-87	9,938	2.90
550-87	2,797	0.74
551-87	9,325	1.41
552-87	2,454	0.80
HA-174	61	0.87
HA-181	147	0.96
HA-190	245	0.99
547-87	1,595	0.47
548-87	3,926	1.68
642-87	2,781	1.11
Total	38,913	1.67

TABLE 4 - Sheet 1B

1981 Drive-Pipe Holes below 15 feet

Hole No.	Tonnage	Grade (oz Ag/ton)
001-81	23,370	1.79
002-81	2,888	0.95
003-81	3,384	0.94
004-81	4,776	4.12
005-81	7,721	1.19
006-81	3,075	1.17
040-81	7,380	1.09
044-81	7,688	1.13
047-81	3,075	1.28
048-81	7,380	1.50
049-81	5,810	0.96
057-81	1,434	0.62
060-81	6,997	0.77
062-81	6,403	1.41
064-81	3,807	0.92
071-81	2,276	0.97
073-81	3,690	1.12
082-81	3,251	2.69
085-81	5,535	1.16
093-81	7,111	1.11
097-81	8,303	1.69
105-81	9,220	0.72
113-81	5,937	0.91
Total	140,511	1.37

TABLE 5 - Sheet 1B

1987 Sonic Drilling below 15 feet

Hole No.	Tonnage	Grade (oz Ag/ton)
SDK-S1	9,840	2.14
SDK-S2	4,533	1.07
SDK-S3	9,508	0.74
007-87	667	1.25
042-87	4,524	1.29
043-87	1,082	1.09
050-87	1,343	2.21
063-87	5,228	0.78
070-87	6,458	1.49
072-87	6,150	1.34
075-87	3,152	2.26
086-87	13,838	1.22
087-87	4,305	1.21
088-87	3,350	0.98
089-87	9,686	1.25
090-87	22,448	1.53
091-87	20,603	1.41
092-87	18,450	1.60
096-87	1,619	1.00
098-87	18,450	1.28
099-87	5,125	1.47
100-87	4,407	1.70
100-87A	6,112	1.31
101-87	3,770	1.31
102-87	15,375	1.28
106-87	6,150	0.94
109-87	5,829	0.81
112-87	1,628	1.79
115-87	585	1.03
118-87	3,688	1.44
201-87	5,535	0.74
202-87	11,450	0.85
203-87	4,327	0.70
204-87	12,300	1.28
567-87	1,413	1.24
Total	252,928	1.30

TABLE 6 - Sheet 2B

1987 Sonic Drilling below 15 feet

Hole No.	Tonnage	Grade (oz Ag/ton)
SDK-S4	7,565	1.39
SDK-S5	4,759	1.69
DP-035	9,835	1.64
008-87	3,998	1.13
009-87	16,303	1.52
011-87	6,871	0.95
025-87	16,054	0.80
029-87	18,522	1.11
036-87	8,394	1.57
038-87	548	0.66
116-87	7,108	2.22
119-87	939	0.97
121-87	770	2.26
123-87	13,917	1.64
124-87	680	1.10
125-87	1,401	1.26
126-87	13,829	1.34
128-87	6,541	0.73
129-87	11,685	2.08
130-87	17,220	1.39
131-87	9,840	0.57
132-87	1,813	1.22
134-87	17,141	1.86
135-87	19,342	1.76
547-87	5,847	1.20
548-87	15,375	1.48
549-87	1,413	1.67
551-87	3,935	0.99
558-87	2,792	0.84
559-87	1,586	1.01
Total	246,023	1.41

APPENDIX A-I

DRAWINGS BY

WATTS GRIFFIS & MCOUAT LIMITED

Revised by Robert L.V. Ekstrom

Drawing Title	Sheet Number
Tailings Plan	1B
Tailings Plan	2B
Tailings Reserve (Surface to 15′ Depth)	1A
Tailings Reserve (Surface to 15′ Depth)	2A
Tailings Reserve (Surface to Bottom)	3
Tailings Reserve (Surface to Bottom)	4
Tailings Reserve (Surface to Bottom)	5

APPENDIX B

Metallurgical Testwork and Head
Assays - Witteck Development Inc.

WITTECK DEVELOPMENT INC.
2640 South Sheridan Way, Mississauga, Ontario, Canada L5J 2M4
Telephone: (416) 823-7381 Telex: 06-982328

METALLURGICAL TESTWORK ON SANDY K TAILINGS

PHASE I REPORT

Project 87-5271
for

Watts, Griffis, & McOuat
8 King St. East
Suite 400
Toronto, Ontario
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April 16, 1987

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1.0 SUMMARY

Preliminary metallurgical testwork was conducted on Sandy K tailings samples to provide data for project assessment. The test program included cyanidation, filtration and preg robbing studies.

Most of the testwork was conducted on a sample composite averaging 1.57 oz/ton Ag. The "80% passing" particle size of the material was 385 microns (about 35 mesh). No mineralogical examination was undertaken, however silver distribution on an assay basis was 44.2% retained on 48 mesh and 12.3% passing 325 mesh.

Cyanidation of unground tailings yielded silver extractions averaging 69.4%. However, silver extraction was increased to 87.0% by grinding in a laboratory ball mill for 10 minutes. A review of project economics by Watts Griffis and McOuat (WGM) indicated that a mill product of 80% passing 145 microns might be the minimum allowable grind size. Testwork at this grind size yielded silver extractions averaging 78.8%.

Initial leaching tests were conducted in agitated vats and at pH 10.0 to ensure maximum silver extraction. This resulted in inadequate sand suspension, excessive cyanide consumption and extremely poor residue filtration. All subsequent work was carried out on a bottle rolls at higher pH. Cyanide consumption at pH 10.5 varied from 0.36 to 1.00 lb/ton, while consumptions at pH 11.2-11.4 were less than 0.25 lb/ton. There was no notable decrease in silver extraction with increased leach pH, however this requires verification through additional work.

WDI

Cyanidation of an unground 40/60 mixture of slimes and tailings yielded an overall silver extraction of 79.4%. This indicated that slimes do not adversely affect the cyanidation of tailings, and that a high recovery of silver may be expected from the slimes component alone, (94.3% in this case). The consumption of cyanide during this test at 2.5 lb/ton was high compared to that for a normal tailings or sands leach.

Preg robbing tests were carried out on a humus sample using synthetic silver-cyanide solution. The humus did not adsorb silver from solution but it did consume 27 lb/ton of lime.

Testwork indicated that the minimum pH for successful residue filtration is 10.3-10.5. The residue is unusual in that it consists of two distinct fractions.

- o A sands fraction which is difficult to suspend, and
- o A slimes or colloidal precipitate fraction which can blind the filter media.

Filtration responded to lime addition, flocculant use and mode of slurry contact. Solids filtration rates up to 50 $lb/ft^2/hr$ were achieved.

Work during this phase was only exploratory in nature- to determine major process parameters and sensitivities. Further testwork is recommended in the following areas prior to final project assessment:

1. Tailings grindability - bond work index determination for mill sizing.
2. Optimization of leach conditions- pH, cyanide level, reagent consumptions.
3. Residue filtration
   - characterization of slimes/colloid component
   - effect of desliming on filterability
   - sufficient data to allow specification of filter type and size.

2.0 INTRODUCTION

Following acceptance of proposal No. 1002 by W. Fredenburg of Watts, Griffis and McOuat, Witteck Development began testwork on the Sandy K tailings samples.

The Sandy K tailings deposits are located near the town of Gowganda, Ontario. Due to the low silver grade, operating parameters such as degree of grinding, extraction and reagent consumptions are critical to the economics of the project. Therefore, testwork was to focus on achieving high silver extraction with minimum grinding and reagent use.

The preliminary work reported herein constitutes the first phase of investigations. A second tailings composite will be prepared for more definitive Phase II studies to commence in April, 1987.

3.0 TEST PROCEDURES AND RESULTS

3.1 Sample Preparation

Forty-two samples of Sandy K tailings were received on January 26, 1987 and designated WDI #87-006. Four composites (A through D) were prepared using equal weights of each sample without size reduction. Each composite was assayed for silver. Composite E was later prepared by combining composites A through D.

Four slimes samples and one humus sample were received on March 18, 1987 and designated WDI #87-017. Each was filtered, screened at 325 mesh and each fraction was analyzed for silver.

A complete list of sample numbers is included in Appendix I.

3.2 Head Assays

Head assays for the tailings or sands composites are presented in Table 1. Slimes and humus assays are shown in Table 2. Sands Composite E averaged 1.57 oz/ton Ag, with a range of 1.25-1.96 oz/ton Ag from different sub-samples and assay trials. This range was larger than is generally acceptable. After a concerted effort was made to alleviate the problem, it was concluded that the intrinsic mineralogical distribution of silver in the sands was the source of sub-sampling difficulties.

The head assay of slimes composite F was 1.57 oz/ton Ag. Composite G (60% sands Composite E, 40% slimes composite) was 1.58 oz/ton Ag. The assays of the individual slimes samples received are listed in Table 2. Screening at 325 mesh revealed that the silver and weight distributions were similar.

The grade of the single humus sample was 0.64 g/ton Ag.

1010A

Table 1

TAILINGS COMPOSITES - HEAD ASSAYS

Composite/ Sample No./Assay Trial	I		II		III	IV	V	VI	Avg.
	1	2	1	2					
A	1.18	1.33	1.40	1.27	-	-	-	-	1.30
B	0.92	0.93	1.14	1.05	-	-	-	-	1.01
C	1.81	1.99	2.16	2.07	-	-	-	-	2.00
D	1.18	1.23	1.14	1.18	-	-	-	-	1.18
E	1.25	1.35	1.34	1.57	1.61	1.60	1.96	1.52	1.57
Composite G*	-	-	-	-	-	-	-	-	1.58

*Prepared with 60% composite E, 40% slimes composite F.

Table 2

HUMUS AND SLIMES HEAD ASSAYS

Sample	Type	+325 Mesh			-325 Mesh			Overall Ag oz/ton
		Wt %	Ag, oz/ton	% Ag Dist	Wt %	Ag, oz/ton	% Ag Dist	
548 12'-20'	Slimes	38.2	0.61	11.2	61.8	3.00	88.8	2.09
548 5'-12'	Slimes	10.0	0.96	5.2	90.0	1.96	94.8	1.87
032 10'-16'	Slimes	38.4	1.09	34.8	61.6	1.27	65.2	1.21
547 9'-14'	Slimes	42.7	0.49	71.1	57.3	0.15	28.9	0.38
Composite F	*Slimes	-	-	-	-	-	-	1.58
87-550 0'-6'	Humus	-	-	-	-	-	-	0.64

*Prepared from equal weights of 548 12' - 20', 548 5' - 12' and 032 10' - 16'.

3.3 Screen Analyses

A summary of particle size disturbances can be found in Table 3. Screening of fractions finer than 200 mesh was by wet screening, followed by RoTapping of the dried oversize. Although composites A and B were referred to as "sands" and composites C and D were called "slimes", the particle size distribution of A, B, and D were quite similar. Composite D was somewhat coarser. The eighty percent passing size (P_{80}) of Composites A through E were 320, 295, 420, 295, and 385 microns, respectively.

A silver distribution by screen size was determined for Composite E. The results are shown in Table 4. Silver was concentrated in the coarser fractions. Consideration of desliming at 325 mesh would result in rejection of 19% of the weight and 12% of the silver.

3.4 Cyanidation

Agitated vat cyanidation tests were conducted on unground Composites A through D. The heavy sand was difficult to suspend, and agitation was therefore unsatisfactory. Subsequent tests were conducted in rolled bottles. Test conditions and mass balances for all leaching work are included in Appendix II.

3.4.1 Agitated Leaches

The agitated leaches were conducted without grinding at 40% solids, pH 10.0, 1.0 gpL NaCN for 70 hours. The results are summarized in Table 5.

Silver extractions were 66-71% for composites A, B and C. Extraction for composite D was 59%. Average extraction was 66.6%. Leaching was essentially complete after 48 hours. High cyanide consumptions were a result of a combination of low pH, low pulp density, and the required vigorous agitation.

Table 3

PARTICLE SIZE DISTRIBUTIONS

Cumulative % Passing

Size (Mesh)	Comp. "A"	Comp. "B"	Comp. "C"	Comp. "D"	Comp. "E"	Comp. Grind 5 min	'E' Time 10 min	Slimes 548 12'-20'	Slimes 548 5'-12'	Slimes 547 9'-14'	Slimes 032 10'-16'
20	99.5	-	99.4	-	-	-	-	-	-	-	-
28	96.4	99.1	94.6	98.1	97.8	-	-	-	-	-	-
35	90.3	94.4	78.0	90.2	91.0	-	-	-	-	-	-
48	76.3	81.2	65.9	82.1	81.1	-	-	-	-	-	-
65	62.6	66.3	48.3	67.2	66.2	93.8	99.7	-	-	-	-
100	42.3	45.5	26.8	48.8	50.7	83.6	98.5	-	-	-	-
150	27.2	31.3	17.0	32.0	38.9	64.5	90.3	-	-	-	-
200	16.7	19.5	9.9	21.8	28.0	44.2	68.7	-	-	-	-
270	-	-	-	-	21.4	34.4	52.6	-	-	-	-
325	-	-	-	-	18.9	30.0	46.7	61.8	90.0	57.3	61.6
P_{80} (microns)	320	295	420	295	385	145	90	-	-	-	-

Table 4

SILVER DISTRIBUTION - COMPOSITE "E"

Size (Mesh)	Ag Assay oz/ton	Discrete % Dist Wt	Discrete % Dist Ag	Cum % Ret Wt	Cum % Ret Ag	Cum % Pass Wt	Cum % Pass Ag
28	1.97	2.2	3.3	2.2	3.3	97.8	96.7
35	3.97	6.8	21.4	9.0	24.7	91.0	75.3
48	2.51	9.9	19.5	18.9	44.2	81.1	55.8
65	2.08	14.9	18.9	33.8	63.1	66.2	36.9
100	2.17	15.5	10.7	49.3	73.8	50.7	26.2
150	1.66	11.8	5.7	61.1	79.5	38.9	20.5
200	1.52	10.9	4.7	72.0	84.2	28.0	15.8
270	0.93	6.6	2.5	78.6	86.7	21.4	13.3
325	0.34	2.4	1.0	81.0	87.7	19.0	12.3
-325	0.82	19.0	12.3	100.0	100.0	0.0	0.0
Total	(1.27)	100.0	100.0				

Table 5

AGITATED LEACH RESULTS

Test No.	Feed (As Received)	P_{80} (microns)	Consumptions (lb/ton) NaCN	Lime	Silver Calc.Hd (oz/ton)	Assayed Hd. (oz/ton)	Extraction (%)
AG-1	A Sands	320	7.22	0.70	1.59	1.29	68.2
2	B Slimes	295	9.44	2.82	1.11	1.01	66.6
3	C Sands	420	8.70	2.78	2.42	2.00	7.16
4	D Slimes	295	6.00	0.82	1.40	1.18	59.8
Average					1.63	1.37	66.6

3.4.2 Bottle Leaches

The two major objectives of the bottle leach test series were to reduce cyanide consumptions to acceptably low levels and to investigate the effect of grinding. Tests were performed at 50% solids and pH 10.3-10.5 for 52 hours. Grinding was conducted with a sized ball charge for five or ten minutes. Test results are shown in Table 6 and Figure 1.

Cyanide consumptions were less than 1.0 lb/ton. In tests BT-5 and BT-7 excessive amounts of lime were added in error. However, this resulted in lower cyanide consumption and no notable decrease in silver dissolution. Silver extraction without grinding averaged 69.4%, which compares well with the average of 66.4% in the agitated leach tests. Leaching was essentially complete after approximately 30 hours.

3.4.3 Effect of Grinding

The effect of grinding is shown in Figure 2. Grinding for 5 minutes (from P_{80} = 385 to 145 microns) increased extraction to 86-87%. Cyanide consumption also increased with finer grinding.

3.4.4 Effect of Slimes

One bottle leach was conducted on Composite G (60% Composite E/40% slimes Composite F). The test was conducted as per the other bottle tests, with no grinding. The results are shown in Table 6 and Figure 3.

Silver extraction reached a maximum of 79.4% after 30 hours of leaching. Assuming a 69.4% extraction of silver from normal tailings sands, a mass balance on this leach showed high extraction (94.3%) from the slimes component alone. Reagent consumptions of 2.52 lb/ton cyanide and 6.96 lb/ton lime were high compared to sands leaching. Therefore, deliberate introduction or entrainment of slimes into the plant feed will increase reagent consumptions but not hinder silver dissolution.

3.5 Humus and Preg Robbing

One test was conducted to determine whether humus contamination would rob silver from solution. A cyanide-lime solution was spiked with 42.7 ppm Ag, and added to 24 gpL of humus. The slurry was agitated for five hours at pH 10.5 and 1.0 gpL NaCN. A detailed mass balance is shown in Appendix III.

The filtrate contained 43.4 ppm Ag, indicating no silver adsorption. Preg robbing by the humus should not therefore be a concern. Cyanide consumption was 1.02 lb/ton. The humus was, however, a lime consumer - 27.2 lb/ton being used to maintain solution pH.

3.6 Environmental Assays

Pregnant solution from Tests BT 4-7 was analyzed for base metals, trace elements and cyanide complexes. The results are shown in Tables 7 and 8.

The total cyanide and free cyanide levels were 473 and 460 ppm respectively. Iron, copper, zinc, antimony, molybdenum, and nickel as well as thiocyanate and cyanate were each less than 10 ppm. The level of arsenic was 10.3 ppm by semi quantitative analysis. Cyanide destruction will be eased by the fact that most of the cyanide is present as free cyanide.

Table 6

BOTTLE LEACH RESULTS

Test #	Grind Time (min)	Grind P_{80} (microns)	Reag. Consumptions (lb/ton) pH	NaCN	Lime	Silver Preg. (ppm)	Res. (oz/ton)	Calc.Hd (oz/ton)	Extracted (%)
BT-1	-	385	10.5	0.36	0.84	34.6	14.49	1.46	71.2
4	-	385	10.5	0.98	1.08	36.8	17.3	1.59	68.3
6	-	385	10.5	0.52	0.68	35.0	16.3	1.52	68.7
2	5	145	10.5	0.68	1.26	42.3	12.3	1.74	79.4
5	5	145	11.2*	0.24	2.50	41.4	10.2	1.64	81.8
7	5	145	11.4*	0.14	3.26	41.6	14.7	1.76	75.4
3	10	90	10.5	1.00	0.96	50.4	8.8	1.85	86.2
8	10	90	10.5	1.04	0.94	48.3	7.09	1.47	87.8
9**	-	-	10.5	2.52	6.96	42.4	13.25	1.87	79.4

* pH 10.5 intended; too much lime added, resulting in lower NaCN consumption.

** Feed in BT-9 was Composite G (1.58 oz/ton Ag); Feed to all other tests - Composite E (1.57 oz/ton)

Table 7

ENVIRONMENTAL ASSAYS

BASE METALS AND CYANIDE COMPLEXES

Element	Level ppm
Cu	9.96
Ni	2.64
Fe	4.50
Zn	3.97
Total CN	473
WAD CN*	473
FREE CN	460
Thiocyanate	0.11
Cyanate	9.0

* WAD CN = Weak acid dissociable cyanide.

Table 8

ENVIRONMENTAL ASSAYS
SEMI QUANTITATIVE SCAN

Element	Level ppm	Element	Level ppm
Al_2O_3	1.7	Cu	12.9
Fe_2O_3	6.0	Hg	LT 0.01
CaO	77.4	La	0.02
MgO	0.43	Mo	0.52
Na_2O	756	Nb	0.03
K_2O	34	Ni	2.28
TiO_2	LT 0.001	Pb	0.10
MnO	0.06	S	32.88
P_2O_5	0.9	Sb	1.2
		Se	0.25
Ag	41.81	Sn	0.06
As	10.3	Sr	0.14
B	0.1	Te	0.07
Ba	0.007	Th	0.08
Be	0.08	U	0.2
Bi	0.06	V	0.1
Cd	0.26	W	0.07
Ce	0.08	Y	0.1
Co	7.8	Zn	4.0
Cr	0.01	Zr	0.06

Figure 1

SILVER EXTRACTION VERSUS

CYANIDATION TIME

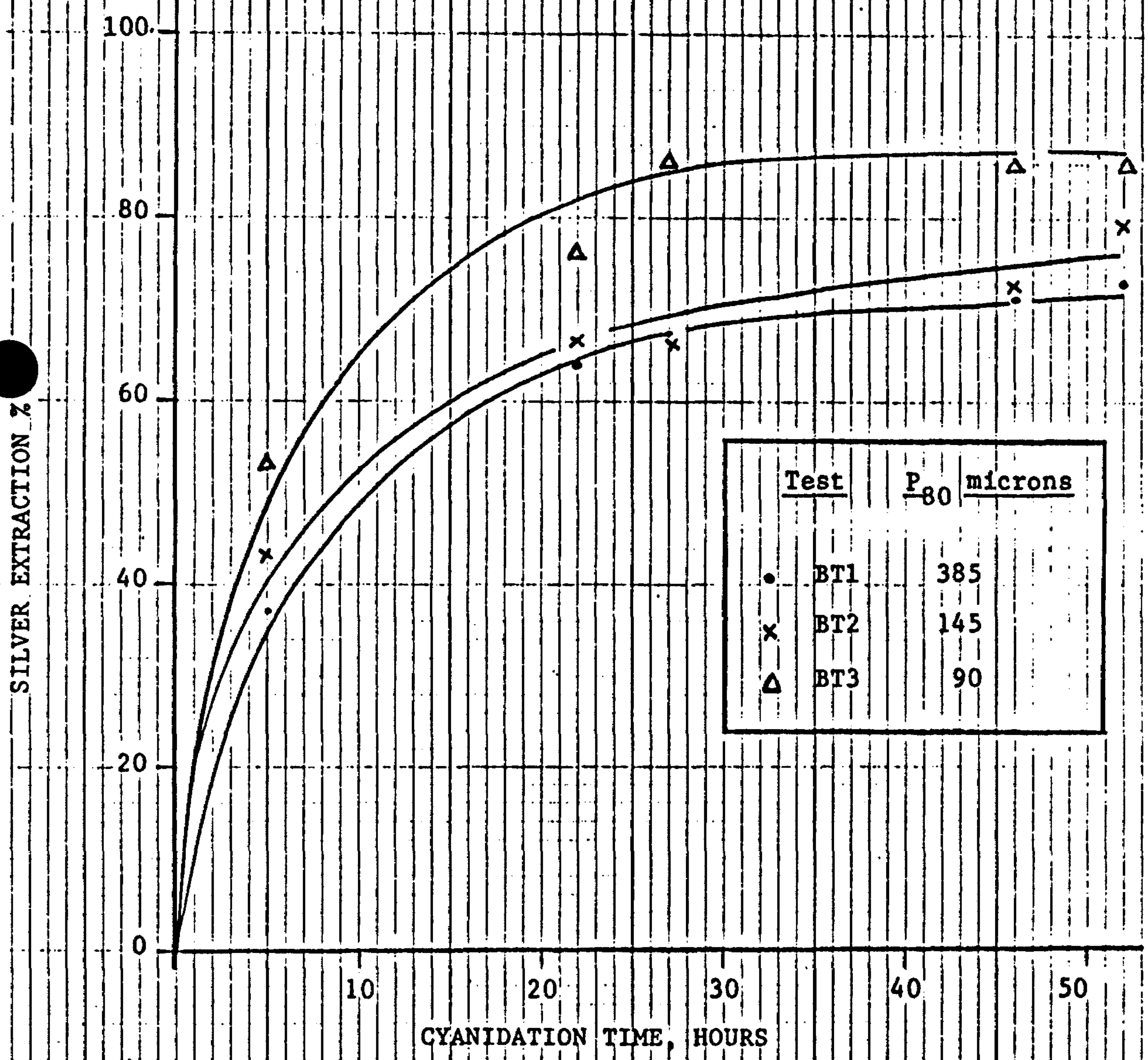


Figure 2

SILVER EXTRACTION VERSUS GRIND SIZE

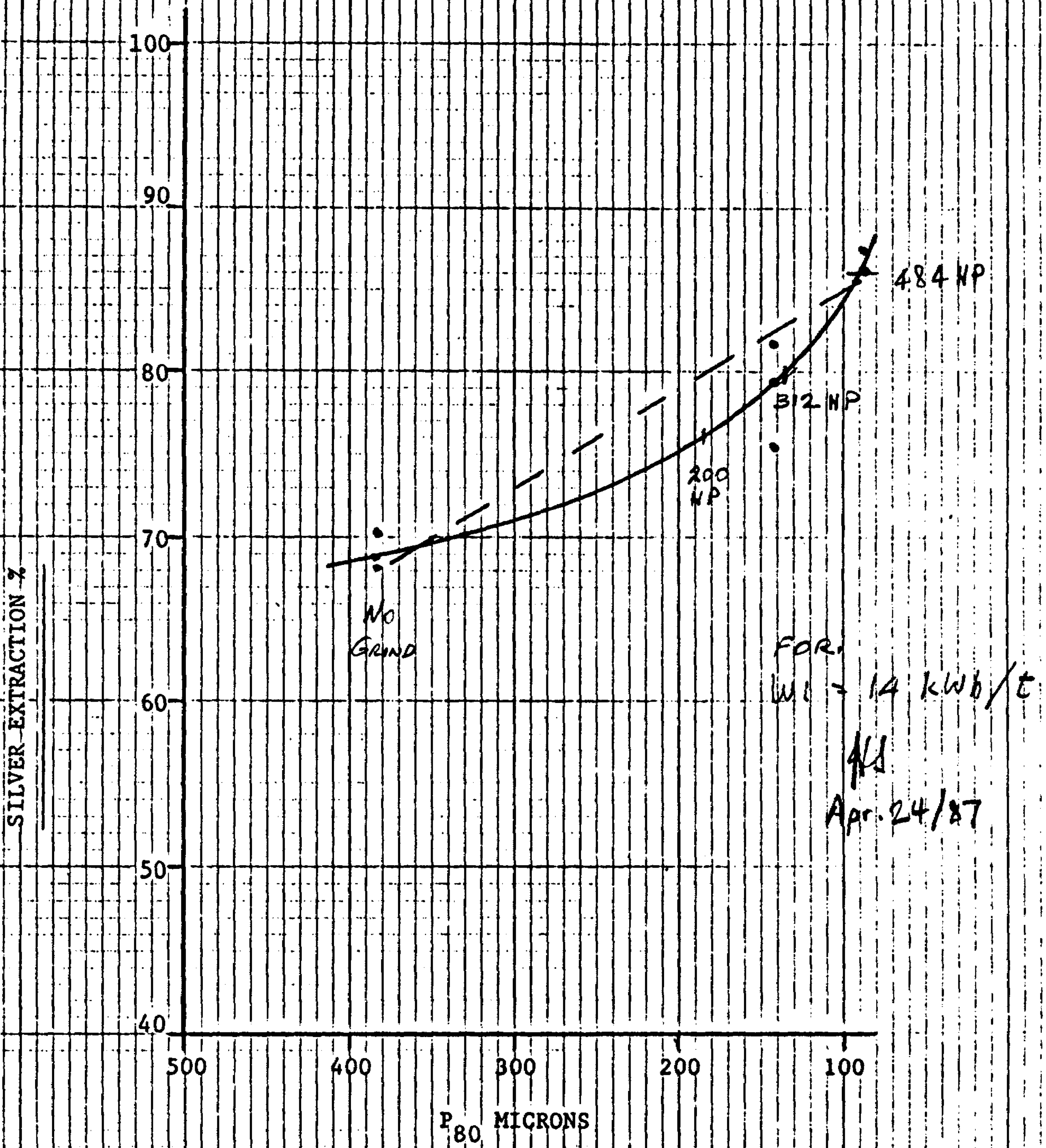
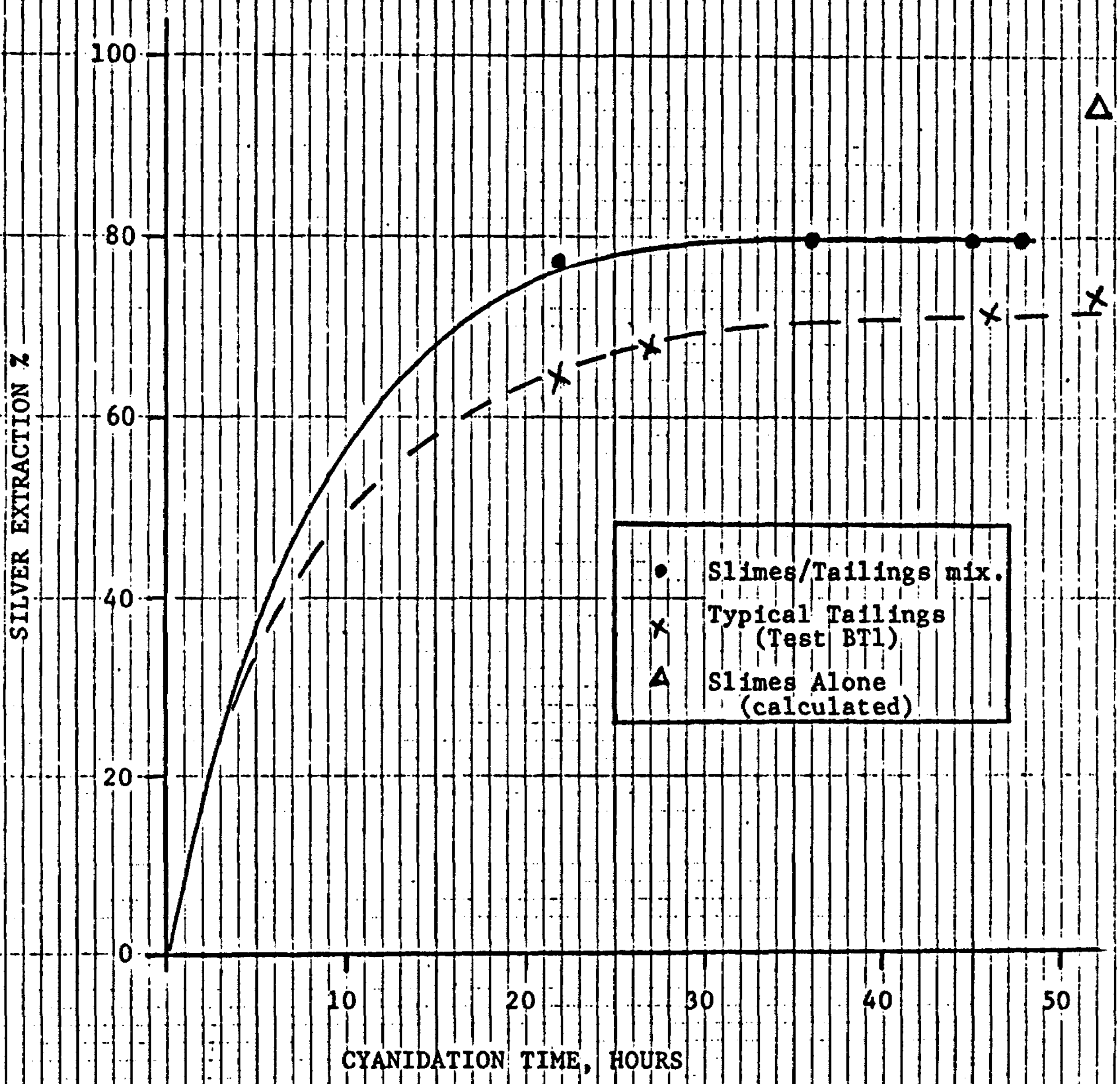


