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Minerex Resources Inc.
P O Box 10075
Reno, NV 89510-0075
Tel & Fax: (702) 851-4265

PRELIMINARY FEASIBILITY STUDY
EAST HUMBOLDT VEIN
AURORA, NEVADA
- 1981 ORE BLOCKS -

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Miller-Kappes Company

P. O. Box 13687, Reno, Nevada 89507 702-356-7107

1845 Glendale Avenue, Sparks, Nevada 89431

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PRELIMINARY FEASIBILITY STUDY

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- 1981 ORE BLOCKS -

INTRODUCTION AND GEOLOGIC SETTING

This report presents a very brief evaluation of the economics of an open pit mine and heap leach on a near-surface ore block drilled out in the fall of 1981 (hereafter called ore block 1981 A), on the east Humboldt Vein System, Aurora, Nevada.

Figure 1 presents a large scale map of the property showing the proposed open pit and the proposed heap leach site. In the initial 1982 mining program, a total of 236,000 tons of material - 56,000 tons of ore and 180,000 tons of waste - will be mined. Waste will be dumped near the mine and ore will be hauled approximately one mile to a stockpile at the heap leach site. Stockpiled ore will be crushed to 1/2-inch in a closed-circuit, two-stage crusher, stacked on plastic pads 15 feet deep, and heap leached. Based on extensive test work (Appendix A), estimated recoveries will be 65 percent of fire-assayable gold, in approximately two months of leaching. Some minor additional recovery might be possible after the heap sits dormant during the winter months.

The ore zone consists of two parallel, continuous quartz veins that dip about 70 degrees to the west. An area designated "Ore Block A" on the accompanying map contains proven ore which has been extensively drilled and sampled. A second area of possible ore, which will be excavated as part of the access ramp into the proposed pit, has been designated "Ore Block B". Figure 2 is a plan view on a scale of 1" = 20', and Figures 3 through 9 are ore reserve cross-sections, as prepared by Bruce W. Miller assisted by Joanne Seegelken and Andy Glatiotis. Based on an arbitrary pit depth of 100 feet, the overall ore tonnage which he calculates in ore block 1981 A, is 41,224 tons, with an average grade of 0.116 ounce gold per ton. Stanley Reamsbottom, a geologist in Vancouver, B.C., independently calculated a slightly larger pit to contain 49,503 tons of ore, assaying 0.105 ounces gold.

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per ton. The two pit designs contain within 8 percent of the same amount of gold, and in this study, ore block 1981 A will be assumed to contain 41,000 tons of open pit mineable ore, with a head grade of 0.116 ounces gold per ton.

Because of the pit design, it seems to be overly conservative to base the financial analysis on only the proven reserves mentioned above. The West vein, as it extends north from the pit, is exposed on the surface as a strong, continuous quartz outcrop for several hundred feet. The final pit design calls for construction of a pit access road from the north, which, during construction, will mine out over 15,000 tons of this West vein material lying north of the proven ore reserves. Surface and limited underground sampling indicate that this 15,000 ton zone (designated ore block 1981 B) is lower grade than the target ore block. However, it seems reasonably safe to impute an ore grade of, at least, 0.05 ounces gold per ton, over the 15 foot projected width (haulage road cross-sections shown in Figures 10 through 14) that was used to calculate the tonnage. Ore reserves in this area are still classified as potential by the project geologists. Thus, the ore may not materialize. To allow for this probability, after calculation of the feasibility analysis, a short analysis was run, assuming only ore block A is present. That analysis shows that the overall project will still show a slight operating profit, hence, it appears justified to base the mine design and economic projections on the assumption that this ore zone is there.

ADDITIONAL MINING

This feasibility study is based on two ore blocks, containing 41,000 tons of proven ore, and another 15,000 tons of possible ore. Geologic projections previously made for the property (see S. Reamsbottom report dated April 3, 1981), indicate that there are other potential ore reserves within 100 feet of the surface, which may be open-pittable under economic conditions similar to those present here. At-depth reserves, greater than 100 feet deep and accessible by underground mining, also represent considerable tonnage, which is currently classified as probable, and has not been considered as part of this feasibility study.

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It is expected that mining will commence in 1982 on the limited pit proposed here, with all mining and most leach operations completed by the end of 1982. Additional geologic evaluation of other reserves should lead to a continuation of mining in subsequent years.

MINING

DESCRIPTION OF OPERATION

Access. Access to the orebody for mining will require the construction of two access roads onto the orebody, as shown in orange in Figure 1. The natural outcrop of the ore zone ranges in elevation from 7380 feet to about 7460 feet. (In Figures 2 through 14, add 6200 feet to the elevation shown to get the approximate true elevation corresponding to Figure 1.) The upper access road cuts the ore zone at elevation 7400, and the lower access road cuts the ore zone at elevation 7300. Minor offsets from these roads can be used to access the orebody on intermediate 20 foot levels. The upper road will be used for haul truck access for ore only. Detailed pit designs have not yet been developed, but it is thought that all waste from the upper levels of the ore zone can be moved by bulldozer for access from the 7300 foot road level. The two roads can be placed along existing old roads, or along contours where little or no cut and fill work is needed, except to level the road as it traverses the hillsides.