Figure 3

EFFECT OF SLIMES ON SILVER EXTRACTION



3.7 Filtration of Leach Residue

Numerous filtration data were obtained on cyanidation leach slurries using both Buchner funnel and vacuum filter leaf (0.1 sq. ft. Eimco) techniques. The slurry was introduced to the leaf by both dipping (normal) and pouring to simulate a rotary vacuum filter and a belt filter, respectively. The data are listed in Table 9 and the key findings are highlighted below:

1. Filtration rates at less than pH 10.3 were extremely slow (i.e. less than 2 $lb/ft^2/hr$). This was attributed to cloth blinding due to mineral slimes and/or iron precipitate. Polymeric flocculants did have an appreciable influence.

2. Increasing the pH above 10.3 with lime resulted in a dramatic increase in both settling and filtration rates due to flocculation of slimes. A further but modest increase in the filtration rate was noted by raising the pH above 11.0.

3. Cake formation was virtually non-existent using the normal vacuum leaf dip method. Increasing the pH did not measurably improve the situation. This indicates that a rotary vacuum filter may be inappropriate for this application.

4. Successful filtration tests could only be achieved by pouring the leach slurry onto the vacuum leaf positioned horizontally. Presumably the coarse sands settled quickly to provide a three dimensional filter media for the colloidal slimes. This technique and the associated filter data are applicable to vacuum belt filter operation.

5. The maximum filtration rates obtained were 50 $lb/ft^2/hr$ of dry solids. Therefore, using a 25% safety factor, a rate of 38 $lb/ft^2/hr$ may be achievable in practice.

Table 9

FILTRATION TEST RESULTS

Test #	Feed		pH	Form Time (min)	Filter Cake		(1)Filtration Rates		Method
	Comp	% Solids			cm	$\%H_2O$	Solids $lb/ft^2/hr$	Filtrate $USG/ft^2/hr$	
SKF-1	A,C	40	10.0	33.5	0.8	14.8	0.7	0.1	Leaf-Horiz. filter
2	B,D	40	10.0	30.0	0.8	-	-	-	"
3	B,D	40	10.0	1.3	0.8	-	32.9	5.2	" -Deslimed
4	B,D	40	10.0	45.0	-	-	2.6	0.5	Buchner
5	B,D	40	11.0	2.0	-	-	58.6	10.5	" -Lime as floc
6	B,D	40	10.0	30.0	-	-	3.9	0.7	" -Anionic floc
7	B,D	40	10.0	40.0	-	-	2.9	0.5	" -nonionic floc
8	B,D	40	10.0	40.0	-	-	2.9	0.5	" -cationic floc
9	E	50	10.5	1.5	-	-	-	-	Leaf-Rot. drum filter
12	E	50	10.5	1.3	0.7	15.0	29.3	2.8	Leaf-horiz., filter
11	E	50	10.5	1.8	0.9	15.7	32.3	3.3	"
10	E	50	10.5	2.0	1.1	15.4	34.8	3.8	"
13	E	50	10.5	10.3	1.6	13.1	9.8	1.0	"
14	E	50	11.5	1.5	-	-	-	-	Leaf-Rot.drum Filter; lime
15	E	50	11.5	1.5	1.3	15.6	49.0	5.3	Leaf-Horiz, filter, lime
16	E	50	10.5	1.5	1.2	14.7	50.3	5.3	Leaf-Horiz.filter; anionic floc

(1) Note: Filtration rates were calculated on the basis of a typical vacuum filter cycle, i.e. Total cycle time = 3.0 x form time.

4.0 CONCLUSIONS AND RECOMMENDATIONS

1. The tailings samples were similar to beach sand in appearance, having an 80% passing size of 385 microns (about 35 mesh).

2. There was concentration of the silver in the coarser fractions of the tailings. The +48 mesh fraction contained 44.2% of the silver in 18.9% of the weight.

3. Silver extraction was dependent on the fineness of grind. Cyanidation without grinding yielded silver extractions of 68-71%. Grinding to 80% -145 microns and 80% -90 microns resulted in silver extractions of 75-81% and 86-87% respectively.

4. Preliminary calculations indicated that a grind size of 80% passing 145 microns would be roughly equivalent to the product obtained by milling 1000 tpd in a 200 hp ball mill. Further testwork, including Bond Work Index determination, would be required to confirm these results.

5. Consumptions of lime and cyanide varied greatly during the tests conducted due to differing test techniques and leach conditions. The following reagent consumptions are considered appropriate for calculating operating costs at this point:

$Ca(OH)_2$	2 lb/ton
NaCN	0.5 lb/ton

6. Humus did not rob silver from leach solution but it was a high lime consumer.

7. The level of base metals in the leach pregnant solution was low; less than 40 ppm in total.

8. Fine dispersed material in the leached tailings resulted in lower than desirable filtration rates. This material may be mineral slime or amorphous chemical precipitate. It is recommended that this "slime" be characterized to provide confidence in process technical control.

9. Liquid-solid separation of leach residue was highly dependent on leach slurry pH. Below pH 10.3, slimes settling rates and slurry filtration rates were very low. An increase of pH from 10.3 to 10.5 resulted in successful settling of the slimes and a 10-20 fold increase in filtration rates. Indications from the test data are that plant filtration rates of 38 $lb/ft^2/hr$ dry solids and 4.0 $USG/ft^2/hr$ filtrate might be possible.

10. Filter cake formation was negligible using the laboratory filter leaf in a manner to simulate a rotary vacuum filter, i.e., vertical pick-up position. This indicated that the slimes, although somewhat flocculated by lime, blinded the filter cloth. With the leaf in a horizontal position, the coarse sands settled rapidly to form a multi-dimensional filter media. Available data indicate that a horizontal vacuum belt filter should be considered.

11. It is highly recommended that further filtration testwork be carried out prior to equipment purchase. It has not been established that a vacuum drum filter will be successful in this application. The rapid settling of residue sands (especially for unground feed) is another factor which must be considered in filter selection. The effects of lime, polymeric flocculants and desliming on filtration rate also deserve further study.

APPENDIX I

SAMPLES USED IN METALLURGICAL TESTWORK

TAILINGS SAMPLES

COMPOSITE A "SANDS" Sample #		COMPOSITE B "SLIMES Sample #	
201	0'-5'	201	20'-25'
	5'-10'		25'-30'
	10'-15'		30'-33'
106	0'-5'	106	15'-20'
	5'-10		20'-25'
	10'-15'		25'-30'
			30'-35'
203	0'-5'	203	10'-15'
	5-'10'		15'-20'
	10'-15'		20'-25'
202	0'-5'	202	15'-20'
	5'-10'		20'-25'
	10'-15'		25'-30'

COMPOSITE C "SANDS" Sample #		COMPOSITE D "SLIMES" Sample #	
86	0'-5'	86	15'-24'
	5'-10'		24'-29'
	10'-15'		29'-33'
92	0'-5'	92	20'-25
	5'-10'		25'-30'
	10'-15'		30'-35'
	15'-20'		
87	0'-4'	87	4'-9'
			9'-14'
			14'-19'
			19'-22'

Composite E - Equal weights used of all samples except #201 5'-10' and 201 15'-20'

1010A

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HUMUS AND SLIMES SAMPLES

Sample Designation	Sample Type
548 12' - 20'	Slimes
548 5' - 12'	Slimes
032 10' - 16'	Slimes
547 9' - 14'	Slimes
Composite F *	Slimes
87-550 0' -6'	Humus

*Prepared from equal weights of 548 12' - 20', 548 5' - 12', and 032 10' - 16'.

APPENDIX II

CYANIDATION TEST MASS BALANCES

CYANIDATION TEST SUMMARY

TEST NO. : AG-1

PURPOSE : Agitated leach

FEED : 487.1 grams of Sandy K Composite A

GRIND : —— minutes; —— % solids

PRE-AER : —— hours; —— % solids

LEACH : 48 hours; 40 % solids

REPULP :

WASH : 5 x 200 mLs water - twice solids weight

REAGENTS : Grind - water
Pre-aer -

Leach - NaCN - 1 gpL
- Lime - pH 10.5

- Pb Nitrate -

CONSUMPTIONS (lb/t) NaCN: 7.22 Lime: 0.70 Pb Nitr.

MASS BALANCE

Product	Wt/Vol g/mL	Ag ppm/oz/t	Ag mg	Ag Dist'n
Grind Filtrate 1	570	27.4	15.62	61.1
Filtrate 2 Wash	970	1.34	1.30	5.1
Titration Samples	30		0.52	2.0
Residue	469	17.34	8.13	31.8
Total			25.57	100.0

Ag Extraction: 68.2 % Calc. Head : 1.59 oz/t
Assayed Head : 1.29 oz/t Mass Balance : 123.3 %

0803A

CYANIDATION TEST SUMMARY

TEST NO. : AG-2

PURPOSE : Agitated leach

FEED : 503.5 grams of Sandy K Composite B

GRIND : —— minutes; —— % solids

PRE-AER : —— hours; —— % solids

LEACH : 48 hours; 40 % solids

REPULP :

WASH : 5 x 200 mLs water - twice solids weight

REAGENTS : Grind - water
Pre-aer -

Leach - NaCN - 1 gpL
- Lime - pH 10.0 - 10.5

- Pb Nitrate -

CONSUMPTIONS (lb/t) NaCN: 9.44 Lime: 2.82 Pb Nitr.

MASS BALANCE

Product	Wt/Vol g/mL	Ag ppm/oz/t	Ag mg	Ag Dist'n
Grind Filtrate 1	600	18.4	11.04	59.7
Filtrate 2 Wash	1180	0.74	0.87	4.7
Titration Samples	30	13.3	0.40	2.2
Residue	487	12.7	6.18	33.4
Total			18.49	100.0

Ag Extraction: 66.6 %
Assayed Head : 1.01 oz/t

Calc. Head : 1.11 oz/t
Mass Balance : 109.9 %

0803A

CYANIDATION TEST SUMMARY

TEST NO. : AG - 3

PURPOSE : agitated leach

FEED : 504.3 grams of Sandy K Composite C

GRIND : —— minutes; —— % solids

PRE-AER : —— hours; —— % solids

LEACH : 48 hours; 40 % solids

REPULP :

WASH : 5 x 200 mLs water - twice solids weight

REAGENTS : Grind - water
Pre-aer -

Leach - NaCN - 1 gpL
- Lime - pH 10.0 - 10.5

- Pb Nitrate -

CONSUMPTIONS (lb/t) NaCN: 8.7 Lime: 2.78 Pb Nitr.

MASS BALANCE

Product	Wt/Vol g/mL	Ag ppm/oz/t	Ag mg	Ag Dist'n
Grind Filtrate 1	600	44.1	26.46	63.8
Filtrate 2 Wash	1250	1.72	2.15	5.2
Titration Samples	30	35.3	1.06	2.6
Residue	499	23.52	11.74	28.4
Total			41.41	100.0

Ag Extraction: 71.6 % Calc. Head : 2.42 oz/t
Assayed Head : 2.00 oz/t Mass Balance : 121.0 %

0803A

CYANIDATION TEST SUMMARY

TEST NO. : AG-4

PURPOSE : Agitated leach

FEED : 493 grams of Sandy K Composite D

GRIND : —— minutes; —— % solids

PRE-AER : —— hours; —— % solids

LEACH : 48 hours; 40 % solids

REPULP :

WASH : 5 x 200 mLs water - twice solids weight

REAGENTS : Grind - water
Pre-aer -

Leach - NaCN - 1gpL
- Lime - pH 10.0 - 10.5

- Pb Nitrate -

CONSUMPTIONS (lb/t) NaCN: 6.00 Lime: 0.82 Pb Nitr.

MASS BALANCE

Product	Wt/Vol g/mL	Ag ppm/oz/t	Ag mg	Ag Dist'n
Grind Filtrate 1	680	17.9	12.17	52.0
Filtrate 2 Wash	1200	1.07	1.28	5.5
Titration Samples	30	18.0	0.54	2.3
Residue	488	19.28	9.41	40.2
Total			23.4	100.0

Ag Extraction: 59.8 % Calc. Head : 1.40 oz/t
Assayed Head : 1.18 oz/t Mass Balance : 118.6 %

0803A

CYANIDATION TEST SUMMARY

TEST NO. : BT-1

PURPOSE : Bottle Rolls Test

FEED : 502 grams of Sandy K Composite E

GRIND : —— minutes; —— % solids

PRE-AER : —— hours; —— % solids

LEACH : 48 hours; 50 % solids

REPULP :

WASH : 5 x 200 mLs water - twice solids weight

REAGENTS : Grind —— - water
Pre-aer —— -

Leach - NaCN - 1 gpL
- Lime - pH 10.5

- Pb Nitrate -

CONSUMPTIONS (lb/t) NaCN: 0.36 Lime: 0.84 Pb Nitr.

MASS BALANCE

Product	Wt/Vol g/mL	Ag ppm/oz/t	Ag mg	Ag Dist'n
Grind Filtrate 1	380	34.55	13.13	53.1
Filtrate 2 Wash	1070	2.51	2.69	10.9
Titration Samples	60	29.67	1.78	7.2
Residue	493.4	14.49	7.15	28.8
Total			24.75	100.0

Ag Extraction: 71.2 % Calc. Head : 1.46 oz/t
Assayed Head : 1.57 oz/t Mass Balance : 93.0 %

* Average of and lb/t Ag

0803A

CYANIDATION TEST SUMMARY

TEST NO. : BT-2

PURPOSE : Bottle Rolls Test

FEED : 506 grams of Sandy K Composite E

GRIND : 5 minutes; 65 % solids

PRE-AER : —— hours; —— % solids

LEACH : 48 hours; 50 % solids

REPULP : ——

WASH : 5 x 200 mLs water - twice solids weight

REAGENTS : Grind - water
Pre-aer -

Leach - NaCN - 1 gpL
- Lime - pH 10.5

- Pb Nitrate -

CONSUMPTIONS (lb/t) NaCN: 0.68 Lime: 1.26 Pb Nitr.

MASS BALANCE

Product	Wt/Vol g/mL	Ag ppm/oz/t	Ag mg	Ag Dist'n
Grind Filtrate 1	415	42.26	17.54	59.1
Filtrate 2 Wash	1070	3.57	3.82	12.9
Titration Samples	60	36.67	2.20	7.4
Residue	497.5	12.3	6.12	20.6
Total			29.68	100.0

Ag Extraction: 79.4 % Calc. Head : 1.74 oz/t
Assayed Head : 1.57 oz/t Mass Balance : 110.8 %

0803A

CYANIDATION TEST SUMMARY

TEST NO. : BT-3

PURPOSE : Bottle Rolls Test

FEED : 500 grams of Sandy K Composite E

GRIND : 10 minutes; 65 % solids

PRE-AER : —— hours; —— % solids

LEACH : 48 hours; 50 % solids

REPULP :

WASH : 5 x 200 mLs water - twice solids weight

REAGENTS : Grind - water
Pre-aer -

Leach - NaCN - 1gpL
- Lime - pH 10.5

- Pb Nitrate -

CONSUMPTIONS (lb/t) NaCN: 1.0 Lime: 0.96 Pb Nitr.

MASS BALANCE

Product	Wt/Vol g/mL	Ag ppm/oz/t	Ag mg	Ag Dist'n
Grind Filtrate 1	290	50.44	14.62	47.1
Filtrate 2 Wash	1010	9.22	9.31	30.0
Titration Samples	60	47.0	2.82	9.1
Residue	489.4	8.75	4.28	13.8
Total			31.03	100.0

Ag Extraction: 86.2 % Calc. Head : 1.85 oz/t
Assayed Head : 1.57 oz/t Mass Balance : 117.8 %

0803A

CYANIDATION TEST SUMMARY

TEST NO. : BT-4

PURPOSE : Bottle Rolls Test

FEED : 505 grams of Sandy K Composite E quartz

GRIND : — minutes; — % solids

PRE-AER : — hours; — % solids

LEACH : 48 hours; 50 % solids

REPULP :

WASH : 5 x 200 mLs water - twice solids weight

REAGENTS : Grind - water
Pre-aer -

Leach - NaCN - 1gpL
- Lime - pH 10.5

- Pb Nitrate -

CONSUMPTIONS (lb/t) NaCN: 0.98 Lime: 1.08 Pb Nitr.

MASS BALANCE

Product	Wt/Vol g/mL	Ag ppm/oz/t	Ag mg	Ag Dist'n
Grind Filtrate 1	425	36.8	15.64	57.0
Filtrate 2 Wash	1050	2.47	2.60	9.5
Titration Samples	15	33.16	0.49	1.8
Residue	502	17.3	8.68	31.7
Total			27.41	100.0

Extraction: 68.3 % Calc. Head : 1.59 oz/t
Assayed Head : 1.57 oz/t Mass Balance : 101.3 %

0803A

CYANIDATION TEST SUMMARY

TEST NO. : BT5

PURPOSE : Bottle Rolls Test

FEED : 508 grams of Sandy K Composite E quarter

GRIND : 5 minutes; 65 % solids

PRE-AER : —— hours; —— % solids

LEACH : 48 hours; 50 % solids

REPULP :

WASH : 5 x 200 mLs water - twice solids weight

REAGENTS : Grind - water
Pre-aer -

Leach - NaCN - 1gpL
- Lime - pH 10.5 - 11.0

- Pb Nitrate -

CONSUMPTIONS (lb/t) NaCN: 0.24 Lime: 2.5 Pb Nitr.

MASS BALANCE

Product	Wt/Vol g/mL	Ag ppm/oz/t	Ag mg	Ag Dist'n
Grind Filtrate 1	420	41.4	17.39	63.3
Filtrate 2 Wash	1015	4.42	4.49	16.3
Titration Samples	15	40.30	0.60	2.2
Residue	488	10.24	5.0	18.2
Total			27.48	100.0

Ag Extraction: 81.8 % Calc. Head : 1.64 oz/t
Assayed Head : 1.57 oz/t Mass Balance : 104.4 %

0803A

CYANIDATION TEST SUMMARY

TEST NO. : BT-6

PURPOSE : Bottle Rolls Test

FEED : 506 grams of Sandy K Composite E quarter

GRIND : — minutes; — % solids

PRE-AER : — hours; — % solids

LEACH : 48 hours; 50 % solids

REPULP :

WASH : 5 x 200 mLs water - twice solids weight

REAGENTS : Grind - water
Pre-aer -

Leach - NaCN - 1 gpL
- Lime - pH 10.5 - 10.7

- Pb Nitrate -

CONSUMPTIONS (lb/t) NaCN: 0.52 Lime: 0.68 Pb Nitr.

MASS BALANCE

Product	Wt/Vol g/mL	Ag ppm/oz/t	Ag mg	Ag Dist'n
Grind Filtrate 1	330	35.0	11.55	44.9
Filtrate 2 Wash	1040	5.42	5.64	21.9
Titration Samples	15	31.88	0.49	1.9
Residue	493	16.3	8.04	31.3
Total			25.72	100.0

Ag Extraction: 68.7 % Calc. Head : 1.52 oz/t
Assayed Head : 1.57 oz/t Mass Balance : 103.3 %

0803A

CYANIDATION TEST SUMMARY

TEST NO. : BT-7

PURPOSE : Bottle Rolls Test

FEED : 516 grams of Sandy K Composite E quarter 4

GRIND : 5 minutes; 65 % solids

PRE-AER : —— hours; —— % solids

LEACH : 48 hours; 50 % solids

REPULP :

WASH : 5 x 200 mLs water - twice solids weight

REAGENTS : Grind - water
Pre-aer -

Leach - NaCN - 1 gpL
- Lime - pH 10.5 - 11.4

- Pb Nitrate -

CONSUMPTIONS (lb/t) NaCN: 0.14 Lime: 3.26 Pb Nitr.

MASS BALANCE

Product	Wt/Vol g/mL	Ag ppm/oz/t	Ag mg	Ag Dist'n
Grind Filtrate 1	450	41.6	18.72	61.5
Filtrate 2 Wash	1020	3.61	3.68	12.1
Titration Samples	15	41.25	0.62	2.0
Residue	504	14.7	7.41	24.4
Total			30.43	100.0

Ag Extraction: 75.4 % Calc. Head : 1.76 oz/t
Assayed Head : 1.57 oz/t Mass Balance : 112.1 %

0803A

CYANIDATION TEST SUMMARY

TEST NO. : BT-8

PURPOSE : Bottle Rolls Test

FEED : 512 grams of Sandy K Composite E

GRIND : 10 minutes; 65 % solids

PRE-AER : —— hours; —— % solids

LEACH : 48 hours; 50 % solids

REPULP :

WASH : 5 x 200 mLs water - twice solids weight

REAGENTS : Grind - water
Pre-aer -

Leach - NaCN - 1gpL
- Lime - pH 10.3 - 10.5

- Pb Nitrate -

CONSUMPTIONS (lb/t) NaCN: 1.04 Lime: 0.94 Pb Nitr.

MASS BALANCE

Product	Wt/Vol g/mL	Ag ppm/oz/t	Ag mg	Ag Dist'n
Grind Filtrate 1	432	48.3	20.87	70.9
Filtrate 2 Wash	1000	4.96	4.96	16.9
Titration Samples				
Residue	508	7.09	3.6	12.2
Total			29.43	100.0

Ag Extraction: 87.8 % Calc. Head : 1.47 oz/t
Assayed Head : 1.57 oz/t Mass Balance : 93.6 %

0803A

CYANIDATION TEST SUMMARY

TEST NO. : BT-9

PURPOSE : Bottle Rolls Test - Ore/slimes cyanidation

FEED : 506 grams of Composite G (60% Comp E, 40% Comp F)

GRIND : —— minutes; —— % solids

PRE-AER : —— hours; —— % solids

LEACH : 48 hours; 50 % solids

REPULP :

WASH : 5 x 200 mLs water - twice solids weight

REAGENTS : Grind - water
Pre-aer -

Leach - NaCN - 1.0 gpL
- Lime - pH 10.3 - 10.5

- Pb Nitrate -

CONSUMPTIONS (lb/t) NaCN: 2.52 Lime: 6.96 Pb Nitr.

MASS BALANCE

Product	Wt/Vol g/mL	Ag ppm/oz/t	Ag mg	Ag Dist'n
Grind Filtrate 1	357	42.4	15.14	47.1
Filtrate 2 Wash	1015	7.50	7.61	23.7
Titration Samples	60	(45.8)	2.75	8.6
Residue	501	13.25	6.63	20.6
Total			32.13	100.0

Ag Extraction: 79.4 % Calc. Head : 1.85 oz/t
Assayed Head : 1.58 oz/t Mass Balance : 117.1 %

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APPENDIX III

PREG ROBBING TEST MASS BALANCE

PREG ROBBING TEST SUMMARY

TEST NO. : PR-1

PURPOSE : Preg Robbing

FEED : 24 grams of Humus

GRIND : —— minutes; —— % solids

PRE-AER : —— hours; —— % solids

LEACH : 5 hours; % solids

REPULP : none

WASH : 5 x 200 mLs water - twice solids weight

REAGENTS : Grind - water
Pre-aer -

Leach - NaCN - 1 gpL
- Lime - pH 10.5

- Pb Nitrate -

CONSUMPTIONS (lb/t) NaCN: 1.02 Lime: 27.2 Pb Nitr.

MASS BALANCE

Product	Wt/Vol g/mL	Ag ppm/oz/t	Ag mg	Ag Dist'n
Grind Filtrate 1	930	43.4	40.36	90.76
Filtrate 2 Wash	1120	3.21	3.60	8.10
Titration Samples				
Residue	24.1	21.0	0.61	1.14
Total			44.47	100.00

Liquid feed = 39.90 mg Ag x 980/1000 = 39.10 mg Ag
Solid feed = 21.88 g/t Ag x 24.1 g = 0.53 mg Ag
Total feed = 39.63 mg Ag

% Mass Balance = 112

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APPENDIX C

Procedures and Schedule for Tailings Reclaim - Kilborn Limited

APPENDIX "C"
PROCEDURES AND SCHEDULE FOR TAILINGS RECLAIM

1.0 INTRODUCTION

Since early ore processing commenced in 1910 tailings grades and methods of disposal have varied. Photo interpretation[1] has indicated that early tailings disposal in the R.S.C. 91 claim was probably deposited over a meadow slightly higher in elevation than the former Percy Lake while the later stages of disposal in claim R.S.C. 84 utilized containment dykes with discharges directed into treed areas. Similarly variations in the physical and chemical properties of tails will vary according to age. Field sampling and laboratory testing of tails indicate historical variations in grind and grade. The variations in the tailings and containment basins will require flexibility in recovery operations to maintain efficient throughput. Trial and error solutions to recovery problems will be practiced.

The general arrangement of the tailings areas is shown on Drawing No. 100-30-001. Contingency planning for this early recovery period will include paralleled working areas to ensure that reclaimed tails are in constant supply to meet production schedules in the mill. For example, Area IA is considered to be a former meadow filled with tails. The worst case scenario would require a dragline working off mats to reclaim the tails with trucking to the slurry pumping and pipeline system located in Area ID. Temporary production shortfalls from Area IA would be augmented by supplies from Areas IC or ID where reclaiming is from better terrain.

The general plan for reclaiming tails is:

[1] Using Ministry of Natural Resources aerial photos from 1946 and 1970 photography.

.1 Dewatering the working areas with surface ditches and sump pumps driven by portable generators.

.2 Disposal of sump pump discharges to the tailings reclaim pump box as a portion of slurry make up water.

.3 A three cubic yard front end loader working over the dewatered tails deposit will load haul and dump tails in the Feeder Hopper as shown on Drawing No. 100-10-F04.

.4 The reclaimed tails will be slurried and pumped via a 6 inch diameter polyethelene pipeline to the mill for processing.

.5 Dragline truckhaul operations will be scheduled for day shift (weekdays), and the balance of mill feed will be supplied from front end loader working in near vicinity of slurry/pumping station.

.6 The slurry/pumping station is to be a skid mounted semi portable unit powered from a diesel generator set. Flexible polyethelene tailings line (6 inch diameter) will be used. Small moves of the portable slurry/pumping station and power plant will be carried out on day shift (weekdays) with major relocations being carried out as a seasonal operation during winter shutdown.

.7 Tailings recovery areas will be dewatered using sumbersible pumps powered by small portable generators and surface drainage ditches. As working areas are extended dewatering systems will be advanced. Relocation of dewatering system will be carried out on day shift (weekdays).

.8 Fueling and maintenance of tailings reclaim plant will be carried out during the day shift to minimize the afternoon and night shift workload. Extended duty fuel tanks will be supplied to slurry station generator and dewatering pumps.

2.0 PREPRODUCTION DEVELOPMENT AND FIELD INVESTIGATIONS

Plant startup is scheduled for Spring 1988. Area IA is required to be mined out early in year one (1) to provide pondage for settling and clarification prior to recycle. Area IB is to be a tailings disposal area for the first year of operation and therefore tails scheduled for reclaim must be reclaimed ahead of disposal operations in year one. Construction of a dyke between Area IB and Area ID is required at startup. Early aging of water may be accomplished in Area IB prior to infilling with reprocessed tails. Dykes must be in place at the upstream and downstream ends of Area IB.

During the late summer and early fall of 1987, contractual arrangements should be made for a dragline to dig drainage ditches in Area IA, IB and ID. The material should be cast in piles well back from the ditches and allowed to drain. This will provide a capability for early drainage of Area IB in the spring of 1988 and a first hand knowledge of the tails behavior to aid detailed planning of reclaim operations over the winter. Area IA should be pumped down in the late fall of 1987 to drain the soils prior to freeze-up.

The dykes at either end of Area IB should be constructed in the early fall of 1987 to take advantage of dry frost-free weather for earth work.

3.0 LONG RANGE PLAN FOR TAILS RECOVERY AND RESTORATION

By the completion of year one reclaim operations will have advanced into Area ID and perhaps as far as Area IE. The actual advance northward towards Area II will depend in large measure on how high a tails production rate from Area IA and IB may be maintained. In the event that these tails are overly wet for efficient handling and trafficability or exhibit high fluctuations in grade and contaminants, make up tonnages will be used from alternate sources. High quality tailings are known to exist in Area IC and at Area II where the last phase of tails deposition took place. Volumes extracted from Areas IE and II to meet production shortfalls and grade control requirements will reduce the advance northward into Phase II tailings. It should be noted that temporary tonnage makeup operations will add the additional cost of truck haul for transportation of tailings to the slurry pump box.

The best case scenario would have all of tails from Areas IC, ID, IE and II harvested directly from the face and transported to the slurry station positioned in close proximity to the working face.

The penalties to be paid for less idealized field conditions are:

.1 extended cycle times and costs for front end loader feeding slurry station or,

.2 trucking costs where the haul distance from the face to the slurry station exceeds the economic haul distance of the front end loader.

The less makeup material hauled in from Areas II and IE, the lower will be the production costs of tails from those areas.

It will be of great advantage to develop reliable production procedures for the more difficult working areas. The sooner constant reclaim methods are developed, the more efficient recovery operations will become for more centralized mining operations. Early dragline work in the fall of 1987 will provide senior operating personnel with first hand experience in tails handling and afford early behavioral characteristics of draglined tails to plan the Area IA mining strategy over the winter of 1987/88. This work will provide data on:

.1 Tails drainage rates in the draglined piles.

.2 Drainage rates from the ditch walls. Test pits should be dug at intervals back from the ditches to establish the in-pile drawdown curve of the in-situ tails. This will be required to determine criteria for dewatering ditch spacing for trafficable surface in reclaim areas. The cost benefit of such simple field tests is considered to far outweigh potential benefits of geotechnical engineering design work required for ditch design and spacing.

It should be recongnized that dragline work completed in 1987 will be required for the production dewatering program. The additional field testing required would be test pits and monitoring water levels in pits during drawdown.

.3 The tails/soils profiles along dewatering trenches.

.4 The tails grades and grinds in pile if sampling of ditch spoil is carried out.

.5 The potential existance of waste materials such as tree roots, solid waste dumps from past activities (pipe, barrels, garbage etc.) and organic soils layers underlying or within the pile areas.

The recovery of tails is to eventually proceed from the south to the north end of Area II. In order to maximize the early capital recovery the higher grade tails with easiest accessibility are to be processed first. In Area II this calls for proceeding up the middle and deepest portions of the tailings pile. It is anticipated that physical obstructions such as trees and organic soils will be minimal in the centre (deeper) sections of the pile which will be processed first. There are two advantages to this plan, first the higher profits will be produced during the early months of the project and operating experience will be gained prior to working in shallower areas where recovery is considered to be more difficult. By the completion of year two (2) the tailings in the former

Percy Lake should have been reprocessed and returned to this basin. The former Percy Lake will have been filled to capacity and future redeposition will be carried out in Area II. This will require a series of intermediate dykes as the redeposition in Area II follows the reclaim advance northward.

APPENDIX D

Environmental Overview
SENES Consultants Limited

ENVIRONMENTAL OVERVIEW
THE SANDY "K"
TAILINGS REMILLING PROJECT

by

SENES Consultants Limited
499 McNicoll Avenue
Willowdale, Ontario
M2H 2C9

February 1987

Project #30367

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30367

14 April 1987

Watts Griffis and McOuat Limited
8 King Street East
Suite 400
Toronto, Ontario
M5C 1B5

ATTENTION: Mr. Bill Fredenburg

Dear Sirs:

RE: Sandy 'K' - Environmental Overview

Enclosed please find 3 copies of our report on the Sandy 'K' tailings remilling project.

We met with the Ministry of the Environment in Timmins to discuss the project. The MOE were very pleased that WGM had come in to discuss the project and had no major concerns over the proposed plans. The MOE stated that they accept the natural degradation plans as long as it works. If the discharge objectives are not met then treatment will be required. Mr. Lalonde of the MOE also noted that WGM should stress this project will reduce the existing impact of tailings by consolidating several small deposits into one central area.

There are no major roadblocks that we can forsee at this time. However, there are potential concerns that could arise in the environmental permitting process. These are:

i) the existing dykes to the north and east are pervious and will leak cyanide contaminated water. This water will flow north away from the treatment system. The MOE may require monitoring and interception of this seepage.

ii) Miller Creek currently exceeds provincial water quality objectives for zinc. Gold mines normally have elevated copper levels. This may elevate the copper levels in Miller Creek to above the objective.

.../2

iii) there is a potential that arsenic removal may be required. This would be accomplished with ferric chloride addition to the recycle pond overflow.

Should WGM decide to proceed with this project several forms, reports and approvals will be required. The Ministry of the Environment will require:

- an environmental monitoring plan (water quality)
- a minerals industry information sheet
- a permit to take water from the recycle pond
- a Certificate of Approval to own and operate a sewage works (tailings area and effluent works)
- permit for septic system

The other agency approvals were previously outlined in our letter of 7 January 1987 to your Mr. Ehrlich.

Should all go well, there is very little involved in obtaining your permit to operate from the MOE. For budget purposes we suggest you allow $10,000 for environmental assistance and $10,000 for additional data collection.

The annual operating costs associated with environmental control should be limited to sample collection and analysis and reporting. A allowance for 100 environmental samples with meetings and annual reports would be $30,000 to $50,000 per year.

We trust this meets your immediate requirements and remain.

Yours very truly,

SENES CONSULTANTS LIMITED

R A Knapp

R.A. Knapp, P.Eng.
Principal

ls

Encl.

1.0 INTRODUCTION

The Sandy K mining property as shown on Figure 1 is located 2 km northeast of Gowganda. The silver mine operated on an intermittent basis from the 1940's through to 1970 producing some 1,000,000 tons of tailings. Because milling was restricted to gravity separation processes, these tailings contain substantial levels of residual silver. The quantity of silver is highly variable but typically ranges between 1 and 3 ounces/ton. Consideration is being given to remilling the tailings for silver recovery using a standard cyanidization process. Tailings would be excavated by mechanical methods and slurried to the mill. After processing, the tailings would be discharged back into the mined out areas of the tailings basin. Details of the recovery and remilling processes are addressed in the body of the report.

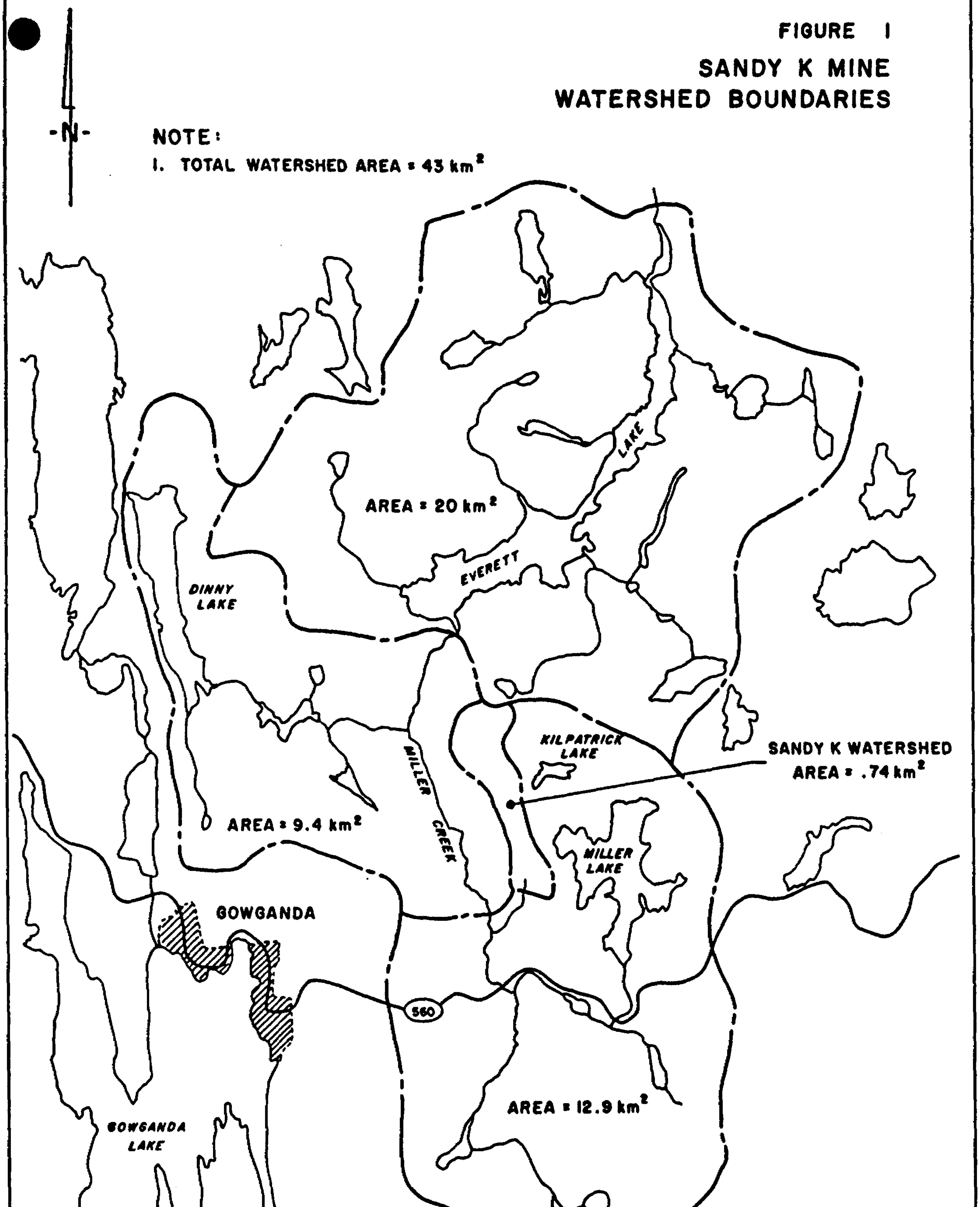
The waste management system for the project will include:

- a tailings disposal area for remilled tailings
- a recycle pond which will serve as the primary mill water supply
- a polishing pond to permit natural degradation of residual cyanide in the overflow from the recycle pond.

This report reviews the existing environmental conditions and provides an assessment of the waste management plans and their potential effects on the environment.

FIGURE 1

SANDY K MINE
WATERSHED BOUNDARIES



2.0 EXISTING CONDITIONS

2.1 Description of the Watershed

The Sandy "K" mine and waste management facilities are located in a small subwatershed which drains via Miller Creek to Everett Lake and the Montreal River as shown on Figure 1. The relative watershed sizes are as follows:

.74 km^2 - at the discharge from the polishing pond
12.9 km^2 - Miller Creek upstream of the mine
23.0 km^2 - Miller Creek at Everett Lake
43.0 km^2 - to Everett Lake Outlet

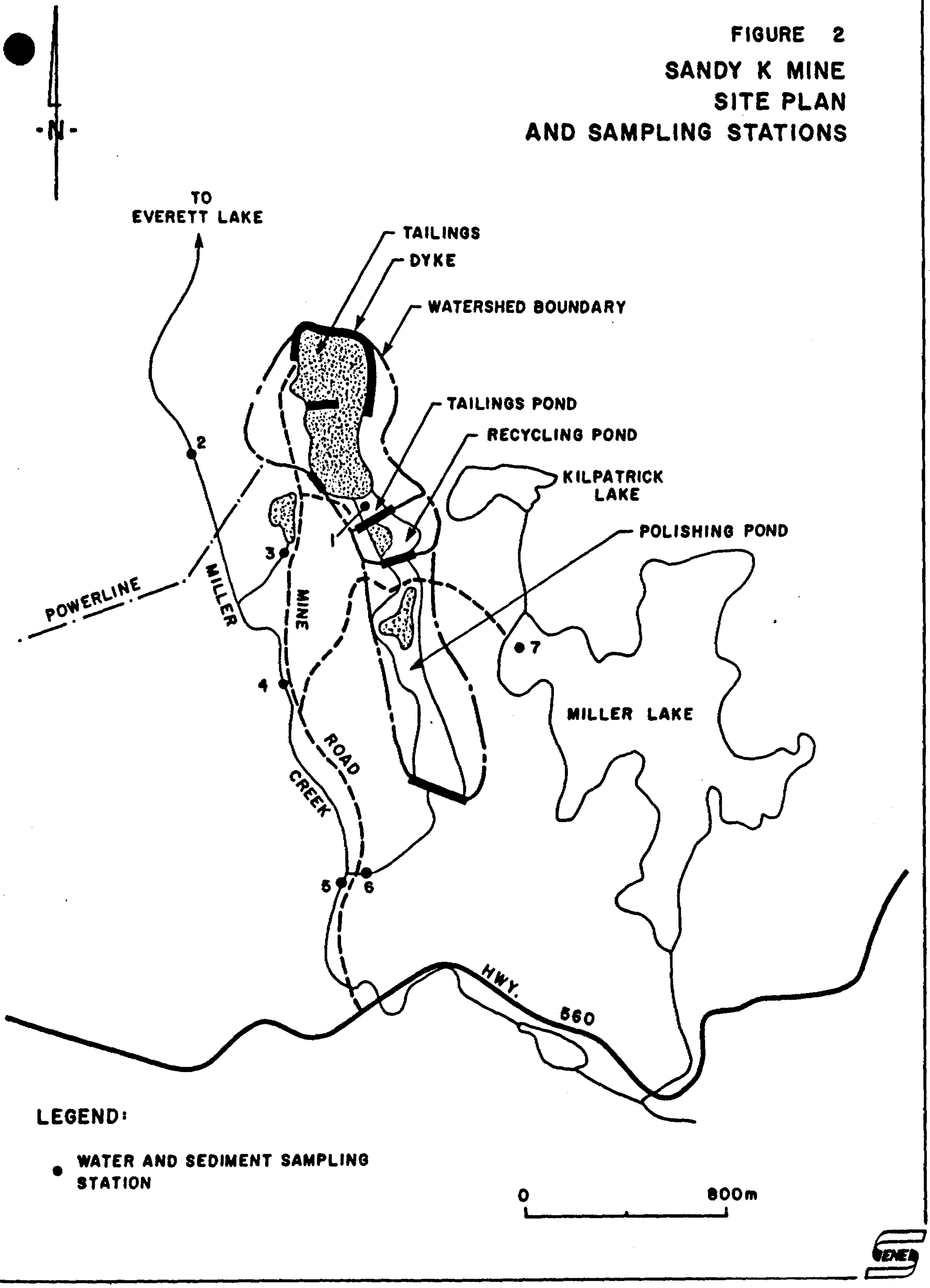
The topography is mildly undulating with a preponderance of lake basins and swampy terrain. Precipitation levels as measured at Kirkland Lake, Earlton, and Englehart indicate annual precipitation will be in the range of 850 mm/a. Of this precipitation approximately 50% will report as runoff and 50% will be evaporated. Typical unit runoff rates for this area are 15 $L/s/km^2$ of drainage basin.

2.2 Water Quality and Sediments

There were no site specific data available for water and sediment quality therefore, a baseline data collection program was initiated. The sampling locations for sediments and water are illustrated on Figure 2 and listed below.

Water Quality
W01 - Percy Lake - Tailings Pond
W02 - Miller Creek - Downstream Mine
W03 - HiGrade Tailings Runoff
W04 - Miller Creek - Middle of Mine Property
W05 - Miller Creek at Southern Property Line
W06 - Property Drainage at Southern Boundary

FIGURE 2
SANDY K MINE SITE PLAN AND SAMPLING STATIONS



W07 - Miller Lake

Two types of waters were sampled, those characteristic of baseline conditions in the Miller Creek system (W02, W04, W05, W06, W07) and those characteristic of tailings and mine site runoff (W01, W03). The laboratory results are presented in Appendix A1 and key constituents are listed in Table 2.1.

Site Runoff Quality (Stations W01, W03)

Site runoff has neutral pH, is highly alkaline and contains low levels of dissolved salts. Heavy metals levels (Cu, Pb, Ni) are all at less than <.05 mg/L while Zn is elevated at .07 and .14 mg/L. The neutral pH and high alkalinities are characteristics of the mineralogy of the tailings which contain elevated levels of calcite. Arsenic levels are marginally elevated and their presence is likely due to arsenopyrite oxidation. The high iron and manganese levels in the tailings pond sample W01 are indicative of reducing conditions in the bottom sediments during the ice covered winter months.

Receiving Waters Quality (Stations W02, W04, W05, W06, W07)

The baseline water quality in both the upstream and downstream waters of Miller Creek is similar. Waters are at neutral pH, well buffered and contain low metal concentrations. Zinc levels are elevated above provincial water quality objectives at all stations. Iron levels are marginally elevated which is not uncommon for these boggy northern watersheds. The receiving waters downstream of the mine do not appear to be impacted by runoff (at least during the winter sampling period) although arsenic levels are marginally elevated.

Sediment Quality

Detailed analyses of the sediments are presented in Appendix A2 and key constituents are summarized in Table 2.2.

All sediments with the exception of W05 have elevated levels of silver,

Table 2.1

WATER QUALITY SAMPLES
(mg/L unless noted)

	Site Runoff		Natural Waters					Drinking Water Objectives[1]	Mining Discharge Objectives
	W01	W03	W02	W04	W05	W06	W07		
pH	6.94	7.76	7.0	7.06	7.14	7.33	7.24	6.5-8.5	5.5-10.6
Alkalinity	162	200	31	29	26	52	--	NLS[2]	NLS
SO_4	1.75	26.8	7.64	7.63	7.49	8.61	8.07	500	NLS[3]
TDS	210	288	71	77	65	92	65	500	NLS[3]
Hg	<.05	<.05	<.05	<.05	<.05	<.05	<.05	.001	NOTE[5]
Cu	<.008	<.008	<.008	<.008	<.008	<.008	<.008	15	
Pb	<.05	<.05	<.05	<.05	<.05	<.05	<.05	.05	1[4]
Ni	<.05	<.05	<.05	<.05	<.05	<.05	<.05	NLS	
Zn	.07	.14	.06	.11	.13	.11	.03	5	
As	.6	.625	.03	.0002	.002	.027	.005	.05	0.5
Fe	9.12	.84	.36	.41	.23	.18	.05	.33	1
Mn	1.23	.43	.08	.08	.01	.04	.01	.05	1

Notes:

1 Maximum permissible concentration (MPC).

2 NLS - no level specified.

3 SO_4 and TDS levels should be as low as reasonably achievable.

4 Total level of Cu, Ni, Pb and Zn should not exceed 1 mg/L.

5 Mercury levels should not exceed background levels.

Table 2.2

SEDIMENT QUALITY

	Tailings Area		Miller Creek				Typical Values
	W01	W03	W02	W04	W05	W06	For Soil/Rock
Ag ppm	31.6	261	65.7	2	<.5	1.1	<.5
Al %	7.81	7.16	7.19	6.96	4.41	6.51	3-8
Ca %	3.45	5.86	3.95	2.10	1.45	2.38	2-4
Co ppm	326	402	218	35	13	18	.3-30
Cu ppm	426	426	99	56	86	29	.10-100
Fe %	7.87	7.05	5.89	6.30	1.64	3.58	1-5
K %	1.16	1.19	1.26	1.19	1.28	1.18	1-3
Mg %	5.74	4.59	3.71	3.57	.85	1.81	.5-2
Mn ppm	11609	1770	3100	1270	492	703	300-1000
Na %	1.49	1.72	1.84	2.50	2.09	2.30	.3-3
Ni ppm	225	169	98	175	34	65	2-100
Pb ppm	140	425	110	20	20	20	10-20
Ti ppm	1830	2680	2660	3630	1820	2960	2000-4000
V ppm	153	159	130	115	45	79	20-200
Zn ppm	170	235	149	163	102	183	1-100
L.O.I. %	7.7	3.2	6.4	5.6	8.5	5.6	--

aluminum, copper, iron, magnesium, nickel, vanadium and zinc. These levels are indicative of the presence of tailings discharged during previous operations. The highest values are measured in the tailings areas and Miller Creek downstream of the mine. The southern drainage stream (Station W06) and Miller Creek at the mine site (W04) only show minor increases in mineralized content over the background levels measured upstream of the mine (W05).

3.0 CHEMICAL AND MINERALOGICAL CHARACTERISTICS TAILINGS SOLIDS AND LIQUIDS

3.1 Mineralogical Characteristics Tailings Solids

The tailings gangue minerals include calcite, iron magnesium silicate minerals such as hornblende and chlorite, feldspar, quartz and sericite. The major trace minerals include native silver, arsenopyrite (with chalcopyrite and pyrite), iron oxide and titanium oxide.

The high carbonate content makes these tailings a net acid consumer as is evidenced by the high alkalinity levels in the tailings runoff.

3.2 Barren Solution and Tailings Discharge

Barren solutions produced during the metallurgical testwork were analyzed for contaminants including cyanide, cyanide complexes and metals. From the analysis of the barren solution as shown in Table 3.1 it is possible to estimate tailings water quality.

The metallurgical process will utilize 0.5 lb/ton of NaCN. Of this cyanide a portion will be retained with tailings, some will be converted to thiocyanate and cyanate and the remainder will be present as free and combined cyanides.

The Sandy "K" mill is to operate at 1000 tons/day for 200 days per year. Therefore the total cyanide usage will be 200 days x 1000 tons/day x .5 lb/ton = 100,000 lb NaCN. Therefore there will be 53,000 lb of CN^- used per year. Of this cyanide, it is estimated that 25% will be adsorbed onto sulphides and trapped in porewaters in the tailings leaving approximately 40,000 lb to report to the tailings pond. Based upon the barren analysis, as shown on Table 3.1, prior to any degradation by stripping, photooxidation, or other natural processes, the remaining 40,000 lb of CN^- will consist of 75 percent free cyanide and 25 percent combined metal cyanides.

Table 3.1

BARREN ANALYSIS
(mg/L)

	Experimental Data Before Zn Precipitation	Estimated Barren[1] Solution
Copper	9.96	9 (11)[3]
Nickel	2.64	2.5 (4.4)
Iron	4.5	4.5 (6.3)
Zinc	3.97	10.0 (15.9)
Total CN^-	473[1]	150[2]

Notes:

1 Analysis was performed on test solutions which contain excess cyanide and have not undergone silver precipitation. Silver recovery will produce reductions in Copper and Nickel and increase Zn levels.

2 Total cyanide levels are expected to be in the range of 150 mg/L for the operating mine.

3 Numbers in brackets are the equivalent level of cyanide associated with the metal.

4.0 WATER REQUIREMENTS/USE

There are two sources of water required for the milling operation, potable water and process water. All potable water will be trucked in from Gowganda and used for all fresh water requirements such as drinking water, showers and sanitary facilities. Sanitary waters would be disposed at an on site septic tank system.

Process water will come from the tailings reclaim pond. No other process water sources are required. A permit to take water from the reclaim pond will be required.

5.0 THE TAILING MANAGEMENT SYSTEM

5.1 Description

The tailings management area will include a tailings pond, reclaim pond and polishing pond. Tailings which are reclaimed from various locations around the site will be consolidated into the main tailings areas and reclaim pond as shown on Figure 2.

Treatment should only be required for cyanide removal and cyanide metal complexes. The proposed treatment system will be a natural degradation system which is ideally suited to this small watershed with large pondage.