Road construction will require an estimated 10 days of D-8 bulldozer work at \$100.00 per hour, and 5 days of road-grader work at \$40.00 per hour, for a total cost of about \$10,000.00.

Mining Methods. Larie Richardson, of Vinnel Corporation, has suggested that it would probably be a mistake to perform significant bulldozer operations within such a small pit. He feels that mixing of ore and waste is inevitable in that case, and suggests that nearly all waste be lightly blasted and hauled.

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Since the vein material is readily distinguishable, waste, in the wide upper third of the pit (above elevation 7360), could probably be ripped and pushed, but all the ore will be drilled and blasted. Ore is uniform enough to permit mining on 20 foot vertical benches, however, equipment selection may dictate a shorter bench height. Pit sideslopes, and the south end wall slope, will be 55 degrees, resulting in a 14 foot horizontal run for each 20 foot high bench to provide a wide cleanup road. Alternate benches, for instance, benches at levels 1240, 1200 and 1160 will be moved to within 5 feet of the wall, and the intermediate benches, 1220, 1180, etc., will be left as horizontal roadways. If benches shallower than 20 feet are used, the same approximate procedure will be employed, leaving 20 foot wide cleanup roads approximately every 40 vertical feet.

Three contractors have provided preliminary estimates of cost, with all costs generally in the $2.00 per ton range for either ore or waste, based on a haul of up to two miles for both ore and waste.

Approximately 20,000 tons of waste, in the area of the 7300 foot level pit access road, and the outer edges of the pit, will be ripped and bulldozed to the dump site. Costs for this waste should be less than $1.00 per ton. The bulk of the remainder of the waste will be truck-hauled 1000 feet to a waste dump located immediately adjacent to the 7300 foot level access road on the north pit edge. Cost to move this 166,000 tons of waste should result in a cost of $1.70 - $0.30 per ton lower than the longer haul on which the estimated bids were based.

Approximately 15 percent of the total ore tonnage exists as a rib of quartz sticking above the present ground surface. This rib is, in many places, a jumble of huge boulders. Mining this ore will require drilling and blasting of many boulders, or alternatively, "mud capping" them with shaped charges. This will add an estimated $2.00 per ton to this ore, or $0.30 per ton to overall ore costs.

The following cost estimates for mining and hauling of ore and waste are used in the feasibility.

Ore hauled to stockpile at heap leach (1 mile haul)	$2.30 per ton
Waste pushed or hauled to north end dump	$1.60 per ton
Waste hauled to a remote dump	$2.00 per ton

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Pad and Pad Preparation	\$ 0.45 per ton
Ponds	\$ 0.05 per ton
Fences and Roads	\$ 0.20 per ton
Chemicals, Lime and Cyanide	\$ 0.80 per ton
Carbon	\$ 0.25 per ton
Pipes and Miscellaneous Supplies	\$ 0.30 per ton
Labor, 3 Men @ \$2,500.00 per Man per Month, 4 Months duration	\$ 0.55 per ton
Power Generated On-Site or Purchased @ \$0.10 per KW Hour	\$ 0.20 per ton
Water pumped from Humboldt Mine workings via 3000 feet of 2-inch PVC pipeline, including electric pump and generator, Total - \$10,000.00 for equipment plus operating costs	\$ 0.20 per ton
Total Leaching Costs, including all services and supplies	\$ 3.00 per ton

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TOTAL OPERATING COST

Total operating cost to mine and treat the 56,000 tons of ore and 180,000 tons of waste are as follows:

Haul Roads	$ 10,000.00
Mining: Ore, 56,000 Tons @ $2.30 per ton;	129,000.00
Waste, at Mine Dump, 180,000 Tons @ $1.60 per ton;	288,000.00
Waste, Remote Dump, -0- Tons @ $2.00 per ton	-0-
Crushing and Stacking: 56,000 Tons @ $3.50 per ton;	196,000.00
Mobilization:	75,000.00
Leaching: 56,000 Tons @ $3.00 per ton;	168,000.00
Miscellaneous Supplies;	5,000.00
Professional Time and Project Management;	60,000.00
TOTAL	$ 931,000.00

CAPITAL COST

Since mining, crushing and stacking are contracted for, capital items for heap leach are included in operating costs, and a camp already exists at the property, there are very few actual capital items. Among the capital items necessary, and not considered elsewhere, are two pick up trucks, better communication facilities, additional office and safety equipment, and an on-site assay laboratory and the recovery plant.

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These items are all portable or re-saleable at the completion of the project. The items include:

On-site Assay Laboratory	$ 40,000.00
Recovery Plant	70,000.00
Pickup Trucks	20,000.00
Safety Equipment and Miscellaneous	20,000.00
Communications Facilities	10,000.00
Total Capital Items	$ 160,000.00

Since all expenses will be made essentially before production begins, the entire capital plus operating expenses are considered, for financial purposes, to be the capital requirement.