5.2 Water Balance

The water balance for an average years operation is shown on Figure 3. For this system, the only discharge is the excess precipitation which lands within the watershed and reports as runoff. The average annual runoff from the 75 ha watershed is 355,000 m^3. With this annual runoff rate, the polishing pond has a theoretical residence time of 1 year.

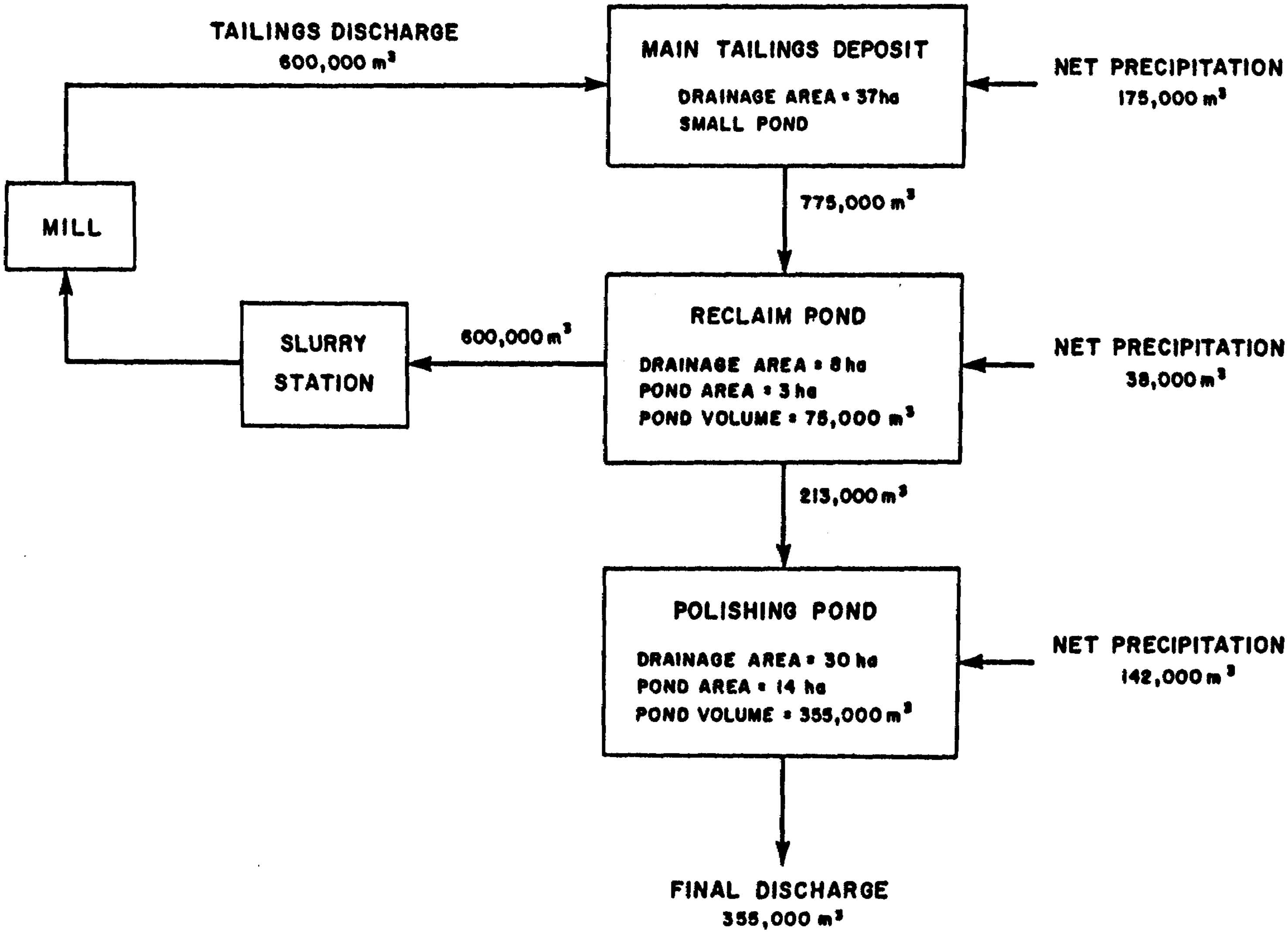
5.3 System Operation

The mine will operate for 8 months during the spring to fall period. No winter operations are proposed. During the initial year of operation, the polishing pond will be empty and given there is a years potential capacity for runoff, no discharge will be necessary. However, monitoring of pond water quality will be undertaken and the pond will be discharged when it meets discharge limits.

Under normal operating conditions it is expected that effluents can be discharged on a continuous basis. Because no cyanide is discharged to the pond during the winter months when the pond would be ineffective as a natural degradation system, it is expected that the very large polishing pond will

FIGURE 3

SANDY K MINE
TAILINGS AREA WATER BALANCE



meet discharge objectives. Should this not be the case, the pond has adequate storage potential to be used as a seasonal discharge system. With the large pondage available, levels in the pond should almost certainly meet discharge objectives between July and October. Therefore should water storage be required, it would only be necessary from startup in April to July. The polishing pond would under this scenario be drawn down during the summer and fall to permit the April to June runoff to be stored.

5.4 Effluent Quality

It is truly difficult to predict effluent quality from a natural degradation system. However, this system is ideally suited to natural degradation as no cyanide is discharged to the tailings pond during the winter and the system has pondage which provides for approximately 1 years holding time. Given these conditions one can estimate the effectiveness of a natural degradation system based upon industry experience.

The most extensive field investigations on the natural degradation of cyanide was conducted at Dome Mines. Dome stored barren bleed solutions during the winter months in holding ponds to reduce the cyanide loading to the tailings system. Monitoring of the cyanide indicates natural degradation in these ponds occurs as a first order reaction as follows:

$$C_t = Co\ e^{-kt} \qquad \text{(Eq.1)}$$

where

C_t = cyanide levels in the pond after time t
Co = cyanide level at t=0
k = first order cyanide decay constant

Schmidt et al., 1981 found that k ranged from .001 to .0052 hr^{-1} in the shallow holding ponds at Dome Mines. Using a median value of .003 hr^{-1}, equation 1 can be rewritten as a treatment efficiency:

$$\text{Treatment efficiency} = (1 - Ct/Co) \times 100$$

where

$$Ct/Co = e^{-.003 \times t} \qquad \text{(Eq.2)}$$

where

t = time in hours

The tailings and polishing pond will receive 40,000 lb (19,160 kg) of cyanide. Given the estimated annual flow of 355,000 m^3/a, the cyanide levels in the polishing pond would average 51 mg/L with no degradation. To reduce this level to 2 mg/L, $C_t/Co = .039$. From equation 1, the time required to reach 2 mg/L CN^- is 1081 hours or 1.5 months. During cooler months, the holding time period would be longer and during the warm months it would be shorter. Given that the polishing pond has a much longer residence time, cyanide levels of < 2 mg/L are expected.

6.0 ENVIRONMENTAL EFFECTS

The mining and milling of tailings will result in several minor impacts. The primary effects will include:

- dusting from tailings handling operations
- noise from excavation and milling
- disturbance of lands
- changes in local downstream water quality

Because of the remote location of the mining operations, concerns with noise and dusting are minor. The remilling of the tailings will actually result in the recovery of some disturbed land areas as tailings will be consolidated into one deposit. The major concerns relate to the release of contaminated waters which could degrade downstream water quality.

Effluent from the mining operations are discharged into Miller Creek near its headwaters. The first major receptor is Everett Lake. The predicted water quality of the mining effluent, Miller Creek upstream of Everett Lake and Everett Lake are provided in Table 6.1. Also shown are the mining effluent guidelines and the provincial water quality objectives.

Under the present conditions, Miller Creek has elevated baseline levels of zinc and arsenic which are due in part to local mineralization and discharges from the existing tailings basin. Zinc levels exceed the provincial objective both upstream and downstream of the mine.

With the effluent discharge from the Sandy "K" mine, only minor changes to water quality should be noticed in Miller Creek and Everett Lake. This is because there is a 30 fold dilution in Miller Creek and almost a 60 fold dilution in Everett Lake. Only copper is likely to exceed the provincial water quality objectives. However, if copper levels are reduced to below 0.5 mg/L (as would be expected if the natural degradation system is highly efficient) then water quality objectives for copper may be met in Miller Creek.

Table 6.1

	Predicted Effluent Quality	Mining Discharge Objectives	Miller Creek[1] Baseline	Everett Lake With Effluent Discharge	Everett Lake Outlet	Surface Water Objectives
pH (units)	7-10	5.5-10.6	7	7	7	5.5-8.5
Avg. Flow L/S	11.3	---	345	345	645	---
Total CN^- mg/L	1.5	2	0	.049	.026	---
Free CN^- (mg/L)	.1	2	0	.003	.002	.005
Cu (mg/L)	.5		<.01	.016	.009	.005
Pb (mg/L)	.1	1.0	<.05	.003	.002	.010[2]
Ni (mg/L)	.1		<.05	.003	.002	.025
Zn (mg/L)	.1		.06	.06	.03	.03
As (mg/L)	.5	.5	.03	.03	.009	.10

Note:

1 At Station W02 below the mine.

2 At alkalinity level of 30 mg/L.

.0 REFERENCES

Ministry of the Environment. "Guidelines for Environmental Control in the Ontario Mineral Industry."

Ministry of the Environment. "Water Management, Goals, Policies, Objectives and Implementation Procedures."

Schmidt, J.W., Simovic, L., Shannon, E., 1981. "Natural Degradation of Cyanides in Gold Milling Effluents." Presented at the Gold Mining Industry Seminar, January.

APPENDIX A1

Water Quality Data

BARRINGER MAGENTA

304 CARLINGVIEW DRIVE
REXDALE, ONTARIO
M9W 5G2

(416) 675-3870

FILE: T7_4058W
DATE: 27/02/87
MATRIX: WATER.

WATTS,GRIFFIS & MCOUAT...WATERS...(BILL W. FREDENBURG)

WO NO: 87-4058W

SAMPLE ID	AG MG/L	AL MG/L	B MG/L	BA MG/L	BE MG/L	CA MG/L	CD MG/L	CO MG/L	CR MG/L	CU MG/L	FE MG/L	K MG/L
SDK W01	<.005	<.01	.013	.136	<.0005	54.6	<.01	<.05	<.01	<.008	9.12	2
SDK W02	<.005	.04	.005	.114	.0006	12.5	<.01	<.05	<.01	<.008	.36	<1
SDK W03	<.005	<.01	.049	.105	.0006	67.6	<.01	<.05	<.01	<.008	.84	5
SDK W04	.005	.04	.005	.142	.0005	11.7	<.01	<.05	<.01	<.008	.41	<1
SDK W05	.006	.04	<.004	.163	.0006	11.1	<.01	<.05	<.01	<.008	.23	<1
SDK W06	<.005	.06	.011	.118	.0008	18.2	<.01	<.05	<.01	<.008	.18	<1
SDK W07	<.005	<.01	.006	.067	.0008	13.2	<.01	<.05	<.01	<.008	.05	<1

SAMPLE ID	MG MG/L	MN MG/L	MO MG/L	NA MG/L	NI MG/L	P MG/L	PB MG/L	SI MG/L	SR MG/L	TH MG/L	TI MG/L	V MG/L
SDK W01	10.2	1.23	<.2	4	<.05	<.5	<.05	4.28	.065	<.05	<.005	.014
SDK W02	2.69	.08	<.2	2	<.05	<.5	<.05	3.15	.024	<.05	<.005	.005
SDK W03	12.1	.43	<.2	16	.05	<.5	<.05	7.37	.065	<.05	<.005	.010
SDK W04	2.59	.06	<.2	2	<.05	<.5	<.05	3.08	.024	<.05	<.005	.012
SDK W05	2.41	.01	<.2	2	<.05	<.5	<.05	2.83	.024	<.05	<.005	<.005
SDK W06	4.75	.04	<.2	2	<.05	<.5	<.05	5.40	.027	<.05	<.005	<.005
SDK W07	2.36	.01	<.2	2	<.05	<.5	<.05	.76	.026	<.05	<.005	<.005

SAMPLE ID	ZN MG/L	ZR MG/L	ALK PPMCACO3	AS UG/L	HG UG/L	PH	TDS MG/L	TOC MG/L	TSS MG/L	TURB. NTU
SDK W01	.07	<.05	162	600	<.05	6.94	210	--	23.0	--
SDK W02	.06	<.05	31	30	<.05	7.00	71	--	<.5	--
SDK W03	.14	<.05	200	625	<.05	7.76	288	--	36.0	--
SDK W04	.11	<.05	29	2	<.05	7.06	77	--	<.5	--
SDK W05	.13	<.05	26	2	<.05	7.14	65	--	<.5	--
SDK W06	.11	<.05	52	27	<.05	7.33	92	--	72.6	--
SDK W07	.03	<.05	--	5	<.05	7.24	65	7.0	--	.6

BARRINGER MAGENTA

304 CARLINGVIEW DRIVE
REXDALE, ONTARIO
M9W 5G2

(416) 675-3870

FILE: T7_4058W
DATE: 27/02/87
MATRIX: WATER.

WATTS,GRIFFIS & MCOUAT...WATERS...(BILL W. FREDENBURG) WO NO: 87-4058W

SAMPLE ID	F- MG/L	CL- MG/L	NO2- MG/L	BR- MG/L	NO3- MG/L	PO4-3 MG/L	SO4= MG/L
SDK W01	.399	2.17	<.02	<.05	.17	<.1	1.75
SDK W02	.043	3.41	<.02	<.05	.87	<.1	7.64
SDK W03	.072	3.31	<.02	<.05	.17	<.1	26.8
SDK W04	.076	3.52	<.02	<.05	.89	<.1	7.63
SDK W05	.078	3.62	<.02	<.05	.84	<.1	7.49
SDK W06	.066	.56	<.02	<.05	1.44	<.1	8.61
SDK W07	.066	2.25	<.02	<.05	.32	<.1	8.07

APPENDIX A2

Sediment Quality Data

BARRINGER MAGENTA
304 CARLINGVIEW DRIVE
REXDALE, ONTARIO
M9W 5G2
(416) 675-3870

FILE: T7_4058S
DATE: 19/02/87
MATRIX: HF

WATTS, GRIFFIS & MC'OUAT...SEDIMENTS...(BILL W. FREDENBURG)

WO NO: 87-4058

SAMPLE ID	AG PPM	AL PPM	BE PPM	CA PPM	CD PPM	CO PPM	CR PPM	CU PPM	FE PPM	K PPM	MG PPM
SDK W01 S	31.6	78100	.72	34500	<1	326	286	426	78700	11600	57400
SDK W02 S	63.7	71900	.50	39500	<1	218	223	99	58900	12600	37100
SDK W03 S	261	71600	.42	58600	<1	402	269	426	70500	11900	45900
SDK W04 S	2.0	69600	.69	21000	<1	35	339	56	63000	11900	35700
SDK W06 S	1.1	65100	.68	23800	<1	18	160	29	35800	11800	18100

SAMPLE ID	MN PPM	MO PPM	NA PPM	NI PPM	P PPM	PB PPM	SR PPM	TH PPM	TI PPM	V PPM	ZN PPM
SDK W01 S	1160	<20	14900	225	840	140	75	<1	1830	153	170
SDK W02 S	3100	<20	18400	98	510	110	146	<1	2660	130	149
SDK W03 S	1770	<20	17200	169	480	425	93	<1	2680	159	235
SDK W04 S	1270	<20	25000	175	800	20	222	<1	3630	115	163
SDK W06 S	703	<20	23000	65	620	20	264	<1	2960	79	183

SAMPLE ID	ZR PPM	+5.6MM %	+2MM %	+425UM %	+180UM %	+125UM %	+75UM %	-75UM %	L.O.I. %	MOISTURE %
SDK W01 S	29	<.01	<.01	.05	7.23	2.79	5.22	84.9	7.70	77.8
SDK W02 S	48	<.01	.82	13.5	41.9	14.6	13.0	14.1	6.40	38.1
SDK W03 S	44	<.01	.30	1.64	16.5	16.8	27.2	37.5	3.20	47.5
SDK W04 S	76	23.4	39.3	26.1	7.93	15.1	1.11	.59	3.60	14.4
SDK W06 S	59	19.1	21.5	17.0	24.0	7.77	5.09	5.62	5.60	22.2

BARRINGER MAGENTA
304 CARLINGVIEW DRIVE
REXDALE, ONTARIO
M9W 5G2
(416) 675-3870

FILE: T7_4066
DATE: 27/02/87
MATRIX: HF

WATTS, GRIFFIS & MC'OUAT...SEDIMENT...(BILL W. FREDENBURG) WO NO: 87-4066

SAMPLE ID	AG PPM	AL PPM	BE PPM	CA PPM	CD PPM	CO PPM	CR PPM	CU PPM	FE PPM	K PPM	MG PPM
SDK W05 S	<.5	44100	.87	14500	<1	13	86	16	16400	12800	8540

SAMPLE ID	MN PPM	MO PPM	NA PPM	NI PPM	P PPM	PB PPM	SR PPM	TH PPM	TI PPM	V PPM	ZN PPM
SDK W05 S	492	<20	20900	34	500	20	245	<1	1820	45	102

SAMPLE ID	ZR PPM	+5.6MM %	+2MM %	+425UM %	+180UM %	+125UM %	+75UM %	-75UM %	L.O.I. %	MOISTURE %
SDK W05 S	56	33.4	7.81	9.59	18.8	6.97	9.93	13.5	8.50	46.8

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 6.00/oz
$4,200,000 Capital Costs

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
REVENUE									
Ton of Ore Milled	-	225,000	225,000	225,000	225,000	225,000	225,000	225,000	1,575,000
Grade (oz/ton)	-	2.0	1.8	1.8	1.5	1.5	1.0	1.0	1.5
Recovery (%)	-	87.0	87.0	87.0	87.0	87.0	87.0	87.0	87.0
Production (ozs)	-	391,500	342,563	342,563	293,625	293,625	195,750	195,750	2,055,375
Revenue	-	3,132,001	2,740,501	2,740,501	2,349,001	2,349,001	1,566,000	1,566,000	16,443,002
OPERATING COSTS									
Mining & Milling	-	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	9,100,000
Refining Charges	-	116,589	102,015	102,015	87,442	87,442	58,294	58,294	612,091
	-	1,416,589	1,402,015	1,402,015	1,387,442	1,387,442	1,358,294	1,358,294	9,712,092
NET OPERATING PROFIT	-	1,715,412	1,338,486	1,338,486	961,559	961,559	207,706	207,706	6,730,914

CASH FLOW TO CLE

	1987	1988	1989	1990	1991	1992	1993	1994	Total
Net Operating Profit	-	1,715,412	1,338,486	1,338,486	961,559	961,559	207,706	207,706	6,730,914
Less: Capital Investment	2,800,000	1,400,000	-	-	-	-	-	-	4,200,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
NPI to Siscoe	-	-	-	-	-	-	-	-	-
Royalty to Sandy K	-	-	-	-	316,191	359,238	83,082	403,082	1,161,593
Federal Income Tax	-	-	-	38,957	130,687	121,970	25,236	100,000	416,851
Provincial Income Tax	-	-	-	18,597	62,386	58,224	12,047	50,000	201,254
Provincial Mining Duty	-	-	-	-	63,465	63,465	-	-	126,930
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Net Cash Flow to CLE	(2,800,000)	15,412	1,338,486	1,280,931	388,831	358,662	87,340	454,624	1,124,285
Accumulated Net Cash Flow	(2,800,000)	(2,784,589)	(1,446,103)	(165,172)	223,659	582,321	669,661	1,124,285	1,124,285
Rate of Return to CLE	-	-	-	-	2.9	6.8	7.5	10.6	10.6
Present Value of NCF at 10%	(2,800,000)	(2,785,990)	(1,679,803)	(717,421)	(451,844)	(229,143)	(179,842)	53,452	53,452
Present Value of NCF at 15%	(2,800,000)	(2,786,599)	(1,774,512)	(932,279)	(709,964)	(531,646)	(493,886)	(322,976)	(322,976)
Present Value of NCF at 20%	(2,800,000)	(2,787,157)	(1,857,653)	(1,116,374)	(928,860)	(784,721)	(755,471)	(628,594)	(628,594)

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 6.00/oz
$4,200,000 Capital Costs

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
SISCOE ROYALTY CALCULATION									
Opening Unrecovered Costs	2,000,000	4,800,001	5,597,091	5,071,106	4,545,121	4,459,528	4,373,935	4,978,730	-
Plus: New Capital Investment	2,800,000	1,700,000	-	-	-	-	-	-	4,500,001
Operating Costs	-	1,416,589	1,402,015	1,402,015	1,387,442	1,387,442	1,358,294	1,358,294	9,712,092
Depreciation	-	812,500	812,500	812,500	812,500	812,500	812,500	812,500	5,687,501
Prov. Mining Duties	-	-	-	-	63,465	63,465	-	-	126,930
Less: Total Revenue	-	3,132,001	2,740,501	2,740,501	2,349,001	2,349,001	1,566,000	1,566,000	16,443,002
Closing Unrecovered Costs	4,800,001	5,597,091	5,071,106	4,545,121	4,459,528	4,373,935	4,978,730	5,583,525	-
Siscoe NPI	-	-	-	-	-	-	-	-	-
CALCULATION OF CASH DISTRIBUTION									
Operating Profit	-	1,715,412	1,338,486	1,338,486	961,559	961,559	207,706	207,706	6,730,914
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Less: NPI to Siscoe	-	-	-	-	-	-	-	-	-
Prov. Mining Duties	-	-	-	-	63,465	63,465	-	-	126,930
Net Cash for Distribution	-	1,715,412	1,338,486	1,338,486	898,094	898,094	207,706	1,007,706	7,403,984
CLE's Recovery of Capital	-	1,715,412	1,338,486	1,338,486	107,618	-	-	-	4,500,002
CLE's Premium Account	-	-	-	-	158,095	179,619	41,541	201,541	580,797
CLE's Payout	-	-	-	-	316,191	359,238	83,082	403,082	1,161,593
Total Payout to CLE	-	1,715,412	1,338,486	1,338,486	581,904	538,856	124,624	604,624	6,242,392
Payout to Sandy K	-	-	-	-	316,191	359,238	83,082	403,082	1,161,593
CLE'S NET CASH FLOW									
Total Payout to CLE	-	1,715,412	1,338,486	1,338,486	581,904	538,856	124,624	604,624	6,242,392
Less: Capital Investment	2,800,000	1,400,000	-	-	-	-	-	-	4,200,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
Federal Income Tax	-	-	-	38,957	130,687	121,970	25,236	100,000	416,851
Prov. Income Tax	-	-	-	18,597	62,386	58,224	12,047	50,000	201,254
Net Cash Flow to CLE	(2,800,000)	15,412	1,338,486	1,280,931	388,831	358,662	87,340	454,624	1,124,286

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 8.00/oz
$4,200,000 Capital Costs

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
REVENUE									
Ton of Ore Milled	-	225,000	225,000	225,000	225,000	225,000	225,000	225,000	1,575,000
Grade (oz/ton)	-	2.0	1.8	1.8	1.5	1.5	1.0	1.0	1.5
Recovery (%)	-	87.0	87.0	87.0	87.0	87.0	87.0	87.0	87.0
Production (ozs)	-	391,500	342,563	342,563	293,625	293,625	195,750	195,750	2,055,375
Revenue	-	4,176,001	3,654,001	3,654,001	3,132,001	3,132,001	2,088,000	2,088,000	21,924,004
OPERATING COSTS									
Mining & Milling	-	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	9,100,000
Refining Charges	-	127,029	111,150	111,150	95,272	95,272	63,514	63,514	666,901
	-	1,427,029	1,411,150	1,411,150	1,395,272	1,395,272	1,363,514	1,363,514	9,766,902
NET OPERATING PROFIT	-	2,748,972	2,242,851	2,242,851	1,736,729	1,736,729	724,486	724,486	12,157,106

CASH FLOW TO CLE

	1987	1988	1989	1990	1991	1992	1993	1994	Total
Net Operating Profit	-	2,748,972	2,242,851	2,242,851	1,736,729	1,736,729	724,486	724,486	12,157,106
Less: Capital Investment	2,800,000	1,400,000	-	-	-	-	-	-	4,200,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
NPI to Siscoe	-	-	-	-	-	-	-	-	-
Royalty to Sandy K	-	-	182,885	784,626	616,594	616,594	280,529	695,969	3,177,198
Federal Income Tax	-	-	123,310	295,290	226,827	226,827	89,901	105,775	1,067,931
Provincial Income Tax	-	-	58,864	140,962	108,280	108,280	42,916	52,757	512,058
Provincial Mining Duty	-	-	34,610	281,285	195,244	195,244	23,163	23,163	752,708
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Net Cash Flow to CLE	(2,800,000)	1,048,972	1,843,182	740,687	589,784	589,784	287,977	646,823	2,947,210
Accumulated Net Cash Flow	(2,800,000)	(1,751,028)	92,154	832,841	1,422,626	2,012,410	2,300,387	2,947,210	2,947,210
Rate of Return to CLE	-	-	2.0	14.8	21.4	25.7	27.1	29.3	29.3
Present Value of NCF at 10%	(2,800,000)	(1,846,389)	(323,098)	233,392	636,222	1,002,432	1,164,988	1,496,910	1,496,910
Present Value of NCF at 15%	(2,800,000)	(1,887,850)	(494,141)	(7,127)	330,084	623,311	747,811	990,976	990,976
Present Value of NCF at 20%	(2,800,000)	(1,925,857)	(645,869)	(217,231)	67,195	304,215	400,658	581,175	581,175

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 8.00/oz
$4,200,000 Capital Costs

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
SISCOE ROYALTY CALCULATION									
Opening Unrecovered Costs	2,000,000	4,800,001	4,563,531	3,167,790	2,018,725	1,289,740	560,755	671,931	-
Plus: New Capital Investment	2,800,000	1,700,000	-	-	-	-	-	-	4,500,001
Operating Costs	-	1,427,029	1,411,150	1,411,150	1,395,272	1,395,272	1,363,514	1,363,514	9,766,902
Depreciation	-	812,500	812,500	812,500	812,500	812,500	812,500	812,500	5,687,501
Prov. Mining Duties	-	-	34,610	281,285	195,244	195,244	23,163	23,163	752,708
Less: Total Revenue	-	4,176,001	3,654,001	3,654,001	3,132,001	3,132,001	2,088,000	2,088,000	21,924,004
Closing Unrecovered Costs	4,800,001	4,563,531	3,167,790	2,018,725	1,289,740	560,755	671,931	783,108	-
Siscoe NPI	-	-	-	-	-	-	-	-	-
CALCULATION OF CASH DISTRIBUTION									
Operating Profit	-	2,748,972	2,242,851	2,242,851	1,736,729	1,736,729	724,486	724,486	12,157,106
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Less: NPI to Siscoe	-	-	-	-	-	-	-	-	-
Prov. Mining Duties	-	-	34,610	281,285	195,244	195,244	23,163	23,163	752,708
Net Cash for Distribution	-	2,748,972	2,208,241	1,961,566	1,541,485	1,541,485	701,323	1,501,324	12,204,398
CLE's Recovery of Capital	-	2,748,972	1,751,028	-	-	-	-	-	4,500,001
CLE's Premium Account	-	-	91,443	392,313	308,297	308,297	140,265	109,386	1,350,000
CLE's Payout	-	-	182,885	784,626	616,594	616,594	280,529	695,969	3,177,198
Total Payout to CLE	-	2,748,972	2,025,356	1,176,940	924,891	924,891	420,794	805,355	9,027,198
Payout to Sandy K	-	-	182,885	784,626	616,594	616,594	280,529	695,969	3,177,198
CLE'S NET CASH FLOW									
Total Payout to CLE	-	2,748,972	2,025,356	1,176,940	924,891	924,891	420,794	805,355	9,027,198
Less: Capital Investment	2,800,000	1,400,000	-	-	-	-	-	-	4,200,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
Federal Income Tax	-	-	123,310	295,290	226,827	226,827	89,901	105,775	1,067,931
Prov. Income Tax	-	-	58,864	140,962	108,280	108,280	42,916	52,757	512,058
Net Cash Flow to CLE	(2,800,000)	1,048,972	1,843,182	740,687	589,784	589,784	287,977	646,823	2,947,210

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 10.00/oz
$4,200,000 Capital Costs

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
REVENUE									
Ton of Ore Milled	-	225,000	225,000	225,000	225,000	225,000	225,000	225,000	1,575,000
Grade (oz/ton)	-	2.0	1.8	1.8	1.5	1.5	1.0	1.0	1.5
Recovery (%)	-	87.0	87.0	87.0	87.0	87.0	87.0	87.0	87.0
Production (ozs)	-	391,500	342,563	342,563	293,625	293,625	195,750	195,750	2,055,375
Revenue	-	5,220,002	4,567,502	4,567,502	3,915,001	3,915,001	2,610,001	2,610,001	27,405,004
OPERATING COSTS									
Mining & Milling	-	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	9,100,000
Refining Charges	-	137,469	120,285	120,285	103,102	103,102	68,734	68,734	721,711
	-	1,437,469	1,420,285	1,420,285	1,403,102	1,403,102	1,368,734	1,368,734	9,821,712
NET OPERATING PROFIT	-	3,782,533	3,147,216	3,147,216	2,511,899	2,511,899	1,241,266	1,241,266	17,583,300

CASH FLOW TO CLE

	1987	1988	1989	1990	1991	1992	1993	1994	Total
Net Operating Profit	-	3,782,533	3,147,216	3,147,216	2,511,899	2,511,899	1,241,266	1,241,266	17,583,300
Less: Capital Investment	2,800,000	1,400,000	-	-	-	-	-	-	4,200,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
NPI to Siscoe	-	-	-	68,076	342,551	436,975	226,050	226,050	1,299,702
Royalty to Sandy K	-	-	826,277	1,057,646	736,930	854,164	452,100	852,101	4,779,219
Federal Income Tax	-	57,213	328,240	409,353	290,065	247,204	114,031	133,031	1,579,137
Provincial Income Tax	-	-	184,002	195,411	138,467	118,007	54,435	65,768	756,090
Provincial Mining Duty	-	-	364,057	435,027	327,023	327,023	111,015	111,015	1,675,160
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Net Cash Flow to CLE	(2,800,000)	2,025,320	1,444,640	981,704	676,863	528,527	283,635	653,302	3,793,990
Accumulated Net Cash Flow	(2,800,000)	(774,681)	669,959	1,651,663	2,328,527	2,857,053	3,140,688	3,793,990	3,793,990
Rate of Return to CLE	-	-	16.6	31.7	37.6	40.4	41.4	42.9	42.9
Present Value of NCF at 10%	(2,800,000)	(958,800)	235,118	972,687	1,434,994	1,763,167	1,923,272	2,258,519	2,258,519
Present Value of NCF at 15%	(2,800,000)	(1,038,852)	53,502	698,989	1,085,987	1,348,758	1,471,382	1,716,982	1,716,982
Present Value of NCF at 20%	(2,800,000)	(1,112,234)	(109,012)	459,104	785,523	997,926	1,092,915	1,275,239	1,275,239

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 10.00/oz
$4,200,000 Capital Costs

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
SISCOE ROYALTY CALCULATION									
Opening Unrecovered Costs	2,000,000	4,800,001	3,529,969	1,559,311	(340,378)	(1,712,754)	(3,085,130)	(3,402,881)	-
Plus: New Capital Investment	2,800,000	1,700,000	-	-	-	-	-	-	4,500,001
Operating Costs	-	1,437,469	1,420,285	1,420,285	1,403,102	1,403,102	1,368,734	1,368,734	9,821,712
Depreciation	-	812,500	812,500	812,500	812,500	812,500	812,500	812,500	5,687,501
Prov. Mining Duties	-	-	364,057	435,027	327,023	327,023	111,015	111,015	1,675,160
Less: Total Revenue	-	5,220,002	4,567,502	4,567,502	3,915,001	3,915,001	2,610,001	2,610,001	27,405,004
Closing Unrecovered Costs	4,800,001	3,529,969	1,559,311	(340,378)	(1,712,754)	(3,085,130)	(3,402,881)	(3,720,632)	-
Siscoe NPI	-	-	-	68,076	342,551	436,975	226,050	226,050	1,299,702
CALCULATION OF CASH DISTRIBUTION									
Operating Profit	-	3,782,533	3,147,216	3,147,216	2,511,899	2,511,899	1,241,266	1,241,266	17,583,300
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Less: NPI to Siscoe	-	-	-	68,076	342,551	436,975	226,050	226,050	1,299,702
Prov. Mining Duties	-	-	364,057	435,027	327,023	327,023	111,015	111,015	1,675,160
Net Cash for Distribution	-	3,782,533	2,783,159	2,644,114	1,842,326	1,747,901	904,201	1,704,201	15,408,436
CLE's Recovery of Capital	-	3,782,533	717,468	-	-	-	-	-	4,500,001
CLE's Premium Account	-	-	413,138	528,823	368,465	39,574	-	-	1,350,000
CLE's Payout	-	-	826,277	1,057,646	736,930	854,164	452,100	852,101	4,779,219
Total Payout to CLE	-	3,782,533	1,956,883	1,586,469	1,105,396	893,738	452,100	852,101	10,629,218
Payout to Sandy K	-	-	826,277	1,057,646	736,930	854,164	452,100	852,101	4,779,219
CLE'S NET CASH FLOW									
Total Payout to CLE	-	3,782,533	1,956,883	1,586,469	1,105,396	893,738	452,100	852,101	10,629,218
Less: Capital Investment	2,800,000	1,400,000	-	-	-	-	-	-	4,200,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
Federal Income Tax	-	57,213	328,240	409,353	290,065	247,204	114,031	133,031	1,579,137
Prov. Income Tax	-	-	184,002	195,411	138,467	118,007	54,435	65,768	756,090
Net Cash Flow to CLE	(2,800,000)	2,025,320	1,444,640	981,705	676,864	528,527	283,635	653,302	3,793,992

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CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 12.00/oz
$4,200,000 Capital Costs

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
REVENUE									
Ton of Ore Milled	-	225,000	225,000	225,000	225,000	225,000	225,000	225,000	1,575,000
Grade (oz/ton)	-	2.0	1.8	1.8	1.5	1.5	1.0	1.0	1.5
Recovery (%)	-	87.0	87.0	87.0	87.0	87.0	87.0	87.0	87.0
Production (ozs)	-	391,500	342,563	342,563	293,625	293,625	195,750	195,750	2,055,375
Revenue	-	6,264,002	5,481,002	5,481,002	4,698,002	4,698,002	3,132,001	3,132,001	32,886,004
OPERATING COSTS									
Mining & Milling	-	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	9,100,000
Refining Charges	-	147,909	129,420	129,420	110,932	110,932	73,954	73,954	776,521
	-	1,447,909	1,429,420	1,429,420	1,410,932	1,410,932	1,373,954	1,373,954	9,876,522
NET OPERATING PROFIT	-	4,816,093	4,051,581	4,051,581	3,287,070	3,287,070	1,758,046	1,758,046	23,009,490

CASH FLOW TO CLE

	1987	1988	1989	1990	1991	1992	1993	1994	Total
Net Operating Profit	-	4,816,093	4,051,581	4,051,581	3,287,070	3,287,070	1,758,046	1,758,046	23,009,490
Less: Capital Investment	2,800,000	1,400,000	-	-	-	-	-	-	4,200,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
NPI to Siscoe	-	-	29,834	559,896	565,654	565,654	311,836	311,836	2,344,709
Royalty to Sandy K	-	124,542	1,373,192	1,161,167	1,121,032	1,131,307	623,671	1,023,671	6,558,584
Federal Income Tax	-	241,289	394,583	471,930	324,078	402,622	222,086	214,086	2,270,673
Provincial Income Tax	-	47,517	256,027	225,284	154,704	153,711	79,512	90,845	1,007,599
Provincial Mining Duty	-	4,736	588,769	588,769	458,802	458,802	198,868	198,868	2,497,613
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Net Cash Flow to CLE	(2,800,000)	2,698,009	1,409,177	1,044,536	662,800	574,974	322,074	718,740	4,630,312
Accumulated Net Cash Flow	(2,800,000)	(101,992)	1,307,186	2,351,722	3,014,522	3,589,497	3,911,571	4,630,312	4,630,312
Rate of Return to CLE	-	-	33.9	47.6	52.2	54.6	55.3	56.4	56.4
Present Value of NCF at 10%	(2,800,000)	(347,264)	817,346	1,602,121	2,054,823	2,411,837	2,593,639	2,962,466	2,962,466
Present Value of NCF at 15%	(2,800,000)	(453,905)	611,635	1,298,434	1,677,392	1,963,256	2,102,497	2,372,698	2,372,698
Present Value of NCF at 20%	(2,800,000)	(551,659)	426,936	1,031,413	1,351,050	1,582,119	1,689,981	1,890,568	1,890,568

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 12.00/oz
$4,200,000 Capital Costs

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
SISCOE ROYALTY CALCULATION									
Opening Unrecovered Costs	2,000,000	4,800,001	2,501,143	(149,169)	(2,799,481)	(4,815,250)	(6,831,018)	(7,577,697)	-
Plus: New Capital Investment	2,800,000	1,700,000	-	-	-	-	-	-	4,500,001
Operating Costs	-	1,447,909	1,429,420	1,429,420	1,410,932	1,410,932	1,373,954	1,373,954	9,876,522
Depreciation	-	812,500	812,500	812,500	812,500	812,500	812,500	812,500	5,687,501
Prov. Mining Duties	-	4,736	588,769	588,769	458,802	458,802	198,868	198,868	2,497,613
Less: Total Revenue	-	6,264,002	5,481,002	5,481,002	4,698,002	4,698,002	3,132,001	3,132,001	32,886,004
Closing Unrecovered Costs	4,800,001	2,501,143	(149,169)	(2,799,481)	(4,815,250)	(6,831,018)	(7,577,697)	(8,324,376)	-
Siscoe NPI	-	-	29,834	559,896	565,654	565,654	311,836	311,836	2,344,709
CALCULATION OF CASH DISTRIBUTION									
Operating Profit	-	4,816,093	4,051,581	4,051,581	3,287,070	3,287,070	1,758,046	1,758,046	23,009,490
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Less: NPI to Siscoe	-	-	29,834	559,896	565,654	565,654	311,836	311,836	2,344,709
Prov. Mining Duties	-	4,736	588,769	588,769	458,802	458,802	198,868	198,868	2,497,613
Net Cash for Distribution	-	4,811,357	3,432,979	2,902,917	2,262,615	2,262,615	1,247,343	2,047,343	18,967,168
CLE's Recovery of Capital	-	4,500,001	-	-	-	-	-	-	4,500,001
CLE's Premium Account	-	62,271	686,596	580,583	20,550	-	-	-	1,350,000
CLE's Payout	-	124,542	1,373,192	1,161,167	1,121,032	1,131,307	623,671	1,023,671	6,558,584
Total Payout to CLE	-	4,686,814	2,059,787	1,741,750	1,141,582	1,131,307	623,671	1,023,671	12,408,584
Payout to Sandy K	-	124,542	1,373,192	1,161,167	1,121,032	1,131,307	623,671	1,023,671	6,558,584
CLE'S NET CASH FLOW									
Total Payout to CLE	-	4,686,814	2,059,787	1,741,750	1,141,582	1,131,307	623,671	1,023,671	12,408,584
Less: Capital Investment	2,800,000	1,400,000	-	-	-	-	-	-	4,200,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
Federal Income Tax	-	241,289	394,583	471,930	324,078	402,622	222,086	214,086	2,270,673
Prov. Income Tax	-	47,517	256,027	225,284	154,704	153,711	79,512	90,845	1,007,599
Net Cash Flow to CLE	(2,800,000)	2,698,008	1,409,178	1,044,537	662,801	574,974	322,074	718,740	4,630,314

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 6.00/oz
$4,700,000 Capital Costs

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
REVENUE									
Ton of Ore Milled	-	225,000	225,000	225,000	225,000	225,000	225,000	225,000	1,575,000
Grade (oz/ton)	-	2.0	1.8	1.8	1.5	1.5	1.0	1.0	1.5
Recovery (%)	-	87.0	87.0	87.0	87.0	87.0	87.0	87.0	87.0
Production (ozs)	-	391,500	342,563	342,563	293,625	293,625	195,750	195,750	2,055,375
Revenue	-	3,132,001	2,740,501	2,740,501	2,349,001	2,349,001	1,566,000	1,566,000	16,443,002
OPERATING COSTS									
Mining & Milling	-	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	9,100,000
Refining Charges	-	116,589	102,015	102,015	87,442	87,442	58,294	58,294	612,091
	-	1,416,589	1,402,015	1,402,015	1,387,442	1,387,442	1,358,294	1,358,294	9,712,092
NET OPERATING PROFIT	-	1,715,412	1,338,486	1,338,486	961,559	961,559	207,706	207,706	6,730,914

CASH FLOW TO CLE

	1987	1988	1989	1990	1991	1992	1993	1994	Total
Net Operating Profit	-	1,715,412	1,338,486	1,338,486	961,559	961,559	207,706	207,706	6,730,914
Less: Capital Investment	3,200,000	1,500,000	-	-	-	-	-	-	4,700,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
NPI to Siscoe	-	-	-	-	-	-	-	-	-
Royalty to Sandy K	-	-	-	-	137,109	359,238	83,082	403,082	982,511
Federal Income Tax	-	-	-	-	104,659	121,970	25,236	100,000	351,865
Provincial Income Tax	-	-	-	-	49,961	58,224	12,047	50,000	170,232
Provincial Mining Duty	-	-	-	-	11,170	63,465	-	-	74,635
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Net Cash Flow to CLE	(3,200,000)	(84,588)	1,338,486	1,338,486	658,661	358,662	87,340	454,624	951,670
Accumulated Net Cash Flow	(3,200,000)	(3,284,589)	(1,946,103)	(607,618)	51,044	409,706	497,046	951,670	951,670
Rate of Return to CLE	-	-	-	-	.6	4.0	4.8	7.8	7.8
Present Value of NCF at 10%	(3,200,000)	(3,276,899)	(2,170,712)	(1,165,088)	(715,214)	(492,513)	(443,211)	(209,917)	(209,917)
Present Value of NCF at 15%	(3,200,000)	(3,273,555)	(2,261,469)	(1,381,393)	(1,004,801)	(826,483)	(788,723)	(617,814)	(617,814)
Present Value of NCF at 20%	(3,200,000)	(3,270,490)	(2,340,987)	(1,566,401)	(1,248,759)	(1,104,621)	(1,075,371)	(948,494)	(948,494)

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CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 6.00/oz
$4,700,000 Capital Costs

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
SISCOE ROYALTY CALCULATION									
Opening Unrecovered Costs	2,000,000	5,200,001	6,159,591	5,696,107	5,232,622	5,157,234	5,134,141	5,801,436	-
Plus: New Capital Investment	3,200,000	1,800,000	-	-	-	-	-	-	5,000,001
Operating Costs	-	1,416,589	1,402,015	1,402,015	1,387,442	1,387,442	1,358,294	1,358,294	9,712,092
Depreciation	-	875,000	875,000	875,000	875,000	875,000	875,000	875,000	6,125,001
Prov. Mining Duties	-	-	-	-	11,170	63,465	-	-	74,635
Less: Total Revenue	-	3,132,001	2,740,501	2,740,501	2,349,001	2,349,001	1,566,000	1,566,000	16,443,002
Closing Unrecovered Costs	5,200,001	6,159,591	5,696,107	5,232,622	5,157,234	5,134,141	5,801,436	6,468,731	-
Siscoe NPI	-	-	-	-	-	-	-	-	-
CALCULATION OF CASH DISTRIBUTION									
Operating Profit	-	1,715,412	1,338,486	1,338,486	961,559	961,559	207,706	207,706	6,730,914
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Less: NPI to Siscoe	-	-	-	-	-	-	-	-	-
Prov. Mining Duties	-	-	-	-	11,170	63,465	-	-	74,635
Net Cash for Distribution	-	1,715,412	1,338,486	1,338,486	950,389	898,094	207,706	1,007,706	7,456,279
CLE's Recovery of Capital	-	1,715,412	1,338,486	1,338,486	607,618	-	-	-	5,000,002
CLE's Premium Account	-	-	-	-	68,554	179,619	41,541	201,541	491,255
CLE's Payout	-	-	-	-	137,109	359,238	83,082	403,082	982,511
Total Payout to CLE	-	1,715,412	1,338,486	1,338,486	813,280	538,856	124,624	604,624	6,473,768
Payout to Sandy K	-	-	-	-	137,109	359,238	83,082	403,082	982,511
CLE'S NET CASH FLOW									
Total Payout to CLE	-	1,715,412	1,338,486	1,338,486	813,280	538,856	124,624	604,624	6,473,768
Less: Capital Investment	3,200,000	1,500,000	-	-	-	-	-	-	4,700,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
Federal Income Tax	-	-	-	-	104,659	121,970	25,236	100,000	351,865
Prov. Income Tax	-	-	-	-	49,961	58,224	12,047	50,000	170,232
Net Cash Flow to CLE	(3,200,000)	(84,588)	1,338,486	1,338,486	658,661	358,662	87,340	454,624	951,670

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 8.00/oz
$4,700,000 Capital Costs

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
REVENUE									
Ton of Ore Milled	-	225,000	225,000	225,000	225,000	225,000	225,000	225,000	1,575,000
Grade (oz/ton)	-	2.0	1.8	1.8	1.5	1.5	1.0	1.0	1.5
Recovery (%)	-	87.0	87.0	87.0	87.0	87.0	87.0	87.0	87.0
Production (ozs)	-	391,500	342,563	342,563	293,625	293,625	195,750	195,750	2,055,375
Revenue	-	4,176,001	3,654,001	3,654,001	3,132,001	3,132,001	2,088,000	2,088,000	21,924,004
OPERATING COSTS									
Mining & Milling	-	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	9,100,000
Refining Charges	-	127,029	111,150	111,150	95,272	95,272	63,514	63,514	666,901
	-	1,427,029	1,411,150	1,411,150	1,395,272	1,395,272	1,363,514	1,363,514	9,766,902
NET OPERATING PROFIT	-	2,748,972	2,242,851	2,242,851	1,736,729	1,736,729	724,486	724,486	12,157,106

CASH FLOW TO CLE

	1987	1988	1989	1990	1991	1992	1993	1994	Total
Net Operating Profit	-	2,748,972	2,242,851	2,242,851	1,736,729	1,736,729	724,486	724,486	12,157,106
Less: Capital Investment	3,200,000	1,500,000	-	-	-	-	-	-	4,700,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
NPI to Siscoe	-	-	-	-	-	-	-	-	-
Royalty to Sandy K	-	-	-	781,355	616,594	616,594	280,529	600,529	2,895,603
Federal Income Tax	-	-	59,094	295,953	226,827	226,827	89,901	125,101	1,023,704
Provincial Income Tax	-	-	28,210	141,278	108,280	108,280	42,916	61,982	490,945
Provincial Mining Duty	-	-	-	281,285	195,244	195,244	23,163	23,163	718,098
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Net Cash Flow to CLE	(3,200,000)	948,972	2,155,547	742,980	589,784	589,784	287,977	713,710	2,828,755
Accumulated Net Cash Flow	(3,200,000)	(2,251,028)	(95,481)	647,499	1,237,283	1,827,067	2,115,044	2,828,755	2,828,755
Rate of Return to CLE	-	-	-	10.0	16.4	20.6	22.1	24.7	24.7
Present Value of NCF at 10%	(3,200,000)	(2,337,298)	(555,854)	2,358	405,188	771,398	933,953	1,300,200	1,300,200
Present Value of NCF at 15%	(3,200,000)	(2,374,807)	(744,905)	(256,383)	80,828	374,054	498,555	766,865	766,865
Present Value of NCF at 20%	(3,200,000)	(2,409,190)	(912,283)	(482,318)	(197,892)	39,129	135,571	334,755	334,755

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 8.00/oz
$4,700,000 Capital Costs

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
SISCOE ROYALTY CALCULATION									
Opening Unrecovered Costs	2,000,000	5,200,001	5,126,031	3,758,181	2,671,616	2,005,131	1,338,646	1,512,322	-
Plus: New Capital Investment	3,200,000	1,800,000	-	-	-	-	-	-	5,000,001
Operating Costs	-	1,427,029	1,411,150	1,411,150	1,395,272	1,395,272	1,363,514	1,363,514	9,766,902
Depreciation	-	875,000	875,000	875,000	875,000	875,000	875,000	875,000	6,125,001
Prov. Mining Duties	-	-	-	281,285	195,244	195,244	23,163	23,163	718,098
Less: Total Revenue	-	4,176,001	3,654,001	3,654,001	3,132,001	3,132,001	2,088,000	2,088,000	21,924,004
Closing Unrecovered Costs	5,200,001	5,126,031	3,758,181	2,671,616	2,005,131	1,338,646	1,512,322	1,685,999	-
Siscoe NPI	-	-	-	-	-	-	-	-	-
CALCULATION OF CASH DISTRIBUTION									
Operating Profit	-	2,748,972	2,242,851	2,242,851	1,736,729	1,736,729	724,486	724,486	12,157,106
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Less: NPI to Siscoe	-	-	-	-	-	-	-	-	-
Prov. Mining Duties	-	-	-	281,285	195,244	195,244	23,163	23,163	718,098
Net Cash for Distribution	-	2,748,972	2,242,851	1,961,566	1,541,485	1,541,485	701,323	1,501,324	12,239,008
CLE's Recovery of Capital	-	2,748,972	2,242,851	8,178	-	-	-	-	5,000,002
CLE's Premium Account	-	-	-	390,678	308,297	308,297	140,265	300,265	1,447,801
CLE's Payout	-	-	-	781,355	616,594	616,594	280,529	600,529	2,895,603
Total Payout to CLE	-	2,748,972	2,242,851	1,180,211	924,891	924,891	420,794	900,794	9,343,404
Payout to Sandy K	-	-	-	781,355	616,594	616,594	280,529	600,529	2,895,603
CLE'S NET CASH FLOW									
Total Payout to CLE	-	2,748,972	2,242,851	1,180,211	924,891	924,891	420,794	900,794	9,343,404
Less: Capital Investment	3,200,000	1,500,000	-	-	-	-	-	-	4,700,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
Federal Income Tax	-	-	59,094	295,953	226,827	226,827	89,901	125,101	1,023,704
Prov. Income Tax	-	-	28,210	141,278	108,280	108,280	42,916	61,982	490,945
Net Cash Flow to CLE	(3,200,000)	948,972	2,155,547	742,980	589,784	589,784	287,977	713,710	2,828,755

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 10.00/oz
$4,700,000 Capital Costs

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
REVENUE									
Ton of Ore Milled	-	225,000	225,000	225,000	225,000	225,000	225,000	225,000	1,575,000
Grade (oz/ton)	-	2.0	1.8	1.8	1.5	1.5	1.0	1.0	1.5
Recovery (%)	-	87.0	87.0	87.0	87.0	87.0	87.0	87.0	87.0
Production (ozs)	-	391,500	342,563	342,563	293,625	293,625	195,750	195,750	2,055,375
Revenue	-	5,220,002	4,567,502	4,567,502	3,915,001	3,915,001	2,610,001	2,610,001	27,405,004
OPERATING COSTS									
Mining & Milling	-	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	9,100,000
Refining Charges	-	137,469	120,285	120,285	103,102	103,102	68,734	68,734	721,711
	-	1,437,469	1,420,285	1,420,285	1,403,102	1,403,102	1,368,734	1,368,734	9,821,712
NET OPERATING PROFIT	-	3,782,533	3,147,216	3,147,216	2,511,899	2,511,899	1,241,266	1,241,266	17,583,300

CASH FLOW TO CLE

	1987	1988	1989	1990	1991	1992	1993	1994	Total
Net Operating Profit	-	3,782,533	3,147,216	3,147,216	2,511,899	2,511,899	1,241,266	1,241,266	17,583,300
Less: Capital Investment	3,200,000	1,500,000	-	-	-	-	-	-	4,700,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
NPI to Siscoe	-	-	-	-	209,551	436,975	226,050	226,050	1,098,627
Royalty to Sandy K	-	-	660,277	1,084,876	790,130	757,771	452,100	852,101	4,597,257
Federal Income Tax	-	-	317,818	417,624	306,224	266,723	114,031	133,031	1,555,452
Provincial Income Tax	-	-	151,716	199,360	146,181	127,325	54,435	65,768	744,784
Provincial Mining Duty	-	-	279,057	435,027	327,023	327,023	111,015	111,015	1,590,160
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Net Cash Flow to CLE	(3,200,000)	1,982,533	1,738,349	1,010,330	732,790	596,082	283,635	653,302	3,797,019
Accumulated Net Cash Flow	(3,200,000)	(1,217,468)	520,881	1,531,211	2,264,001	2,860,083	3,143,718	3,797,019	3,797,019
Rate of Return to CLE	-	-	10.9	25.4	31.6	34.7	35.7	37.3	37.3
Present Value of NCF at 10%	(3,200,000)	(1,397,697)	38,955	798,031	1,298,537	1,668,657	1,828,761	2,164,009	2,164,009
Present Value of NCF at 15%	(3,200,000)	(1,476,058)	(161,618)	502,691	921,665	1,218,023	1,340,646	1,586,246	1,586,246
Present Value of NCF at 20%	(3,200,000)	(1,547,889)	(340,703)	243,979	597,369	836,921	931,909	1,114,234	1,114,234

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 10.00/oz
$4,700,000 Capital Costs

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
SISCOE ROYALTY CALCULATION									
Opening Unrecovered Costs	2,000,000	5,200,001	4,092,469	2,099,311	262,122	(1,047,754)	(2,357,630)	(2,612,881)	-
Plus: New Capital Investment	3,200,000	1,800,000	-	-	-	-	-	-	5,000,001
Operating Costs	-	1,437,469	1,420,285	1,420,285	1,403,102	1,403,102	1,368,734	1,368,734	9,821,712
Depreciation	-	875,000	875,000	875,000	875,000	875,000	875,000	875,000	6,125,001
Prov. Mining Duties	-	-	279,057	435,027	327,023	327,023	111,015	111,015	1,590,160
Less: Total Revenue	-	5,220,002	4,567,502	4,567,502	3,915,001	3,915,001	2,610,001	2,610,001	27,405,004
Closing Unrecovered Costs	5,200,001	4,092,469	2,099,311	262,122	(1,047,754)	(2,357,630)	(2,612,881)	(2,868,132)	-
Siscoe NPI	-	-	-	-	209,551	436,975	226,050	226,050	1,098,627
CALCULATION OF CASH DISTRIBUTION									
Operating Profit	-	3,782,533	3,147,216	3,147,216	2,511,899	2,511,899	1,241,266	1,241,266	17,583,300
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Less: NPI to Siscoe	-	-	-	-	209,551	436,975	226,050	226,050	1,098,627
Prov. Mining Duties	-	-	279,057	435,027	327,023	327,023	111,015	111,015	1,590,160
Net Cash for Distribution	-	3,782,533	2,868,159	2,712,190	1,975,326	1,747,901	904,201	1,704,201	15,694,512
CLE's Recovery of Capital	-	3,782,533	1,217,468	-	-	-	-	-	5,000,001
CLE's Premium Account	-	-	330,138	542,438	395,065	232,359	-	-	1,500,000
CLE's Payout	-	-	660,277	1,084,876	790,130	757,771	452,100	852,101	4,597,257
Total Payout to CLE	-	3,782,533	2,207,883	1,627,314	1,185,196	990,130	452,100	852,101	11,097,256
Payout to Sandy K	-	-	660,277	1,084,876	790,130	757,771	452,100	852,101	4,597,257
CLE'S NET CASH FLOW									
Total Payout to CLE	-	3,782,533	2,207,883	1,627,314	1,185,196	990,130	452,100	852,101	11,097,256
Less: Capital Investment	3,200,000	1,500,000	-	-	-	-	-	-	4,700,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
Federal Income Tax	-	-	317,818	417,624	306,224	266,723	114,031	133,031	1,555,452
Prov. Income Tax	-	-	151,716	199,360	146,181	127,325	54,435	65,768	744,784
Net Cash Flow to CLE	(3,200,000)	1,982,533	1,738,349	1,010,330	732,790	596,082	283,635	653,302	3,797,022