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PROFITABILITY

The following table presents a summary of the feasibility economics for mining ore blocks 1981 A and B on the East Humboldt Vein, Aurora, Nevada. Total project cost will be $1,091,000.00, with a production revenue of $1,432,600.00, leaving a $341,000.00 net profit before taxes.

CAPITAL COST

Capital Equipment Items	$ 160,000.00
Operating Expenses	931,000.00
	$ 1,091,000.00

PRODUCTION REVENUE (@ 65 Percent Recovery and $400.00 U.S. per ounce gold)

41,000 Tons @ .116 oz per ton	$ 1,237,000.00
15,000 Tons @ .05 oz per ton	195,000.00
Total	$ 1,432,000.00
Cash Flow through end of 1982	$ 109,000.00
Net Return over Cost: Cash Flow through Project Completion in August, 1983[1]	$ 341,000.00

NOTE: All costs and revenues in this report are shown in U.S. dollars at a gold price of U.S. $400.00 per ounce. All costs and revenues are shown before taxes.

(1) Additional cash flow would accrue from residual value of capital equipment.

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OPERATING CAPITAL REQUIREMENTS AND CASH FLOWS

The project will begin in March, 1982 with completion of mining and heap stacking by September of 1982. Leaching will commence in August, 1982 and it is anticipated that two months of 24-hours per day leaching, and two months of 8-hours per day leaching will be possible before winter temporarily halts the operation. Recoveries through the end of 1982 are expected to be a minimum of $1,200,000.00, with the remaining $31,000.00 being recovered in three months of leaching in early 1983.

SENSITIVITY AND STATISTICS

A statistical study was run to determine the validity of the ore grade determination and is included as Appendix B. Using the worst of 5 cases studied, a lognormal distribution with two high assay values thrown out and zero values excluded, the lower limit of ore grade at the 90 percent confidence level is 0.098 ounce gold per ton.

A sensitivity analysis is included at the end of Appendix B presenting several possible scenarios. One of these uses a gold price of $360.00 per ounce and the minimum ore grade from the statistical study. This analysis shows the venture to still have higher returns than costs.

An alternate sensitivity scenario analyzes the effect on costs if the same total ounces of gold are recovered as originally expected, at a price of $360.00 per ounce and if due to dilution, an additional 20 percent of "waste" is diverted to the heap to be crushed and leached. The venture shows positive returns here also.

SUMMARY

Since the feasibility study presented here shows a net, before tax, profit of $341,000.00; and since capital improvements, such as the leach plant, water system, improved communications, pickup trucks and operating expenses all represent valuable assets to a continued mining operation

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which is anticipated, the project appears to be justified. We recommend that negotiations for acquisition of the heap leach sites amd right-of-access for haul roads be begun immediately, and that funds be committed for finalizing the feasibility study and beginning the necessary permit acquisition. Construction of the carbon recovery circuit would be started in March, with the goal of starting preliminary production work on the property in late spring, as soon as weather and conditions are favorable.

Submitted by,

Michael W. Cassiday

Michael W. Cassiday
MILLER-KAPPES COMPANY

MWC/df

Attachments: Appendix A: Bucket Leach Tests on Aurora Samples
Appendix B: Statistical Analysis of Sample Data and Sensitivity Study

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APPENDIX A

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APPENDIX A

BUCKET LEACH TESTS ON AURORA SAMPLES

This report summarizes the results of a testing program on nine samples of ore taken from the Aurora, Nevada property.

Figure 1 presents sample descriptions and head assays. A summary of the bucket leach test results is presented in Figure 2. Of the nine samples used for bucket leach tests, one sample (1759B) contained only 0.010 ounces gold per ton, with the remaining seven samples averaging 0.161 ounces gold per ton.

Recoveries, overall, were good, with 64 percent of contained gold, on average, being recovered. Recoveries from tests on ore crushed to 2-inches and 1-1/2-inches (7 tests), averaged 59 percent of contained gold. Recoveries from tests on ore crushed to 5/8-inch (6 tests), averaged 72 percent of contained gold.

SAMPLE PREPARATION

Sample 1701 A, B and D. The as-received samples, each weighing approximately 20 Kg, were set up as bucket leach tests. These samples are discussed later in the Test Histories section.

Sample 1701 C. The as-received sample weighed approximately 10 Kg and was treated as follows:

1) Crush entire sample through jaw crusher to 5/8-inch.

2) Split out 3 Kg of 5/8-inch material, using a Jones splitter, and set up a leach test.

3) Crush remaining 5/8-inch material to 100 percent passing 6 mesh through a gyratory crusher, screening repeatedly.

4) Split out 3 Kg of the minus 6 mesh material and set up leach test.

5) Split out a 500 gram portion from the remaining minus 6 mesh material, pulverize, and send out for fire assay.