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 12.00/oz
$4,700,000 Capital Costs

Canadian Dollars

	1987	1988	1989	1990	1991	1992	1993	1994	Total
REVENUE									
Ton of Ore Milled	-	225,000	225,000	225,000	225,000	225,000	225,000	225,000	1,575,000
Grade (oz/ton)	-	2.0	1.8	1.8	1.5	1.5	1.0	1.0	1.5
Recovery (%)	-	87.0	87.0	87.0	87.0	87.0	87.0	87.0	87.0
Production (ozs)	-	391,500	342,563	342,563	293,625	293,625	195,750	195,750	2,055,375
Revenue	-	6,264,002	5,481,002	5,481,002	4,698,002	4,698,002	3,132,001	3,132,001	32,886,004
OPERATING COSTS									
Mining & Milling	-	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	1,300,000	9,100,000
Refining Charges	-	147,909	129,420	129,420	110,932	110,932	73,954	73,954	776,521
	-	1,447,909	1,429,420	1,429,420	1,410,932	1,410,932	1,373,954	1,373,954	9,876,522
NET OPERATING PROFIT	-	4,816,093	4,051,581	4,051,581	3,287,070	3,287,070	1,758,046	1,758,046	23,009,490

CASH FLOW TO CLE

	1987	1988	1989	1990	1991	1992	1993	1994	Total
Net Operating Profit	-	4,816,093	4,051,581	4,051,581	3,287,070	3,287,070	1,758,046	1,758,046	23,009,490
Less: Capital Investment	3,200,000	1,500,000	-	-	-	-	-	-	4,700,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
NPI to Siscoe	-	-	-	423,343	565,654	565,654	311,836	311,836	2,178,322
Royalty to Sandy K	-	-	1,311,562	1,215,788	1,013,145	1,131,307	623,671	1,023,671	6,319,144
Federal Income Tax	-	175,384	402,979	488,521	345,925	336,266	222,086	214,086	2,185,246
Provincial Income Tax	-	11,222	264,869	233,204	165,133	153,711	79,512	90,845	998,496
Provincial Mining Duty	-	-	588,769	588,769	458,802	458,802	198,868	198,868	2,492,877
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Net Cash Flow to CLE	(3,200,000)	2,829,486	1,483,403	1,101,957	738,411	641,331	322,074	718,740	4,635,403
Accumulated Net Cash Flow	(3,200,000)	(370,514)	1,112,889	2,214,846	2,953,257	3,594,588	3,916,662	4,635,403	4,635,403
Rate of Return to CLE	-	-	25.4	39.4	44.6	47.3	48.1	49.3	49.3
Present Value of NCF at 10%	(3,200,000)	(627,739)	598,215	1,426,131	1,930,476	2,328,692	2,510,494	2,879,322	2,879,322
Present Value of NCF at 15%	(3,200,000)	(739,577)	382,088	1,106,643	1,528,832	1,847,686	1,986,928	2,257,129	2,257,129
Present Value of NCF at 20%	(3,200,000)	(842,095)	188,046	825,752	1,181,854	1,439,590	1,547,452	1,748,039	1,748,039

CANADIAN LENCOURT MINES LTD.
Sandy K Project

Silver at $US 12.00/oz
$4,700,000 Capital Costs

Canadian Dollars

SISCOE ROYALTY CALCULATION

	1987	1988	1989	1990	1991	1992	1993	1994	Total
Opening Unrecovered Costs	2,000,000	5,200,001	3,058,909	471,097	(2,116,715)	(4,069,983)	(6,023,251)	(6,707,430)	-
Plus: New Capital Investment	3,200,000	1,800,000	-	-	-	-	-	-	5,000,001
Operating Costs	-	1,447,909	1,429,420	1,429,420	1,410,932	1,410,932	1,373,954	1,373,954	9,876,522
Depreciation	-	875,000	875,000	875,000	875,000	875,000	875,000	875,000	6,125,001
Prov. Mining Duties	-	-	588,769	588,769	458,802	458,802	198,868	198,868	2,492,877
Less: Total Revenue	-	6,264,002	5,481,002	5,481,002	4,698,002	4,698,002	3,132,001	3,132,001	32,886,004
Closing Unrecovered Costs	5,200,001	3,058,909	471,097	(2,116,715)	(4,069,983)	(6,023,251)	(6,707,430)	(7,391,609)	-
Siscoe NPI	-	-	-	423,343	565,654	565,654	311,836	311,836	2,178,322

CALCULATION OF CASH DISTRIBUTION

	1987	1988	1989	1990	1991	1992	1993	1994	Total
Operating Profit	-	4,816,093	4,051,581	4,051,581	3,287,070	3,287,070	1,758,046	1,758,046	23,009,490
Plus: Salvage Value	-	-	-	-	-	-	-	500,000	500,000
Recovery of Working Capital	-	-	-	-	-	-	-	300,000	300,000
Less: NPI to Siscoe	-	-	-	423,343	565,654	565,654	311,836	311,836	2,178,322
Prov. Mining Duties	-	-	588,769	588,769	458,802	458,802	198,868	198,868	2,492,877
Net Cash for Distribution	-	4,816,093	3,462,813	3,039,470	2,262,615	2,262,615	1,247,343	2,047,343	19,138,288
CLE's Recovery of Capital	-	4,816,093	183,908	-	-	-	-	-	5,000,001
CLE's Premium Account	-	-	655,781	607,894	236,325	-	-	-	1,500,000
CLE's Payout	-	-	1,311,562	1,215,788	1,013,145	1,131,307	623,671	1,023,671	6,319,144
Total Payout to CLE	-	4,816,093	2,151,251	1,823,682	1,249,470	1,131,307	623,671	1,023,671	12,819,144
Payout to Sandy K	-	-	1,311,562	1,215,788	1,013,145	1,131,307	623,671	1,023,671	6,319,144

CLE'S NET CASH FLOW

	1987	1988	1989	1990	1991	1992	1993	1994	Total
Total Payout to CLE	-	4,816,093	2,151,251	1,823,682	1,249,470	1,131,307	623,671	1,023,671	12,819,144
Less: Capital Investment	3,200,000	1,500,000	-	-	-	-	-	-	4,700,001
Working Capital	-	300,000	-	-	-	-	-	-	300,000
Federal Income Tax	-	175,384	402,979	488,521	345,925	336,266	222,086	214,086	2,185,246
Prov. Income Tax	-	11,222	264,869	233,204	165,133	153,711	79,512	90,845	998,496
Net Cash Flow to CLE	(3,200,000)	2,829,487	1,483,403	1,101,957	738,412	641,331	322,074	718,740	4,635,403

APPENDIX F

Capital Cost Summary Sheets

- Kilborn Limited

KILBORN

CLIENT: Watts, Griffis & McQuat

PROJECT: Sandy K Tails Remilling

PROJ. No. 3579

DIV. No. 15

AREA

AREA No.

ESTIMATOR: PGD, NN CHECKED

EXTENSIONS: NN. APPROVED

PROJECT ESTIMATE

TYPE

SHEET No. 1 OF

ESTIMATE No. 1

DATE: April, 1997

ACCOUNT CODE	DESCRIPTION	TYPE	QUANTITY	UNIT	LABOUR UNIT MH	LABOUR TOTAL MH	LABOUR RATE MH	LABOUR COST	MATERIAL UNIT COST	MATERIAL COST	SUB-CONTRACT UNIT COST	SUB-CONTRACT COST	TOTAL COST
	Summary												
100	– Sitework & Tailings											334000	
200	– Buildings & Structures											409000	
300	– Building Services											52000	
400	– Mechanical Equipment											2215000	
500	– Process Piping											570000	
600	– Electrical											250000	
700	– Instrumentation											50000	
	Total Direct Costs											3880000	
	Construction Indirects		(use	7%)								270000	
			Sub	Total								4150000	
	E.P.C.M. / Contingency		Excluded									—	
	TOTAL												

FORM KE 776 REV 1

KILBORN

PROJECT ESTIMATE

CLIENT: CON. LENCOURT MINES

PROJECT: SANDY K MINES

AREA:

PROJ. No. 3578

DIV. No. 15

AREA No.

ESTIMATOR: NN

EXTENSIONS: NN,

CHECKED:

APPROVED:

TYPE:

SHEET No. 2 OF

ESTIMATE No.

DATE: APRIL/87

ACCOUNT CODE	DESCRIPTION	TYPE	QUANTITY	UNIT	LABOUR				MATERIAL		SUB-CONTRACT		TOTAL COST
					UNIT MH	TOTAL MH	RATE MH	COST	UNIT COST	COST	UNIT COST	COST	
100	- SPREAD MINE WASTE		670	C.Y.							3-	2010	
	- FILTER FABRIC		18,060	S.F							011	1986	
	- 3" SAND CUSHION		180	C.Y.							12-	2160	
	- 35 MIL P.V.C. IMP. MEMBRANE		18,060	S.F							65	11739	
	- 3" SAND CUSHION		180	C.Y.							12	2160	
	- 3" HL4 ASPHALT		232	TONS							80-	18560	
	- CONSTRUCT BERM		1,660	C.Y.							6-	9960	
										(ADJ)		425	
	TOTAL									(K/E)		49000	

FORM KE 326 REV 1

KILBORN

PROJECT ESTIMATE

CLIENT: WATTS, GRIFFIS & McQUAT

PROJECT: SANDY K TAILS REMILLING

AREA:

PROJ. No. 3578

DIV. No. 15

AREA No.

ESTIMATOR NN. CHECKED

EXTENSIONS NN. APPROVED

TYPE

SHEET No. 8 OF

ESTIMATE No. 1

DATE APRIL 1987

ACCOUNT CODE	DESCRIPTION	TYPE	QUANTITY	UNIT	LABOUR				MATERIAL		SUB-CONTRACT		TOTAL COST
					UNIT MH	TOTAL MH	RATE MH	COST	UNIT COST	COST	UNIT COST	COST	
100	SITEWORK												
	- SITE GRADING, MINOR DEMOLITION, (GENERAL) SITE PREPARATION		L.S.									10000	
	- STRUCTURAL EXCAVATION		430	C.Y.							12-	5160	
	- BACKFILL WITH GRANULAR		300	C.Y.							15-	4500	
	- CRUSHED STONE BASE		145	C.Y.							25-	3625	
	TAILINGS												
	- CONSTRUCT DYKES		8,200	C.Y.							8-	65600	
	- DITCHING (5 CY/L.F.)		2800	L.F.							20-	56000	
	- 6" Ø POLY TAILINGS LINE		2000	L.F.							30-	60000	
										[illegible]		115	
	TOTAL										(C/F)	254000	

FORM KE 224 REV 1

KILBORN

PROJECT ESTIMATE

CLIENT: Watts, Griffis & McOuat

PROJECT: Sandy K Tails Remilling

AREA:

PROJ. No. 3578
DIV. No. 15
AREA No.

ESTIMATOR: RGJ
CHECKED:
EXTENSIONS: NN
APPROVED:

TYPE:
SHEET No. 4 OF
ESTIMATE No. 1
DATE: April, 1987

ACCOUNT CODE	DESCRIPTION	TYPE	QUANTITY	UNIT	LABOUR UNIT MH	LABOUR TOTAL MH	LABOUR RATE MH	LABOUR COST	MATERIAL UNIT COST	MATERIAL COST	SUB-CONTRACT UNIT COST	SUB-CONTRACT COST	TOTAL COST
	Tailings Reclaim Areas –												
	Drainage & Dewatering												
	Pumps												
	2 - Submersible (220V)												
	50 GPM c/w float levels & switches		L.S.									5500	
	2 - 2" ø trash pumps c/w												
	gas motors and extended												
	duty fuel tanks		L.S.									5000	
	5 - 100' lengths of fire hose												
	(1½" ø) c/w couplings and												
	misc. fittings		L.S.									1500	
	Portable generator (5 kW 110V/120V)												
	to drive submersible pumps		L.S.									2000	
	Portable power to drive												
	Tailings Reclaim System												
	Diesel generator set												
	115 kW - 600/575 V c/w voltage												
	regulator		L.S.									30000	
	Motor control centre c/w												
	disconnects & starters in												
	shelter		L.S.									9000	
	Stepdown transformer (600 to 240)		L.S.									2000	
	Maintenance shop		L.S.									25000	
	TOTAL											334000	

FORM KE 324 REV 1

KILBORN

PROJECT ESTIMATE

CLIENT: CAN. LENCOURT MINES

PROJECT: SANDY K MINES

AREA

PROJ. No. 3578

DIV. No. 15

AREA No.

ESTIMATOR NN

CHECKED

EXTENSIONS

APPROVED

TYPE

SHEET No. 5 OF

ESTIMATE No.

DATE APRIL/87

ACCOUNT CODE	DESCRIPTION	TYPE	QUANTITY	UNIT	LABOUR UNIT MH	LABOUR TOTAL MH	LABOUR RATE MH	LABOUR COST	MATERIAL UNIT COST	MATERIAL COST	SUB-CONTRACT UNIT COST	SUB-CONTRACT COST	TOTAL COST
200	BUILDINGS AND STRUCTURES												
	CONCRETE - FOOTINGS		80	C.Y.							300-	24000	
	- PIERS		14	C.Y.							600-	8400	
	- SLAB-ON-GRADE		28	C.Y.							275-	7700	
	- FOUND'N. & JACKING PIER TO BALL MILL		30	C.Y.							600	18000	
	- MISC. BASES & PADS		25	C.Y.							400-	10000	
	WOODEN PLATFORMS		5,500	S.F.							10-	55000	
	HANDRAILING		880	L.F.							15-	13200	
	STAIRS C/W HANDRAILS		80	L.F.							60-	4800	
	PLATFORM & SUPPORT STEEL		39	TONS							3000-	117000	
	PRE-ENGINEERED BUILDING 7500 S.F. (STEEL & ROOFING BUT NO SIDING)		7,500	S.F.							18-	135000	
	ASPHALT FLOOR		121	TONS							80-	9680	
	TRAILERS & CONTAINER SHELTERS		3	EA.							2000-	6000	
										(ADD)		220	
	TOTAL											400000	

FORM NE 220 REV 1

KILBORN

PROJECT ESTIMATE

CLIENT: WATTS, GRIFFIS & McQUAT

PROJECT: SOUDY K TAILS REMILLING

AREA:

PROJ. No. 3578

DIV. No. 15

AREA No.

ESTIMATOR: NW

CHECKED:

EXTENSIONS: NN.

APPROVED:

TYPE:

SHEET No. ___ OF ___

ESTIMATE No. 1

DATE: APRIL, 1989

ACCOUNT CODE	DESCRIPTION	TYPE	QUANTITY	UNIT	LABOUR UNIT MH	LABOUR TOTAL MH	LABOUR RATE MH	LABOUR COST	MATERIAL UNIT COST	MATERIAL COST	SUB-CONTRACT UNIT COST	SUB-CONTRACT COST	TOTAL COST
300.	BUILDING SERVICES												
	– HEATING & VENT		L.S.									5000	
	– PLUMBING & DRAINAGE		L.S.									22000	
	– LIGHTING		L.S.									15000	
	– FIRE PROTECTION – THIBAULT PS09 C/W HOSE & CONNECTIONS		L.S.									10000	
	TOTAL											52000	

FORM KE 320 REV 1

KILBORN

CLIENT: WATTS, GRIFFIS & McQUAT

PROJECT: SANDY K TAILINGS REMILLING

AREA:

PROJ. No. 9579

DIV. No. 15

AREA No.

ESTIMATOR: PED

EXTENSIONS: WW.

CHECKED:

APPROVED:

PROJECT ESTIMATE

TYPE:

SHEET No. 7 OF

ESTIMATE No. 1

DATE: APRIL, 1997

ACCOUNT CODE	DESCRIPTION	TYPE	QUANTITY	UNIT	LABOUR UNIT MH	LABOUR TOTAL MH	LABOUR RATE MH	LABOUR COST	MATERIAL UNIT COST	MATERIAL COST	SUB-CONTRACT UNIT COST	SUB-CONTRACT COST	TOTAL COST
	8' x 8' x 8' PUMPBOX MOUNTED ON SKID C/W FLARED HOPPER		1	LT								12000	12000
	8' x 8' GRIZZLY WITH 1" OPENINGS		1	EA								6000	6000
	5" x 4" SRL PUMP C/W 15 HP MOTOR.		1	EA		40	20-	800		6000			6800
	4' DIA x 6' HIGH PUMPBOX		1	EA		35		700		5000			5700
	5" x 4" SRL MILL FEED PUMP C/W 25 HP MOTOR		1	EA		40		800		6500			7300
	TAILINGS RECLAIM AREA 2½" SUMP PUMP C/W 15 HP MOTOR		2	EA	45	90		1800	5,000	10000			11800
	TOTAL							4100		27500		18000	49600

FORM VE 724 REV 1

KILBORN

PROJECT ESTIMATE

CLIENT: WATTS, GRIFFIS & McQUAT

PROJECT: SANDY K TAILINGS REMILLING

AREA:

PROJ. No. 8578 DIV. No. 15 AREA No.

ESTIMATOR PCD CHECKED

EXTENSIONS NW, APPROVED

TYPE

SHEET No. 8 OF

ESTIMATE No. 1

DATE APRIL, 1997

ACCOUNT CODE	DESCRIPTION	TYPE	QUANTITY	UNIT	LABOUR UNIT MH	LABOUR TOTAL MH	LABOUR RATE MH	LABOUR COST	MATERIAL UNIT COST	MATERIAL COST	SUB-CONTRACT UNIT COST	SUB-CONTRACT COST	TOTAL COST
	7' DIA x 10' MARCY BALL MILL SKID MOUNTED C/W 250HP MOTOR STARL STAND.	USED	2	EA	1000	2000	20^{00}	40000	157,200	314400			354400
	BALL MILL FEED & DISCHARGE CHUTES		12	TON							4000	48000	48000
	4' DIA x 6' MILL DISCHARGE PUMP BOX		1	EA		35		700		5000			5700
	5" x 4" SRL LEACH FEED PUMP C/W 15 HP MOTOR		1	EA		40		800		6000			6800
	BALL BUCKET		1	EA		10		200		2500			2700
	BALL MAGNET		1	EA		18		360		6000			6360
	INITIAL BALL CHARGE.		(EXCL. - PART OF OPERATING COST)										—
	TOTAL							42060		333900		49000	423960

FORM KE 226 REV 1

KILBORN

PROJECT ESTIMATE

CLIENT: WATTS, GRIFFIS & McQUAT

PROJECT: SANDY K TAILINGS REMILLING

AREA:

PROJ. No. 3578 | DIV. No. 15 | AREA No.

ESTIMATOR: PGD | CHECKED: | EXTENSIONS: NN | APPROVED:

TYPE: | SHEET No. 9 OF | ESTIMATE No. 1 | DATE: APRIL, 1987

ACCOUNT CODE	DESCRIPTION	TYPE	QUANTITY	UNIT	LABOUR UNIT MH	LABOUR TOTAL MH	LABOUR RATE MH	LABOUR COST	MATERIAL UNIT COST	MATERIAL COST	SUB-CONTRACT UNIT COST	SUB-CONTRACT COST	TOTAL COST
	22' DIA x 24' HIGH SURGE TANK		11	TON	20	220	20"	4400	3500	38500			42900
	SURGE TANK AGITATOR C/W 20 HP MOTOR		1	EA.		180		3600		30000			33600
	5" x 4" CYCLONE FEED PUMP C/W 20 HP MOTOR		1	EA		40		800		6200			7000
	CLUSTER OF 3 - 6" DIA CYCLONES C/W UNDERFLOW & OVERFLOW LAUNDERS		1	LT		100		2000		32000			34000
	GRINDING AREA 2½" SUMP PUMP C/W 15 HP MOTOR		1	EA		45		900		5000			5900
	TOTAL							11700		111700			123400

FORM KE 226 REV 1

KILBORN

PROJECT ESTIMATE

CLIENT: WATTS, GRIFFIS & McQUAT

PROJECT: SAND K TAILINGS REMILLING

AREA:

PROJ. No. 3578

DIV. No. 15

AREA No.

ESTIMATOR: PGD

EXTENSIONS: NW.

CHECKED:

APPROVED:

TYPE:

SHEET No. 10 OF

ESTIMATE No. 1

DATE: APRIL, 1997

ACCOUNT CODE	DESCRIPTION	TYPE	QUANTITY	UNIT	LABOUR				MATERIAL		SUB-CONTRACT		TOTAL COST
					UNIT MH	TOTAL MH	RATE MH	COST	UNIT COST	COST	UNIT COST	COST	
	31' DIA x 32' HIGH LEACH TANK	MINPRO	6	EA	420	2520	20=	50400	57,200	307200			357600
	LEACH TANK AGITATOR c/w 40 HP MOTOR	MINPRO	6	EA	300	1800		36000	52,500	315000			351000
	4' DIA x 6' HIGH FILTER FEED PUMPBOX		2	EA	35	70		1400	5,000	10000			11400
	5" x 4" SRL FILTER FEED PUMP c/w 15 HP MOTOR		2	EA	40	80		1600	6,000	12000			13600
	LEACH AREA 2½" SUMP PUMP c/w 15 HP MOTOR		2	EA	45	90		1800	5,000	10000			11800
	TOTAL							91200		654200			745400

KILBORN

PROJECT ESTIMATE

CLIENT: WATTS, GRIFFIS & McQUAT

PROJECT: SANDY K TAILINGS

AREA:

PROJ. No. 3578
DIV. No. 15
AREA No.

ESTIMATOR: PGD CHECKED:
EXTENSIONS: PGD APPROVED:

TYPE:
SHEET No. 11 OF
ESTIMATE No. 1
DATE: APRIL, 1987

ACCOUNT CODE	DESCRIPTION	TYPE	QUANTITY	UNIT	LABOUR UNIT MH	LABOUR TOTAL MH	LABOUR RATE MH	LABOUR COST	MATERIAL UNIT COST	MATERIAL COST	SUB-CONTRACT UNIT COST	SUB-CONTRACT COST	TOTAL COST
	11'-6" x 16' DRUM FILTER c/w REPULPER MOTORS, CLOTHS,	USED	4	EA	530	2120	20.00	42400	73,500	294000			336400
	3' DIA x 10' FILTRATE RECEIVER		2	EA	35	70		1400	6,000	12000			13400
	3' DIA x 10' MOISTURE TRAP		1	EA		35		700		6000			6700
	4" x 3" FILTRATE PUMP c/w 5 HP MOTOR		2	EA	20	40		800	1400	2800			3600
	3' DIA x 4' SEAL TANK		1	EA		20		400		3000			3400
	3000 CFM VACUUM PUMP c/w 200 HP MOTOR		2	EA	150	300		6000		76000			82000
	TOTAL							51700		393800			445500

FORM KE 776 REV 1

KILBORN

PROJECT ESTIMATE

CLIENT: WATTS, GRIFFIS & McQUAT

PROJECT: SANDY K TAILINGS

AREA:

PROJ. No. 3578
DIV. No. 15
AREA No.

ESTIMATOR: PGD
CHECKED:
EXTENSIONS: PGD
APPROVED:

TYPE:
SHEET No. 12 OF
ESTIMATE No. 1
DATE: APRIL, 1987

ACCOUNT CODE	DESCRIPTION	TYPE	QUANTITY	UNIT	LABOUR UNIT MH	LABOUR TOTAL MH	LABOUR RATE MH	LABOUR COST	MATERIAL UNIT COST	MATERIAL COST	SUB-CONTRACT UNIT COST	SUB-CONTRACT COST	TOTAL COST
	22' DIA x 24' HIGH SOLUTION TANK		2	EA	220	440	20	8800	38,500	77000			85800
	6"x4" TRANSFER/PROCESS WATER PUMP C/W 15 HP MOTOR.		2	EA	60	120		2400	7,700	15400			17800
	4' DIA x 6' TAILS PUMPBOX		1	EA		35		700		5000			5700
	5"x4" SRL TAILS PUMP C/W 25 HP MOTOR		1	EA		40		800		6500			7300
	2½" SUMP PUMP TO FILTER AREA C/W 15 HP MOTOR		1	EA		45		900		5000			5900
	TOTAL							13600		108900		—	122500

FORM KF 224 REV 1

KILBORN

PROJECT ESTIMATE

CLIENT: WATTS, GRIFFIS & McQUAT
PROJECT: SANDY K TAILINGS
AREA:
PROJ. No. 3578
DIV. No. 15
AREA No.
ESTIMATOR: PGD
EXTENSIONS: PGD
CHECKED:
APPROVED:
TYPE:
SHEET No. 13 OF
ESTIMATE No.
DATE:

ACCOUNT CODE	DESCRIPTION	TYPE	QUANTITY	UNIT	LABOUR UNIT MH	LABOUR TOTAL MH	LABOUR RATE MH	LABOUR COST	MATERIAL UNIT COST	MATERIAL COST	SUB-CONTRACT UNIT COST	SUB-CONTRACT COST	TOTAL COST
	PRECOAT BAG SPLITTER/HOPPER		1	LT		10	20$	200		1500			1700
	4' x 4' PRECOAT MIX TANK C/W 3/4 HP MIXER		1	EA		40		800		7500			8300
	2"x2" PRECOAT MIX FEED PUMP C/W 5 HP MOTOR		1	EA		40		800		3000			3800
	PRECIPITATION AREA 2½" SUMP PUMP C/W 10 HP MOTOR		1	EA		45		900		4800			5700
	MINPRO MMC LD500 P MERRILL CROWE SYSTEM C/W CLARIFIER, VACUUM PUMP, CROWE TOWER, ZINC INJ. SYST. PLATE & FRAME FILTER PRESS	MIN	1	LT		200		4000		162800			166800
	LEAD ACETATE FEEDER		1	EA		15		300		1000			1300
	TOTAL							7000		180600			187600

FORM KE 220 REV 1

KILBORN

PROJECT ESTIMATE

CLIENT: WATTS, GRIFFIS & McQUAT

PROJECT: SANDY K TAILINGS REMILLING

AREA:

PROJ. No. 3578

DIV. No. 15

AREA No.

ESTIMATOR: PGD

EXTENSIONS: NN.

CHECKED:

APPROVED:

TYPE:

SHEET No. 14 OF

ESTIMATE No. 1

DATE: APRIL, 1987

ACCOUNT CODE	DESCRIPTION	TYPE	QUANTITY	UNIT	LABOUR				MATERIAL		SUB-CONTRACT		TOTAL COST
					UNIT MH	TOTAL MH	RATE MH	COST	UNIT COST	COST	UNIT COST	COST	
	55 TON QUICKLIME STORAGE BIN		1	LT								15000	15000
	SCREW CONVEYOR 10' LONG C/W 1HP MOTOR.		1	EA		30	20	600		10000			10600
	LIME BIN VENT		1	EA		40		800		8000			8800
	6' DIA x 6' LIME SLURRY MIX TANK		1	EA		35		700		6000			6700
	AGITATOR FOR LIME MIX TANK. C/W 5 HP MOTOR		1	EA		25		500		11000			11500
	2" x 2" TRANSFER PUMP C/W 5HP MOTOR		1	EA		40		800		3000			3800
	6' DIA x 6' LIME STOCK TANK		1	EA		35		700		6000			6700
	2" x 2" LIME SLURRY DISTRIBUTION PUMP C/W 5 HP MOTOR		1	EA		40		800		3000			3800
	TOTAL							4900		41000		15000	66900

KILBORN

PROJECT ESTIMATE

CLIENT: WATTS, GRIFFIS & McQUAT

PROJECT: SANDY K TAILINGS REMILLING

AREA:

PROJ. No. 3578

DIV. No. 15

AREA No.

ESTIMATOR PSD

CHECKED

EXTENSIONS NN

APPROVED

TYPE

SHEET No. 13 OF

ESTIMATE No. 1

DATE APRIL 1987

ACCOUNT CODE	DESCRIPTION	TYPE	QUANTITY	UNIT	LABOUR UNIT MH	LABOUR TOTAL MH	LABOUR RATE MH	LABOUR COST	MATERIAL UNIT COST	MATERIAL COST	SUB-CONTRACT UNIT COST	SUB-CONTRACT COST	TOTAL COST
	6' DIA x 6' CYANIDE MIX TANK c/w COVER & HOPPER		1	EA		45	20	900		7000			7900
	AGITATATOR FOR CYANIDE MIX TANK c/w 1 HP MOTOR		1	EA		20		400		4000			4400
	7' DIA x 7' CYANIDE STOCK TANK		1	EA		40		800		9000			9800
	2" x 1" CYANIDE DISTRIBUTION PUMP c/w 1½ HP MOTOR		1	EA		10		200		1500			1700
	2" x 1" HCL PUMP c/w 1½ HP MOTOR		1	EA		10		200		1500			1700
	EXHAUST FAN & DUCTWORK c/w 1 HP MOTOR		1	LT								15000	15000
	5' DIA x 8' FRP HCL TANK		1	EA		15		300		2500			2800
	BARREL PUMP		1	EA		5		100		1000			1100
	REAGENTS AREA 1½" SUMP PUMP c/w 5 HP MOTOR		1	EA		35		700		4500			5200
	TOTAL							3600		31000		15000	49600

FORM KE 226 REV 1

KILBORN

CLIENT: WATTS, GRIFFIS & McQUAT

PROJECT: SANDY K TAILINGS

PROJ. No. 3578

DIV. No. 15

AREA

AREA No.

ESTIMATOR: PGD

CHECKED

EXTENSIONS: PGD

APPROVED

PROJECT ESTIMATE

TYPE

SHEET No. 16 OF

ESTIMATE No.

DATE

ACCOUNT CODE	DESCRIPTION	TYPE	QUANTITY	UNIT	LABOUR UNIT MH	LABOUR TOTAL MH	LABOUR RATE MH	LABOUR COST	MATERIAL UNIT COST	MATERIAL COST	SUB-CONTRACT UNIT COST	SUB-CONTRACT COST	TOTAL COST
	MECH EQUIP SUMMARY SHEET												
	SHEET NO 6							4100		27500		18000	49600
	" " 7							42060		333900		48000	423960
	" " 8							11700		111700		—	123400
	" " 9							91200		654200		—	745400
	" " 10							51700		393800		—	445500
	" " 11							13600		108900		—	122500
	" " 12							7000		180600		—	187600
	" " 13							4900		47000		15000	66900
	" " 14							3600		31000		15000	49600
			ADS					140		(400)			(260)
	TOTAL							230000		1889000		96000	2215000

FORM KE 320 REV. 1

KILBORN

PROJECT ESTIMATE

CLIENT: Watts, Griffis & McOuat

PROJECT: Sandy K Tailings Remilling

AREA:

PROJ. No. 3578

DIV. No. 15

AREA No.

ESTIMATOR NW. CHECKED

EXTENSIONS NW. APPROVED

TYPE

SHEET No. 17 OF

ESTIMATE No. 1

DATE April, 1987

ACCOUNT CODE	DESCRIPTION	TYPE	QUANTITY	UNIT	LABOUR				MATERIAL		SUB-CONTRACT		TOTAL COST
					UNIT MH	TOTAL MH	RATE MH	COST	UNIT COST	COST	UNIT COST	COST	
500	PROCESS PIPING												
	(FACTORED) 18% OF NEW												
	PROCESS MECHANICAL EQUIPT.		L.S.									570000	
600	ELECTRICAL												
	(FACTORED) 5% OF TOTAL												
	DIRECT COST		L.S.									250000	
700.	INSTRUMENTATION												
	LUMP SUM ALLOWANCE		L.S.									50000	
	TOTAL												

FORM 22 224 REV 1

APPENDIX G

Flov Sheets and General Layout Dravings

see map pouch

APPENDIX H

Operating Cost Analysis

APPENDIX H - OPERATING COST ANALYSIS

Operating costs for the Sandy "K" operation have been calculated based on 8 months of operations and 4 months of shutdown during the cold winter months.

1. TABLE HI

The following table summarizes all operating costs:

TABLE HI - SUMMARY

ITEM	ANNUAL COST	REMARKS
Labour - Table II	555,000	25% Fringe Incl.
Power - Table III	268,000	Includes Diesel Fuel
Reagents and Supplies - Table IV	345,000	Oper. and Mtce.
Office Rental	5,000	Offices
Environmental	30,000	Sampling Etc.
Property Taxes	By Owner	
Insurance	50,000	
Surface Transport (1 Vehicle)	5,000	Lease (includes gas)
TOTAL OPERATING COSTS	$1,258,000	Annual

2. TABLE HII

TABLE HII - PAYROLL COSTS

Position	Annual Rate	Months Per Year	No.	Salary Cost	Burdens @ 25%	Annual Cost
Mill Supt.	$48,000	8	1	$32,000	$8,000	$40,000
Mill Foreman	36,000	8	1	24,000	6,000	30,000
Assayer	36,000	8	1	24,000	6,000	30,000
Staff Sub-Total			3	80,000	20,000	100,000
Purchasing	15.00	8	1	20,880	5,220	26,100
Sample Prep.	11.00	8	1	15,312	3,828	19,140
Mechanic	13.50	8	1	18,792	4,698	23,490
Electrician	14.00	8	1	19,488	4,872	24,360
Mtce. Helper	12.00	8	1	16,704	4,176	20,880
Mill Operator	13.50	8	4	75,168	18,792	93,960
Loader Operator	13.00	8	4	72,384	18,096	90,480
Mill Helper	12.00	8	4	66,816	16,704	83,520
Labourer	10.50	8	4	58,464	14,616	73,080
Total Hourly			21	364,000	91,000	455,000
Total Payroll			24	444,000	111,000	555,000

3. POWER COST

3.1 MINING OPERATION

	H.P. Each	No.of Units	Total H.P.	Useage Factor	Consumption kW
5 x 4 Transfer Pump	15	1	15	.65	7.3
2 1/2" Vertical Pump	15	2	30	.65	14.5
5 x 4 Mill Feed Pump	25	1	25	.65	12.1
Lighting			1	1.00	.7
Dewatering Pumps	10	2	20	.65	9.7
TOTAL			91	.65	44.3

Power Source - Portable Diesel Powered Generator. Capacity 120 kW
Diesel Fuel Consumption is 6.5 USG per Hour @ 75 kW
4.2 USG per Hour @ 45 kW
Cost of Diesel Fuel is $1.30 per USG
.. Operating cost is $5.46 per Hour for fuel
or $12.13¢ per kWH.
Annual Cost = $0.1213 x 44.3 x 24 x 244 = $31,468.

3.2 MISCELLANEOUS POWER (Ontario Hydro)

Item	Total H.P.	Useage Factor	Consumption kW
Power Tools	25	.15	2.8
Mill Lighting			25.0
Office Lighting			3.0
Yard Lighting			15.0
Other			2.2
TOTAL			48.0

3.3 MILLING OPERATION (Ontario Hydro)

Item	H.P. Each	No.of Units	Total H.P.	Useage Factor	Consumption kW
31' Ø 32 Agitators	40	7	280	.64	133.6
Primary Cyclone Feed Pumps	20	1	20	.50	7.5
Mill Discharge Pump	15	1	15	.50	5.6
7'Ø x 10' Ball Mill	250	2	500	.95	354.4
2 1/2" Grinding Sump Pump	15	1	15	.25	2.8
2 1/2" Leach Area Sump Pump	15	2	30	.25	5.6
Filter Feed Pump	15	1	15	.56	5.6
Drum Filters	3+3	4	24	.50	10.0
3000 CFM Vacuum Pumps	200	2	400	.64	191.0
Filtrate Pumps	5	2	10	.50	3.7

3.3 MILLING OPERATION (Cont'd)

Item	H.P. Each	No.of Units	Total H.P.	Useage Factor	Consumption kW
Filter Cake Repulpers	3	4	12	.50	4.5
Tailing Pump	25	1	25	.50	9.3
2 1/2" Filter Area Sump Pump	10	1	10	.25	1.9
Process Water Pump	15	1	15	.70	7.8
Reclaim Water Pump	15	1	15	.70	7.8
200 CFM Vacuum Pump	10	1	10	.50	3.7
Precoat Mixer	1	1	1	.50	0.4
Precoat Pump	5	1	5	.50	1.9
Filter Press Feed Pump	30	1	30	.50	11.2
Lime Screw Feeder	1	1	1	.50	0.4
Lime Agitator	5	1	5	.50	1.9
Lime Pumps	5	2	10	.50	3.7
1 1/2" Reagent Area Sump Pump	5	1	5	.25	0.9
2 1/2" Precip. Area Sump Pump	10	1	10	.25	1.9
Cyanide Agitator	1	1	1	.50	0.4
Reagent Pumps	1 1/2	2	3	.50	1.1
Exhaust Fan	1	1	1	.50	0.4
TOTAL			1468	.71	779.0

3.4 CONTINGENT ALLOWANCE

	KW
KW (5%) (779.0 + 48) X 0.05	41.4

3.5 ONTARIO HYDRO CHARGES

Demand Charge
Up to 50 kW - No Charge
Next 4950 kW - $4.90 per kW
Based on peak demand

Energy Charge
First 250 kWH - 15.6 ¢/kWH
Next 11,250 kWH - 5.92 ¢/kWH
Next 1,288,000 kWH - 3.80 ¢/kWH

3.6 CONSUMPTION AND COST

A - 8 Months of Operation

Total kW Consumed 868.4W Peak = 1020 kW
Demand Charge (1020-50) x 4.90 = $4,753 per Month
Energy Charge - 24 Hr/Day and 30.5 Days/Month

	250 kWH @	.156 =	39
	11,250 kWH @	.0592 =	666
	624,169 kWH @	.0380 =	23,718
Totals	635,669 kWH		$24,423 per Month

Operating cost for 8 Months = 8 x $29,176
= $233,408

B - 4 Months Remaining

Total kW Consumed 23 kW Peak = 50 kW
Demand Charge (50-50) x 4.90 = 0
Energy Charge

	250 kWH @	.156 =	39
	11,250 kWH @	.0592 =	666
	5,336 kWH @	.0380 =	203
Totals	16,836		$908 per Month

Cost for 4 Months = 4 x 908
= $3,632

3.7 TABLE HIII

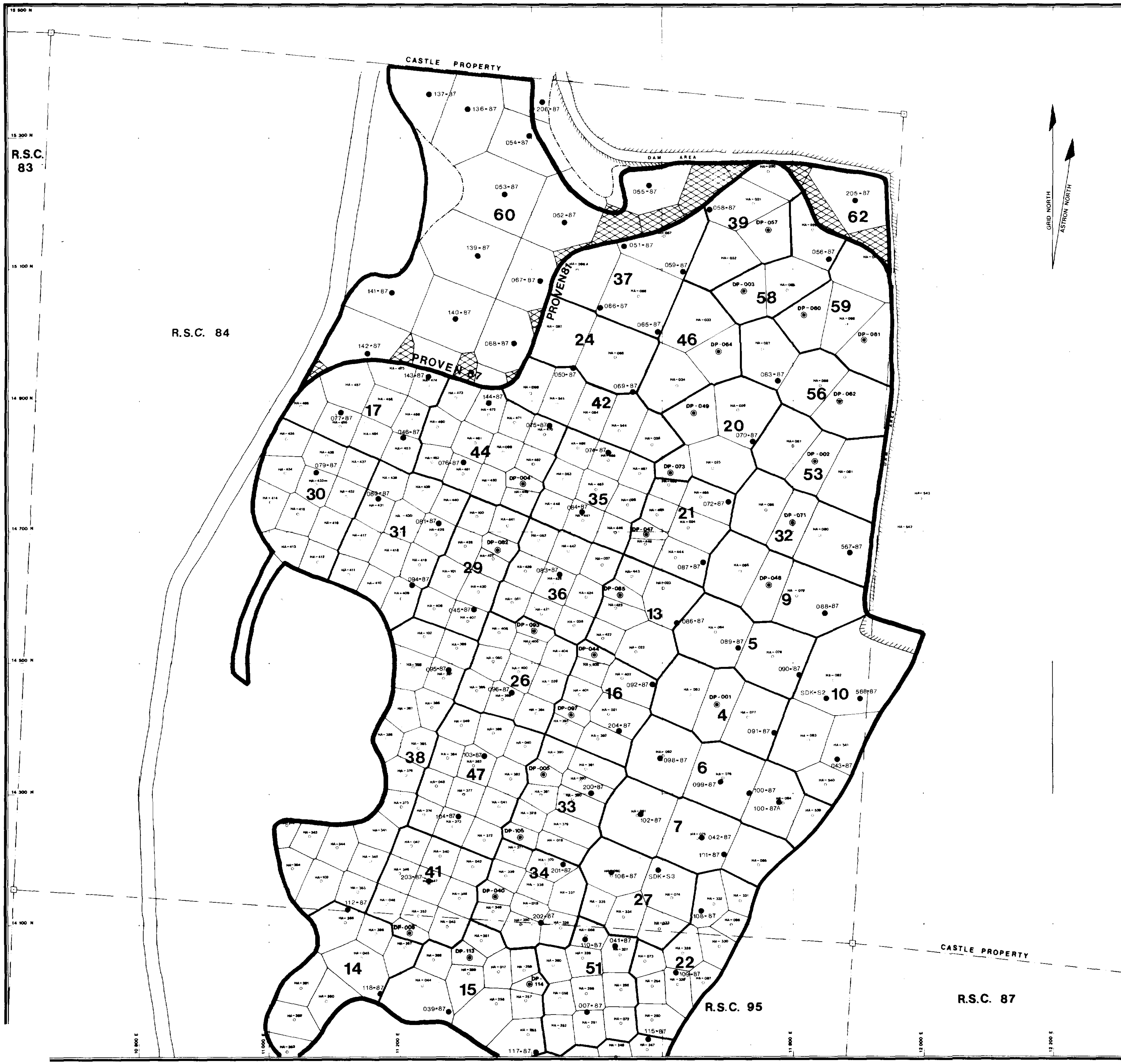
TABLE HIII - SUMMARY POWER COST

Location	Source	kW Consumed	Annual Cost
Mine	Portable Diesel	44.3	31,468
Mill & Office 8 Mo.	Ontario Hydro	645	233,408
Office Only 4 Mo.	Ontario Hydro	23	3,632
Total Power Cost			$268,508

6.4 TABLE IV

TABLE IV - COST OF REAGENTS AND SUPPLIES

Item	Type of Material	Useage Rate kg/ton	Unit Cost $/kg	Annual Cost
Balls	Mn Steel	.20	.69	$31,050
Mill Liners	Rubber	.015	4.40	14,850
NaCN	Bags	.20	1.70	76,500
Lime	Bulk Solid	.50	.13	14,625
HCl	Liquid	.05	.69	7,763
Precoat	Powder	.10	.84	18,900
Zn Dust	Powder	.0624	2.55	35,802
Lead Acetate	Powder	.016	5.04	18,144
Filter Media				37,000
Maintenance	Repair Parts			63,000
Operating	Supplies			15,000
Assaying	Supplies			8,000
Office	Supplies			4,000
TOTAL REAGENTS AND SUPPLIES				$344,634



Watts, Griffis & McOuat Limited
SANDY K MINES LTD.

TAILINGS RESERVE
SHEET Nº: 1A
(SURFACE TO 15' DEPTH)

SCALE 1" - 100'	SURVEY J.R., J.K.L., P.G.L.	APPROVED P.G.L.
PREPARED J.M., H.Z.	DATE 15/05/81	PROJECT Nº 839-1 DWG Nº 3

REVISION Nº		
1	Revised to include 1987 drilling	
2	For Kilborn Report	
3		
4		
5		
6		
7		

4	1	
5	2	
	3	

LEGEND

DRILLED BLOCK FOR RESERVE CALCULATION BOUNDARY INDICATES AREA OF INFLUENCE

○ HAND AUGER HOLE (TAILINGS)

● SONIC DRILL HOLE

◉ DRILLED DRIVE PIPE HOLE (TAILINGS)

ADDITIONS TO THE '81 PROVEN RESERVES

63.5000

OM87-6-P-065

41P10NE0023 63.5000 HAULTAIN

200

Watts,Griffis & McOuat Limited
SANDY K MINES LTD.

TAILINGS PLAN

SHEET Nº: 1B

SCALE 1"-100'	SURVEY: J.RAE	APPROVED: R.G.L.
PREPARED: J.M., H.Z.	DATE: 29/04/81	PROJECT Nº: 639-1 DWG Nº:

REVISION Nº	
1	REVISED AFTER 1987 DRILLING
2	FOR KILBORN REPORT
3	
4	
5	
6	
7	

4	1
5	2
	3

LEGEND

- PROPOSED DRIVE PIPE HOLE (TAILINGS)
- DRILLED DRIVE PIPE HOLE (TAILINGS)
- SONIC DRILL HOLE

Note

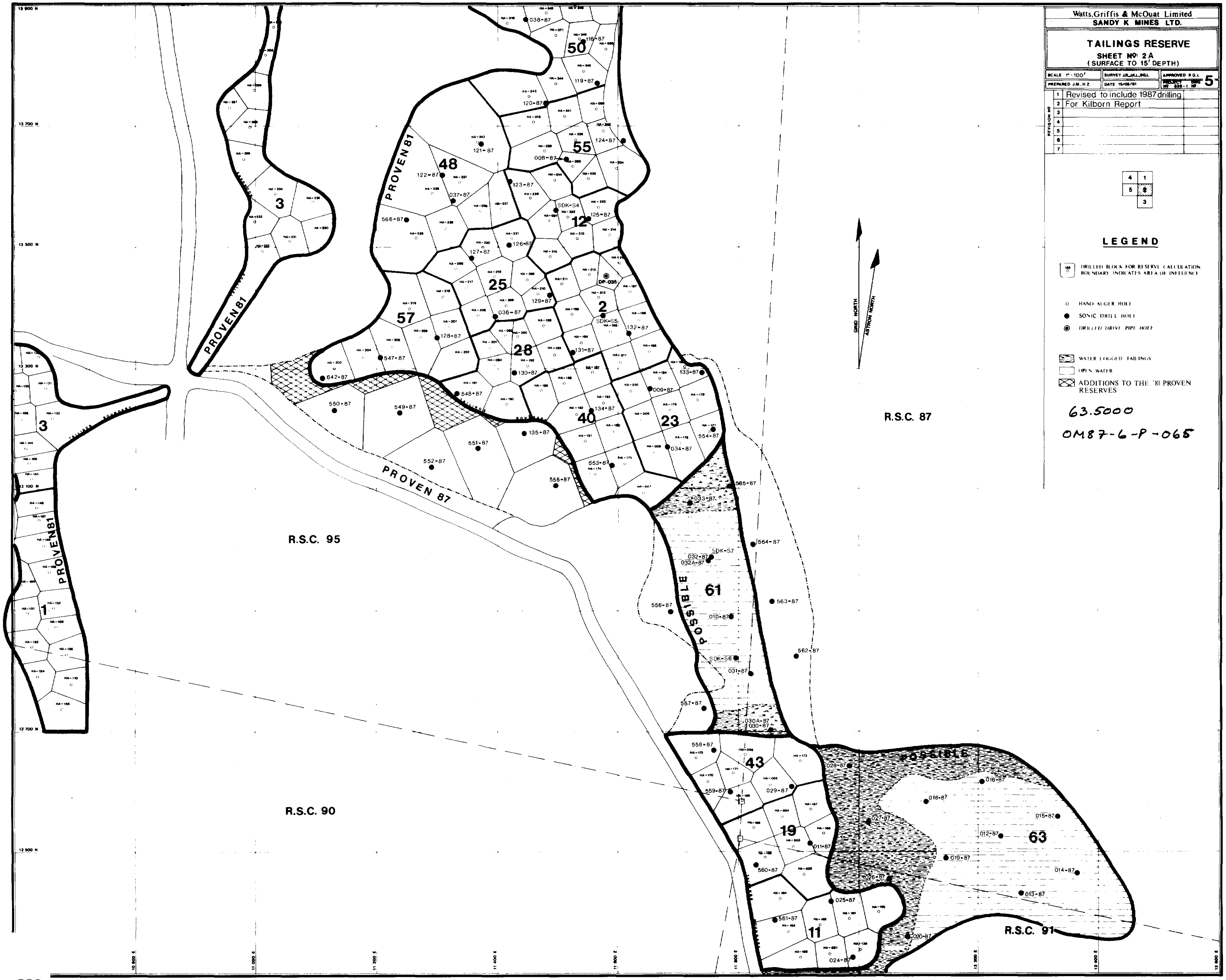
The inferred 15' Isopach is also the limit of the reserve below 15'

63.5000

OM87-6-P-065

41P10NE0023 63.5000 HAULTAIN

210



Watts,Griffis & McOuat Limited
SANDY K MINES LTD.
TAILINGS RESERVE
SHEET Nº: 2A
(SURFACE TO 15' DEPTH)
SCALE 1" - 100'
PREPARED J.M., H.Z.
DATE 15/06/81
APPROVED P.G.L.
5
REVISION Nº
1 Revised to include 1987 drilling
2 For Kilborn Report
LEGEND
DRILLED BLOCK FOR RESERVE CALCULATION. BOUNDARY INDICATES AREA OF INFLUENCE
HAND AUGER HOLE
SONIC DRILL HOLE
DRILLED DRIVE PIPE HOLE
WATER LOGGED TAILINGS
OPEN WATER
ADDITIONS TO THE '81 PROVEN RESERVES
63.5000
OM87-6-P-065
GRID NORTH
ASTRON NORTH
PROVEN 81
PROVEN 87
POSSIBLE
R.S.C. 87
R.S.C. 95
R.S.C. 90
R.S.C. 91
50
55
48
12
25
2
57
28
23
40
3
1
61
43
19
11
63

220

Watts,Griffis & McOuat Limited
SANDY K MINES LTD.

TAILINGS PLAN
SHEET Nº: 2B

SCALE: 1" - 100'	SURVEY:	APPROVED: P.G.L.
PREPARED: J.M., H.Z.	DATE: 29/04/81	PROJECT Nº: 639-1 DWG Nº:

REVISION Nº		
1	REVISED AFTER 1987 DRILLING	
2	FOR KILBORN REPORT	
3		
4		
5		
6		
7		

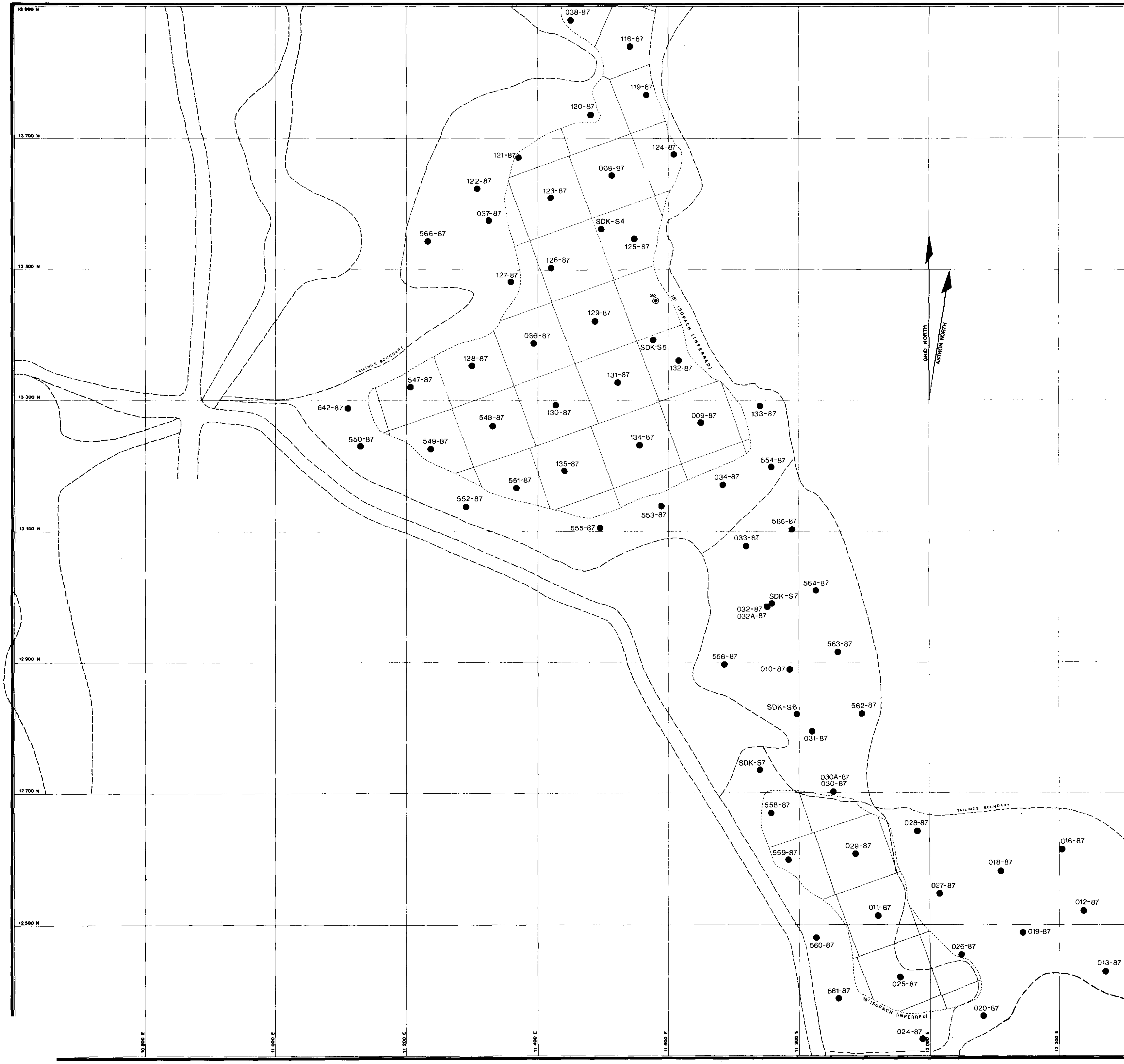
4	1
5	2
	3

LEGEND

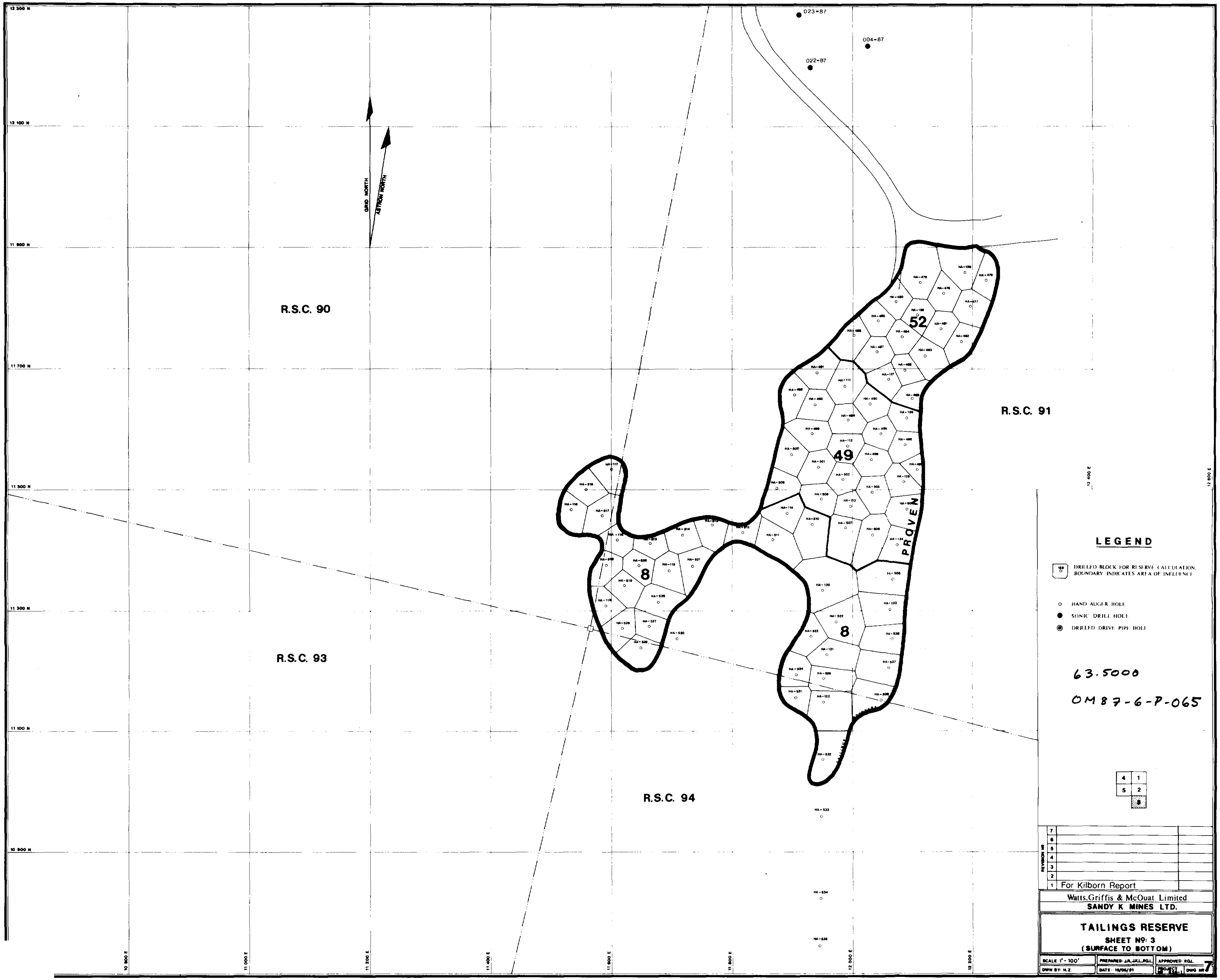
◎ PROPOSED DRIVE PIPE HOLE (TAILINGS)
◉ DRILLED DRIVE PIPE HOLE (TAILINGS)
● SONIC DRILL HOLE