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FIGURE 1. SAMPLE DESCRIPTIONS

AND HEAD ASSAYS

MILLER-KAPPES SAMPLE NUMBER	LOCATION	DESCRIPTION	HEAD ASSAY oz Au/Ton
1701 A	Western edge stope, Humboldt East vein, bottom of gloryhole	Hand-picked rocks; a few fines, but mostly in 2-inch to 4-inch range	---
1701 B	Humboldt East vein, westernmost stope, upper East rib	Same description as 1701 A	.105
1701 C	Same as 1701 B	Same description as 1701 A	.105
1701 D	Humboldt East vein, central zone, vertical shaft dump	Taken off of surface of white dump surrounding vertical shaft. Some rocks up to 4-inches, but mostly 1/2-inch to 2-inches with some fines	---
1759 A	N28E and 460 feet from southern common corner Curry and East Humboldt claim	High grade vein matter at surface, 6 to 8 feet thickness of footwall at main vein	.199
1759 B	N55W from sample and 50 feet to drillhole PH #1	Modest grade quartz vein, surface sample collected from 3 to 9 feet of hanging wall side of vein	.011
1759 C	Stope at Kurt Guide Station, 1400 + 147 NW	Good grade quartz matter. Sample taken across 20 feet of footwall of vein, excluding immediate 5 feet of footwall quartz. Collected from 20 to 40 feet below surface.	.079
1759 D	Humboldt East dump. Old collapsed adit on northerly end and west side of main vein.	Oxidized material hand-picked off dump	.081
1759 E	Same as 1759 D	Pyritic/unoxidized ore, hand-picked off dump	.089
1902	East Humboldt adit	Collected off of the dump of "long drift" on the East side of vein. Sample from approximately 50 to 75 feet below the surface.	.412

FIGURE 2. AURORA BUCKET LEACH TESTS

HEAD ASSAYS AND RECOVERY STATISTICS

TEST NO.	SAMPLE NO.	SIZE (Inches)	DAYS LEACHING	oz Au/Ton RECOVERED	oz Au/Ton TAILS	PERCENT Au RECOVERED	HEAD ASSAY	CALC HEAD	CHEMICAL CONSUMPTION lbs/Short Ton NaCN	$Ca(OH)_2$	FINENESS RECOVERED METAL
1702	1701 A	1-1/2	112	.153	.213	41.80	----	.366	5.16	1.01	667
1703	1701 B	1-1/2	112	.065	.054	54.62	.105	.119	5.18	1.18	659
1704	1701 D	1-1/2	112	.034	.020	62.96	----	.054	5.55	1.45	478
1736	1701 C	- 6M	56	.086	.023	78.90	.105	.109	13.95	0.76	447
1737	1701 C	5/8	56	.085	.031	73.28	.105	.116	10.70	0.71	514
1769	1759 A	2	89	.101	.042	70.63	.199	.143	4.61	0.89	341
1770	1759 A	5/8	89	.137	.049	73.66	.199	.186	4.53	0.74	311
1771	1759 B	2	89	.008	.003	72.73	.010	.011	6.61	1.00	609
1772	1759 B	5/8	89	.007	.002	77.78	.010	.009	5.35	1.58	521
1773	1759 C	2	89	.056	.042	57.14	.079	.098	3.46	0.79	643
1774	1759 C	5/8	89	.062	.023	72.94	.079	.085	3.07	1.49	624
1795	1759 D	5/8	90	.057	.026	68.67	.081	.083	5.78	0.68	602
1959	1902	2	62	.199	.196	50.38	.412	.395	3.92	0.80	431
1960	1902	5/8	62	.270	.158	63.08	.412	.428	3.41	0.76	615

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Samples 1759 A - C were treated as follows:

1) Crush entire sample to 2-inches through jaw crusher.

2) Split out a 5 gallon bucket of 2-inch material and set up bucket leach test.

3) Crush remaining 2-inch material to 5/8-inch through jaw crusher. Split out a 5 gallon bucket of 5/8-inch material and set up bucket leach test.

4) Take remaining 5/8-inch material and split out a 5 Kg sample. Crush to 100 percent passing 6 mesh through a gyratory crusher, screening repeatedly.

5) Split out two 500 gram portions from the minus 6 mesh material, pulverize, and submit for fire assay.

6) Run 24 hour cyanide bottle roll tests on pulverized and minus 6 mesh material.

7) Run 1 hour and 24 hour cyanide centrifuge tests on pulverized samples.

Samples 1759 D and E were treated as follows:

1) Crush entire sample to 5/8-inch through jaw crusher.

2) Split out 5 Kg of 5/8-inch material and crush to 100 percent passing 6 mesh through a gyratory crusher, screening repeatedly.

3) Split out two 500 gram portions from the minus 6 mesh material, pulverize, run centrifuge tests and send out for fire assay.

4) Take remaining 5/8-inch material from 1759 D only, split out a 5 gallon bucket, and set up a bucket leach test.

5) Run 24 hour cyanide bottle roll tests on pulverized and minus 6 mesh material from 1759 D.

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Sample 1902 was treated and prepared the same as samples 1759 A - C, however, only one hour centrifuge tests were run on the pulverized head sample. No bottle roll tests were conducted.

CYANIDE BOTTLE ROLL TESTS

Cyanide bottle roll tests were conducted on pulverized and minus 6 mesh head samples from samples 1759 A - D, according to the following procedure:

1) Weigh out 100 grams of ore and place into 250 ml polybottle.
2) Add 150 mls of water and adjust pH to 10 using lime. Add 0.75 grams NaCN (equivalent to 5 gpl NaCN).
3) Place on rolls for 24 hours.
4) Filter; dry tailings and save.
5) Check solution for pH, Au, Ag and Cu.

Figure 3 shows the percent recovery of contained gold in a 24 hour cyanide bottle roll test (based on calculated head assays), and the product fineness (ratio of gold to gold plus silver; times 1000). On the average, 93 percent of contained gold was recovered in the tests on pulverized samples. Tests on unpulverized samples averaged 63 percent of contained gold recovered.

CENTRIFUGE TESTS ON HEAD SAMPLES

The pulverized pulps from head samples were subjected to cyanide centrifuge tests according to the following procedures:

1) Weigh out 10 grams of pulverized ore and place in centrifuge tube.
2) Add 25 mls of 5 gpl NaCN solution. Adjust pH, if necessary, to pH 10, using lime.

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FIGURE 3. 24-HR CYANIDE BOTTLE ROLL TESTS
ON PULVERIZED AND MINUS 6 MESH SAMPLES

SAMPLE NO.	TEST NO.	SIZE	oz Au/Ton RECOVERED	PERCENT RECOVERY	FINENESS RECOVERED METAL
1759 A	1793 A	P	.184	92.5	130
1759 A	1793 E	-6M	.110	55.3	120
1759 B	1793 B	P	.009	81.8	317
1759 B	1793 F	-6M	.008	72.7	432
1759 C	1793 C	P	.082	103.8	487
1759 C	1793 G	-6M	.053	67.1	550
1759 D	1793 D	P	.077	95.1	454
1759 D	1793 H	-6M	.044	54.3	469

FIGURE 4. AGITATED CYANIDE CENTRIFUGE TESTS
ON PULVERIZED PORTIONS OF SAMPLE

SAMPLE NO.	TEST NO.	LEACH TIME HOURS	oz Au/Ton RECOVERED	PERCENT RECOVERY	FINENESS RECOVERED METAL
1759 A	1776 A	1	.131	65.8	109
1759 A	1776 F	24	.201	101.0	136
1759 B	1776 B	1	.013	118.2	310
1759 B	1776 G	24	.009	81.8	235
1759 C	1776 C	1	.086	108.9	484
1759 C	1776 H	24	.089	112.7	473
1759 D	1776 D	1	.074	91.4	440
1759 D	1776 I	24	.080	98.8	445
1759 E	1776 E	1	.050	56.2	358
1759 E	1776 J	24	.049	55.6	354
1902	1915 B	1	.274	66.5	303

3) Place on wrist-action shaker for specified time.

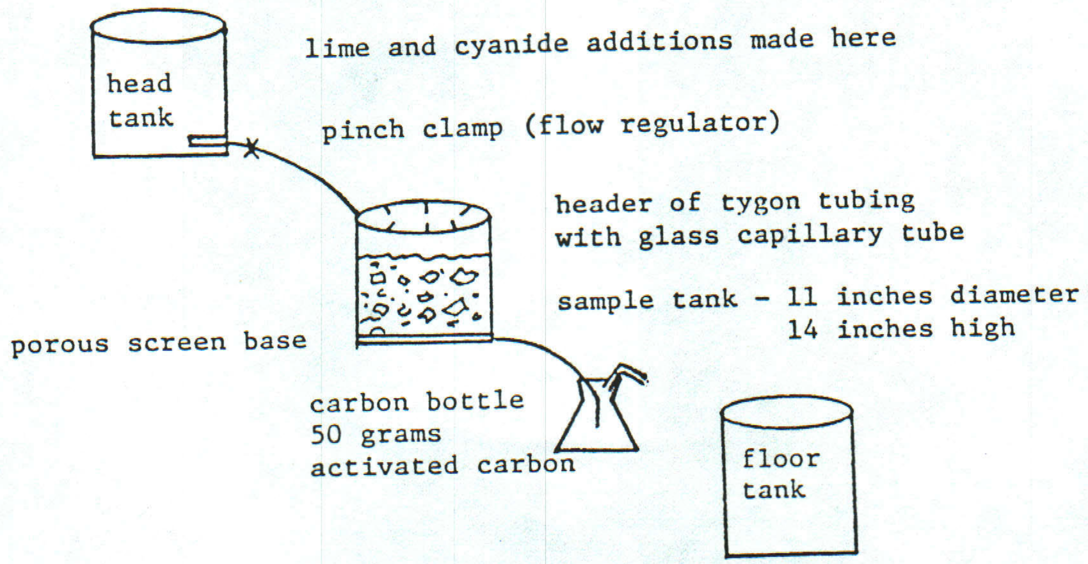
4) Centrifuge and filter through glass wool.

5) Check solution for pH, Au, Ag and discard residue.