Note:
The inferred 15' Isopach is also the limit of the reserve below 15'

65.5000
OM87-6-P-065



41P10NE0023 63.5000 HAULTAIN

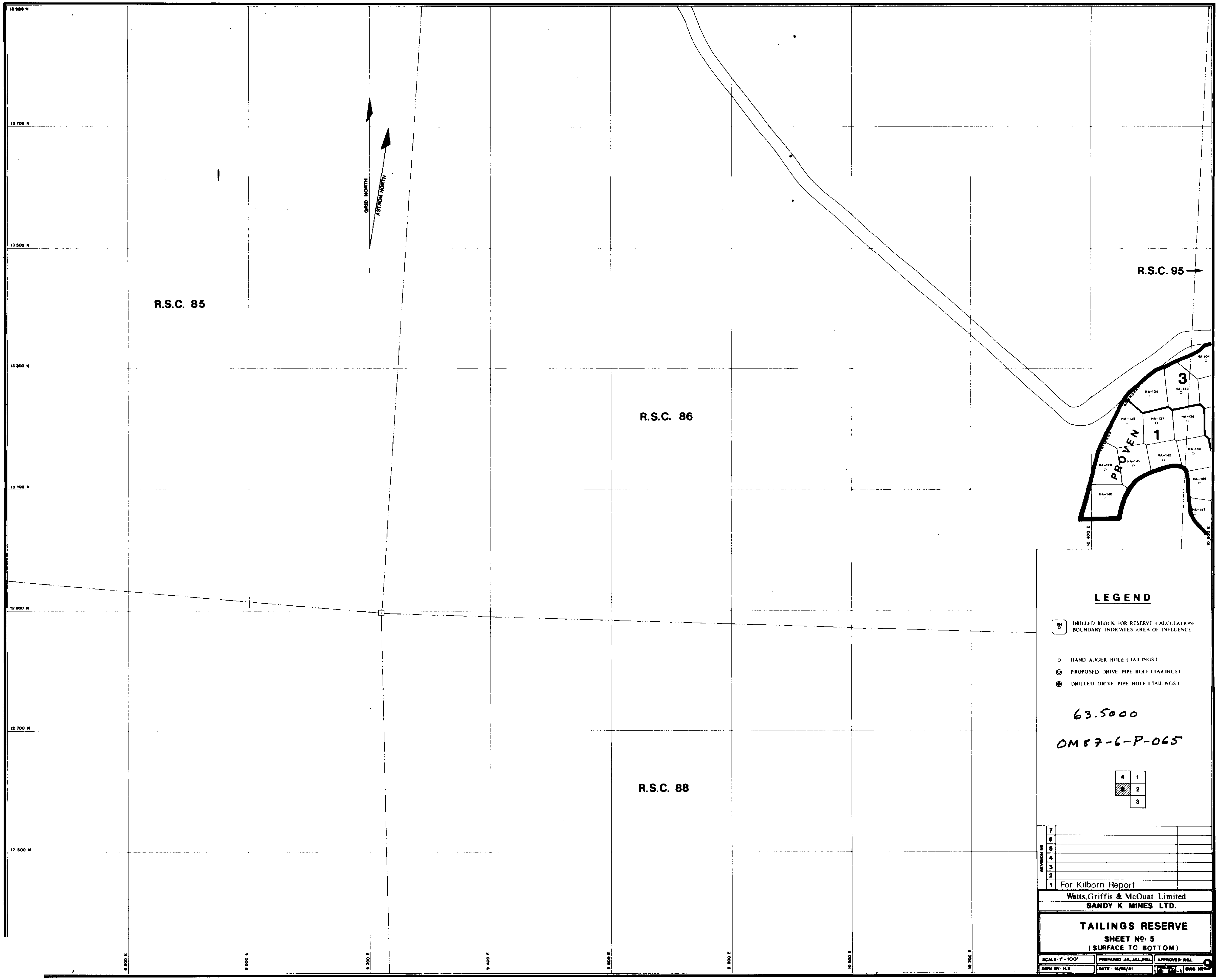


41P10NE0023 63.5000 HAULTAIN

240

41P10NE0023 63.5000 HAULTAIN

250



LEGEND

DRILLED BLOCK FOR RESERVE CALCULATION: BOUNDARY INDICATES AREA OF INFLUENCE

HAND AUGER HOLE (TAILINGS)

PROPOSED DRIVE PIPE HOLE (TAILINGS)

DRILLED DRIVE PIPE HOLE (TAILINGS)

63.5000

OM87-6-P-065

REVISION Nº	
7	
6	
5	
4	
3	
2	
1	For Kilborn Report

Watts, Griffis & McOuat Limited

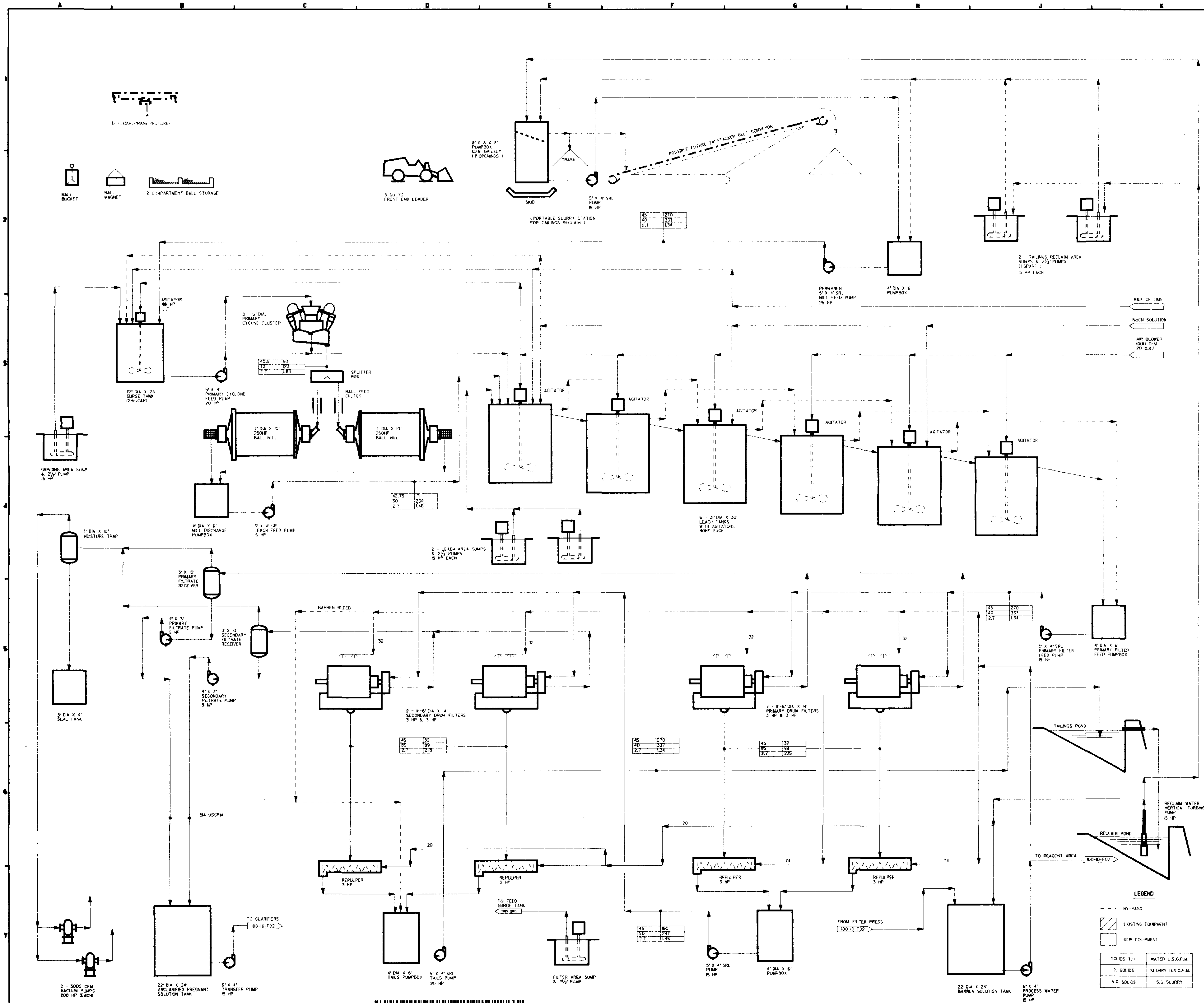
SANDY K MINES LTD.

TAILINGS RESERVE

SHEET Nº: 5

(SURFACE TO BOTTOM)

SCALE: 1" - 100'	PREPARED: J.R., J.K.L., R.G.L.	APPROVED: R.G.L.
DWN BY: H.Z.	DATE: 15/06/81	DWG Nº: 9



63.5000

OM87-6-P-065

NO.	DESCRIPTION	DATE	BY	CHECK	CLIENT	PROJ. ENG.		
C	RELEASED FOR FEASIBILITY STUDY	30 APR 1987	LL					
B	RELEASED FOR FEASIBILITY STUDY	16 APR 1987	SF					
A	RELEASED FOR INFORMATION		JS					

REVISIONS

DWG. NO.	DESCRIPTION

REFERENCE DRAWINGS

CLIENT: CANADIAN LENCOURT MINES LTD.

LOCATION: GOWGANDA ONTARIO

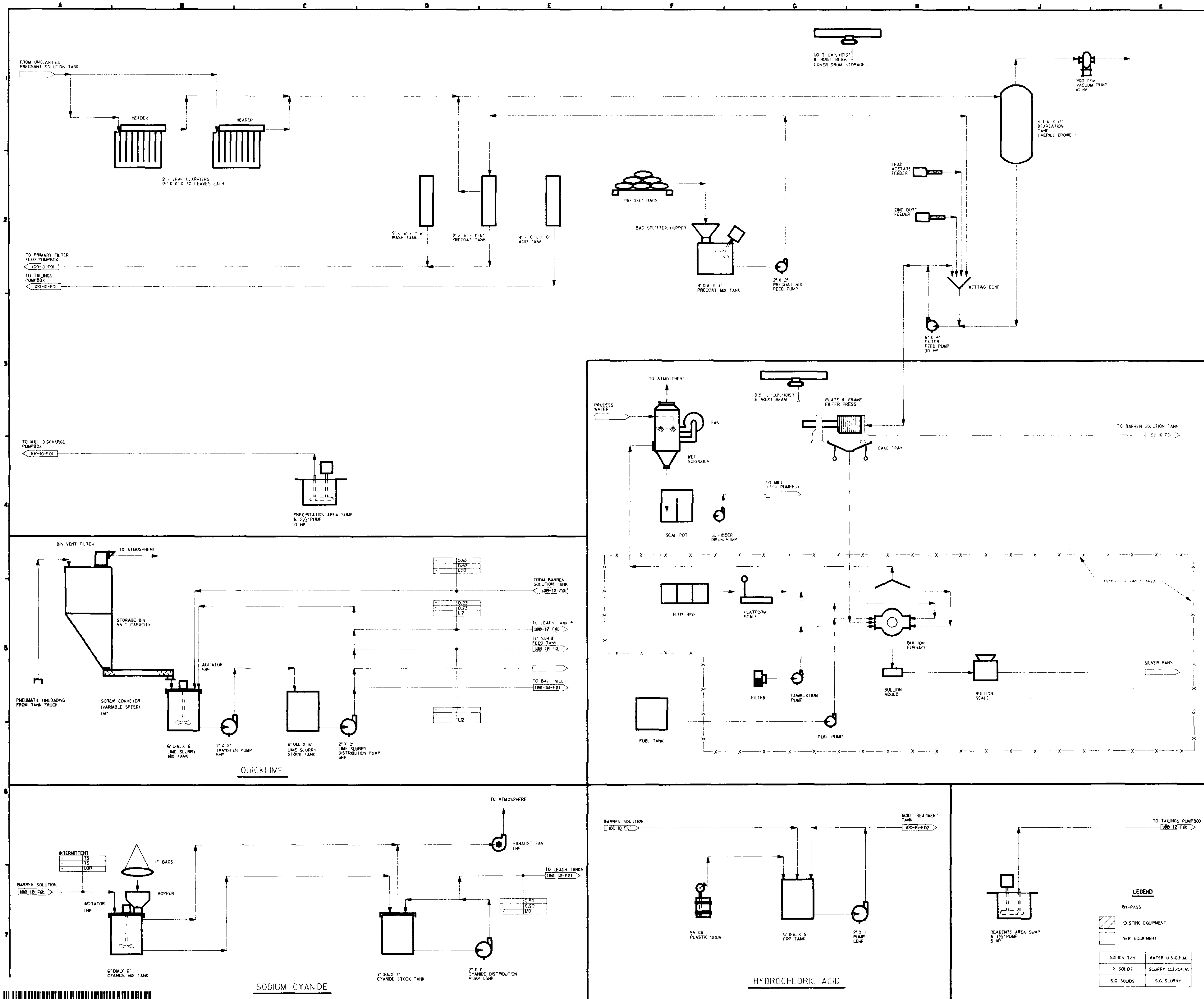
KILBORN

SECTION: GENERAL LAYOUT
SCALE: NONE
DESIGNED: J.H.STARKEY — FEB 87
DRAWN: J.F.SCARLETT — 17 FEB 1987
CHECKED:
APPROVED:
B.O.M. NO.
PROJECT NO.: 3578
DIVISION NO.: 15

TITLE

SANDY K MINES
SILVER TAILINGS RECOVERY
PROCESS FLOWSHEET
GRINDING, LEACHING & FILTRATION

DRAWING NUMBER: 100-10-F01
REV.: C



63.5000

OM87-6-P-065

NO.	DESCRIPTION	DATE	BY	CHECK	CLIENT	FINAL DWG.
C	RELEASED FOR FEASIBILITY STUDY	30 APR 1987	LL			
B	RELEASED FOR FEASIBILITY STUDY	16 APR 1987	SF			
A	RELEASED FOR INFORMATION		JS			

REVISIONS

DWG. NO.	DESCRIPTION

REFERENCE DRAWINGS

CLIENT: CANADIAN LENCOURT MINES LTD.

LOCATION: GOWGANDA ONTARIO

KILBORN

SECTION	GENERAL LAYOUT	
SCALE:	NONE	DATE
DESIGNED:	J.H.STARKEY	FEB 87
DRAWN:	J.F.SCARLETT	17 FEB 1987
CHECKED:		
APPROVED:		
B.O.M. NO.		

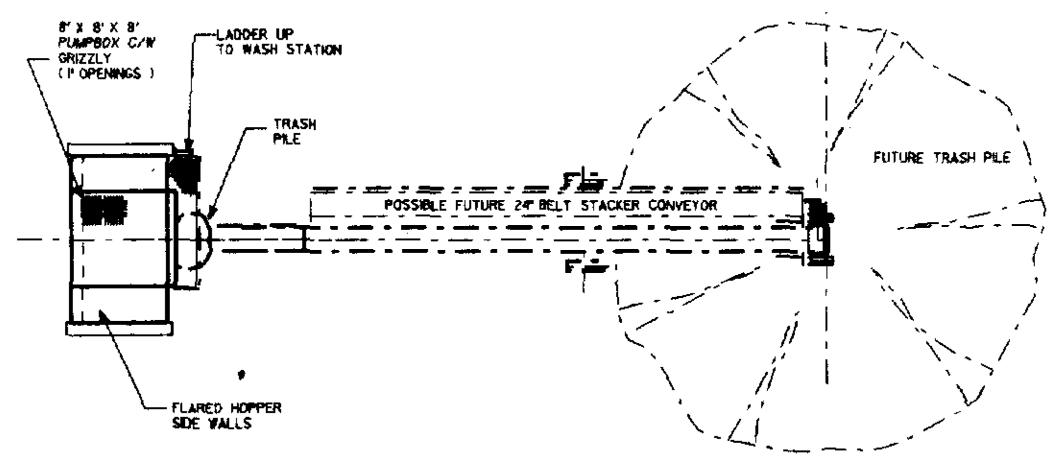
TITLE

SANDY K MINES
SILVER TAILINGS RECOVERY
PROCESS FLOWSHEET
PRECIPITATION AND REAGENTS

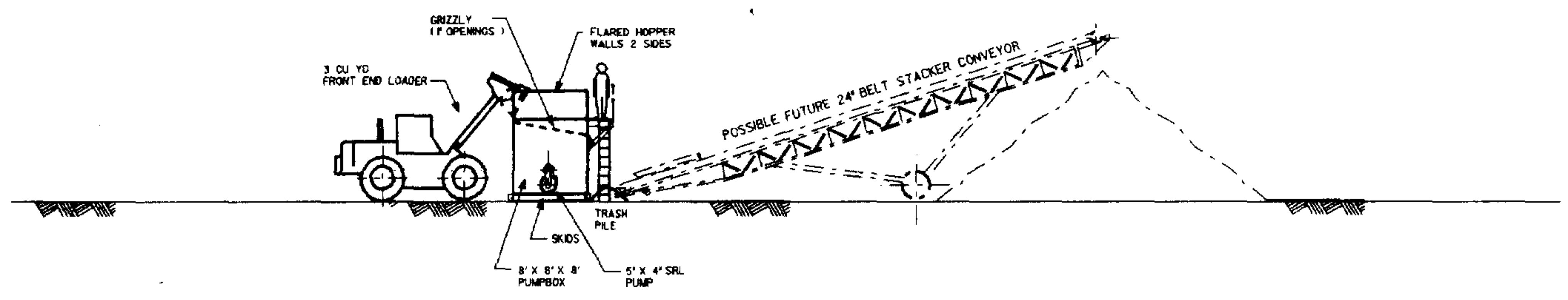
PROJECT NO.	DIVISION NO.
3678	15

DRAWING NUMBER: 100-10-F02 REV. C

NOTES



PLAN



SECTION

63.5000
OM87-6-P-065

LEGEND

CHECKER PLATE
GRATING
CONCRETE

0 5 10 15 20 25
SCALE 1/8" = 1'-0"

NO.	DESCRIPTION	DATE	BY	CHECK	CLIENT	PROJ. ENG.
A	RELEASED FOR FEASIBILITY STUDY	16 APR 1987	SF			

REVISIONS

DWG. NO.	DESCRIPTION

REFERENCE DRAWINGS

CLIENT: CANADIAN LENCOURT MINES LTD.
LOCATION: GOWGANDA ONTARIO

KILBORN

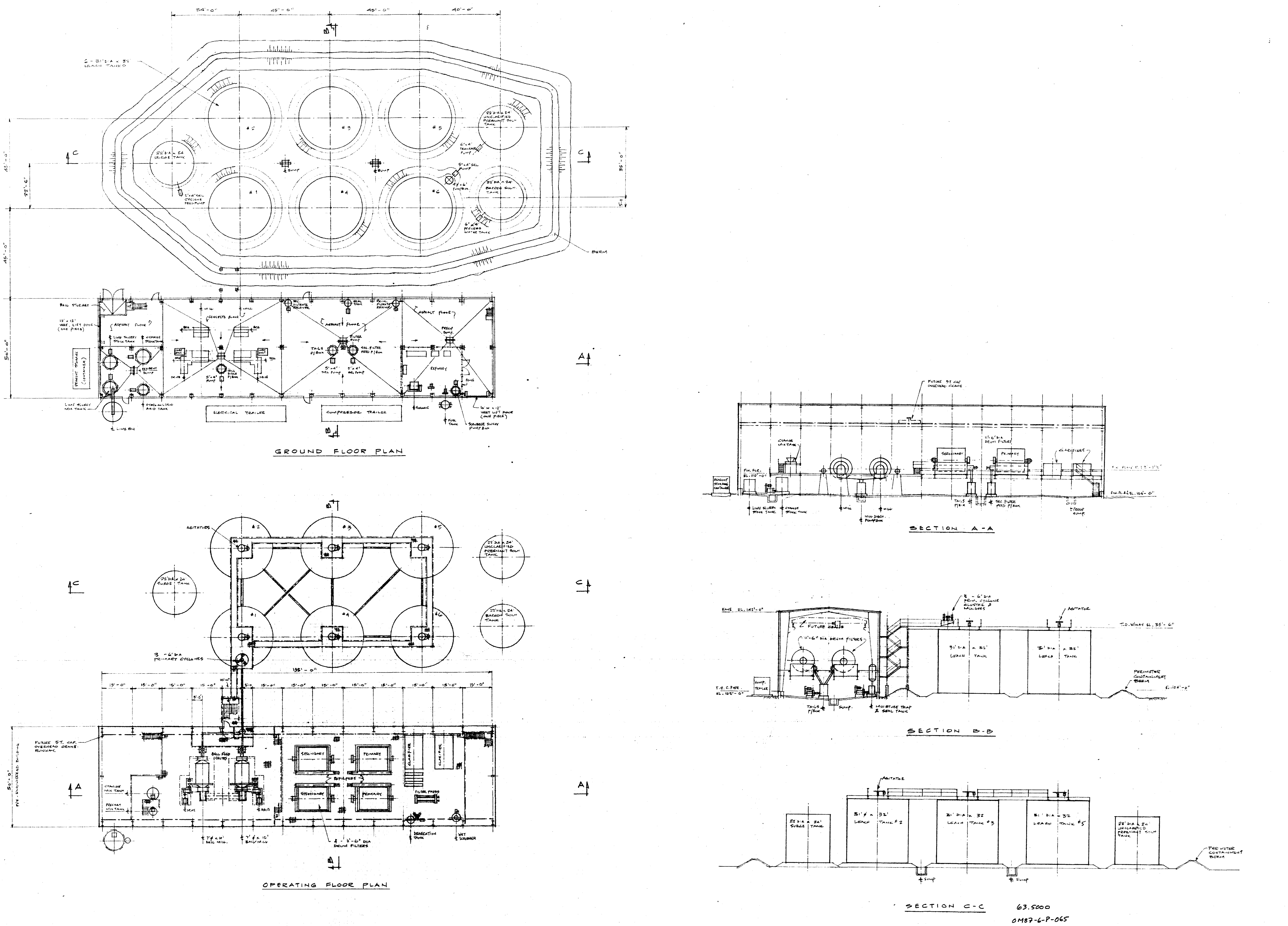
TITLE

SANDY K MINES
SILVER TAILINGS RECOVERY
PORTABLE SLURRY STATION
GENERAL ARRANGEMENT
PLAN AND SECTION

SECTION:	GENERAL LAYOUT	
SCALE:	1/8" = 1'-0"	DATE
DESIGNED:	STEVE FOEY	FEB 87
DRAWN:	STEVE FOEY	23-2-87
CHECKED:		
APPROVED:		
E.G.M. NO.		

PROJECT NO.	DIVISION NO.
3578	15

DRAWING NUMBER: 100-10-F04 REV. A



63.5000
0M87-6-P-065

NO.	DESCRIPTION	DATE	BY
E	RELEASED FOR FEASIBILITY STUDY	1 MAY 87	SF
	2ND BALL MILL & REFINERY ADDED. CRANE BECOMES 'FUTURE'. BLDG REVISED	—	—
D	REDESIGNED & REDRAWN. RELEASED FOR FEASIBILITY STUDY	16 APR 87	SF

REVISIONS

SECTION: GENERAL LAYOUT
SCALE: 1/16" = 1'-0"
DESIGNED: STEVE FOEY — 15 APR 87
DRAWN: STEVE FOEY — 15 APR 87
CHECKED:
APPROVED:

CLIENT: CANADIAN LENCOURT MINES LTD.
LOCATION: GOWGANDA ONTARIO

KILBORN

TITLE
SANDY K MINES
SILVER TAILINGS RECOVERY
MILL BUILDING
GENERAL ARRANGEMENT
PLANS & SECTIONS

PROJECT NO. 3578
DIVISION NO. 15
DRAWING NUMBER 100-10-F05
REV. E

300