Figure 4 shows the percent recovery of contained gold in cyanide centrifuge tests on pulverized material, and the product fineness (ratio of gold to gold plus silver; times 1000).

BUCKET LEACH TEST APPARATUS

The apparatus for the 2, 1-1/2 and 5/8-inch rock tests is shown in the drawing below. The apparatus for tests 1736 and 1737 differed slightly, in that the sample tank was 4 -inches in diameter and 18-inches high.



LEACH TEST PROCEDURE

In the apparatus shown on the preceding page, the center tank, or leach tank, was filled with the rock to be leached.

Alkaline cyanide solution was continuously distributed onto the rock from the head tank, through a set of glass capillary drip tubes. Flowrate of solution dripping onto the rock was controlled using a pinch clamp.

Solutions entering the floor tank were assayed every two cycles for cyanide and lime and reagents were added, as necessary, to maintain solutions at "target levels".

Solutions exiting the leach tank flowed continuously through a bottle of activated carbon and then into a floor tank. The active solution in the system was recycled to the head tank every 48 to 72 hours.

The tanks were kept covered at all times to minimize evaporation and cyanide loss. No makeup water was required.

The charge of activated carbon was removed three times during the tests, with the exception of tests 1702, 1703 and 1704, which had five carbon changes, and assayed to determine the amount of gold and silver leached from the ore.

TEST HISTORIES

Start-up of Tests. The initial leach solution for all tests contained 1.0 grams NaCN per liter and 0.5 grams $Ca(OH)_2$ per liter. Initial solutions exiting all tests were alkaline and contained measureable amounts of cyanide.

Solution Color and Clarity. Solutions remained clear and slightly brown throughout the tests.

Cyanide Strength and Alkalinity. Cyanide strength was allowed to decline slowly for the first 14 days of the tests to a minimum of 0.2 to 0.5 grams NaCN per liter. It was then maintained in the range 0.4 to 0.6 grams per liter for the remainder of the tests.

Alkalinity was generally maintained, with lime, in the range pH 10.0 to 10.6 for the duration of all tests. Lime and cyanide consumption data are tabulated in Figure 2.

Tests 1702, 1703 and 1704 were started, using as-received ore, with a maximum dimension of 4-inches. Initial gold recovery was poor with 10 percent of contained gold (17.5 percent of recoverable gold) being recovered onto carbon by day 14. The tests were stopped on day 15, the ore allowed to air dry, then crushed to 1-1/2-inches and the tests restarted. Analysis of the leach solutions by atomic absorption showed that after 7 days leaching at 1-1/2-inches, an additional 25 percent of contained gold (46 percent of recoverable gold) had been leached, on average.

GOLD AND SILVER RECOVERIES

Figure 2 tabulates gold recoveries and fineness of recovered metal (ratio of gold to gold plus silver; times 1000). Figures 5 and 6 show gold recovery based on carbon assays, versus days leaching.

Average recovery for all tests was 64 percent of contained gold. Recoveries from tests on ore crushed to 2-inches and 1-1/2-inches (7 tests), averaged 59 percent of contained gold. Recoveries from tests on ore crushed to 5/8-inch (6 tests), averaged 72 percent of contained gold. The average fineness of the beads obtained from fire assay of the activated carbon was 540.

TAILINGS ASSAY AND METALLURGICAL BALANCES

At the end of the tests, the test tailings were dried and then screened into various size fractions. The size fractions were crushed, if necessary, to 100 percent passing 6 mesh and then two portions were split out and pulverized. Fire assays were run on each of the pulverized portions. Tailings assays and weights are reported in Figure 7.

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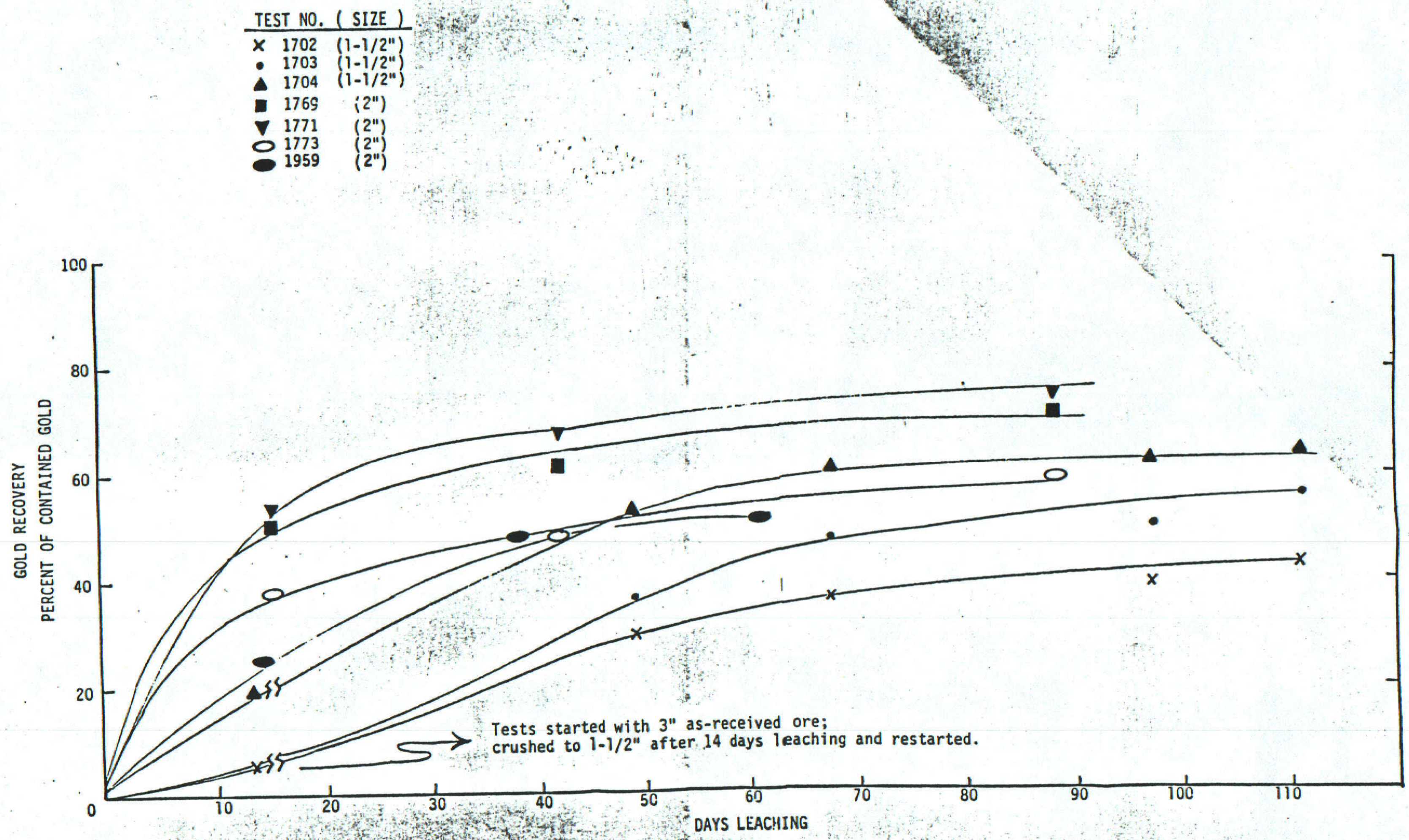

FIGURE 5. GOLD RECOVERY VERSUS TIME
AURORA SAMPLES CRUSHED TO 2" AND 1-1/2"

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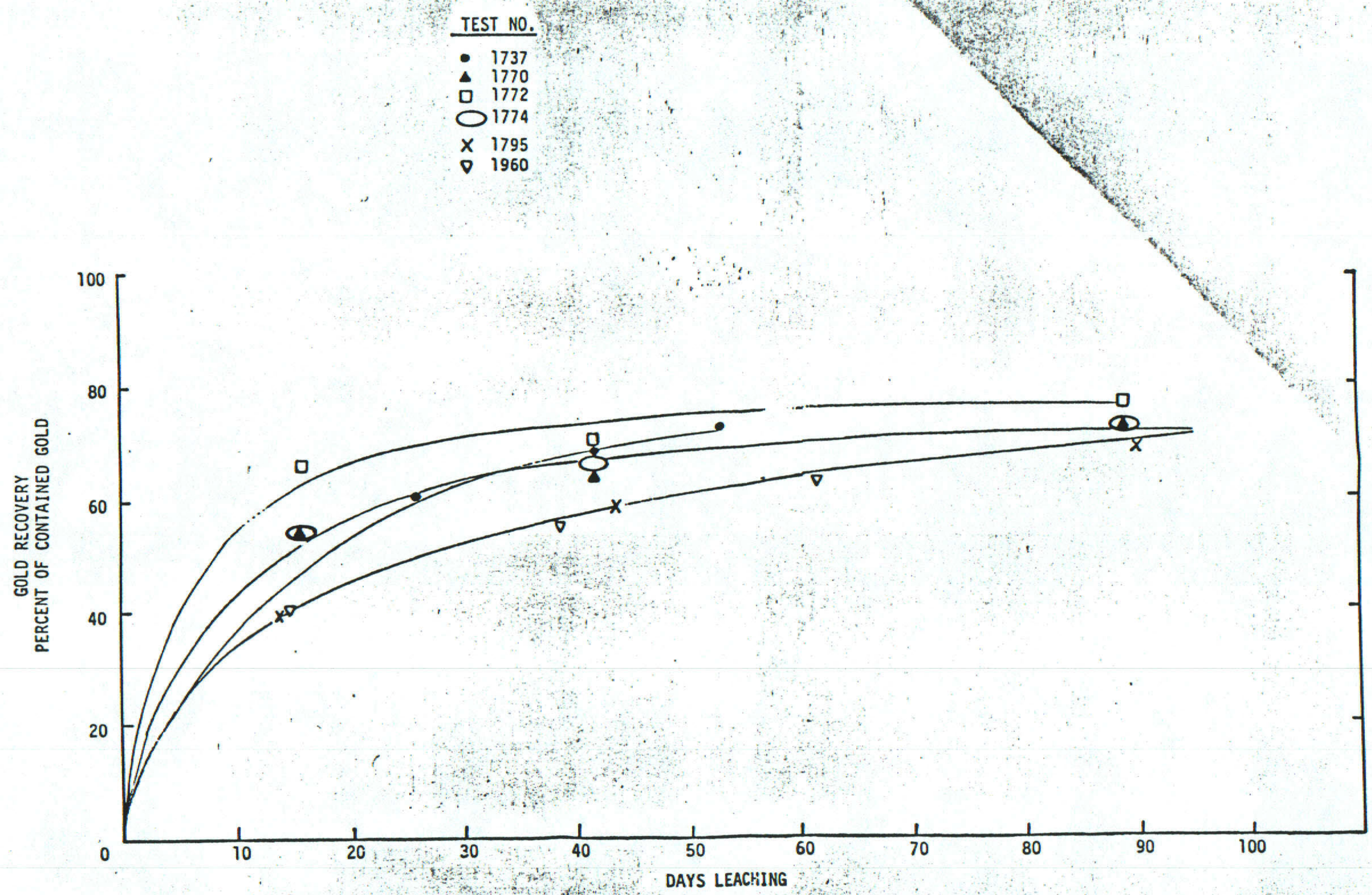

FIGURE 6. GOLD RECOVERY VERSUS TIME
AURORA SAMPLES CRUSHED TO 5/8-INCHES

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FIGURE 7. AURORA BUCKET LEACH TESTS

TAILINGS WEIGHTS AND ASSAYS [1]

TEST NO.	CALC.HEAD ASSAY oz Au/Ton	SIZE FRACTION + 1"	-1" + 1/2"	-1/2" + 3M	-3M + 10M	-10M +65M	-65M	TOTAL
1702	.366	13,780 .260/.224	2,820 .184/.168	1,210 .128/.120	1,660 .110/.100	---	---	19,470 .213
1703	.119	7,450 .054/.040	5,180 .076/.068	1,900 .056/.046	1,360 .038/.039	580 .040/.036	300 .078/.006	16,770 .054
1704	.054	11,460 .025/.025	4,760 .016/.013	2,080 .014/.017	2,010 .017/.016	970 .008/.013	440 .026/.014	21,720 .020
1736	.109	---	---	---	---	---	---	3,150 .023
1737[2]	.116	---	---	---	---	---	---	3,360 .031
1769	.143	12,180 .042/.054	4,640 .038/.045	1,920 .035/.036	1,840 .031/.033	1,200 .019/.024	630 .023/.024	22,410 .042
1771	.011	12,340 .004/.004	3,990 .002/.002	1,480 .002/.003	1,250 .002/.001	870 .002/.003	---	19,930 .003
1772	.009	---	2,680 .000/.007	9,250 .000/.004	5,050 .000/.002	2,060 .000/.001	880 .000/.007	19,920 .002
1773	.098	11,420 .048/.042	4,970 .042/.046	1,260 .030/.025	1,420 .026/.024	1,090 .034/.043	---	20,160 .042
1774	.085	---	2,120 .032/.023	9,390 .026/.032	5,730 .018/.015	2,060 .014/.011	740 .016/.014	20,040 .023
1795	.083	---	4,040 .050/.036	11,240 .028/.024	5,270 .018/.019	1,860 .018/.010	1,080 .012/.010	23,490 .026
1959	.395	13,720 .232/.206	3,610 .180/.134	1,350 .156/.137	1,190 .106/.108	---	---	19,870 .196
1960	.428	---	12,370 .190/.176	4,430 .148/.125	2,800 .138/.113	1,060 .086/.078	450 .090/.083	21,110 .158

1- Weight in grams; duplicate fire assays (ounces Au per short ton).

2- Entire tailings crushed to 100 percent passing 6 mesh.

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Metallurgical balances, or comparisons between original head assays and calculated head assays (gold recovered onto carbon + gold remaining in the tailings) were good for all tests, averaging 90.3 percent. Eight of the 12 tests showed higher calculated heads than assay heads, four showed the reverse; three showed calculated heads within .005 ounces per ton of the assay heads. The calculated head, based on actual recoveries and assays of the fine-crushed tailings, is considered more accurate than the assay head.

ASSAYING PROCEDURES

Heads and Tailings Assays. Heads and tailings assays were all run in duplicate, half as one assay ton fire assays and half as half assay ton fire assays.

Carbon Assays. The loaded activated carbon was dried and weighed. Two samples were split out and assayed and the remainder saved for reference. The carbon for assays was roasted to convert it to ash, then conventionally fire assayed.

Solution Assays. Approximate solution assays were made periodically on an atomic absorption spectrophotometer, using as a standard, a gold cyanide solution which had been calibrated by fire assay. The solution assays were used merely to check on the progress of the leach, since actual recovery was based on fire assay of the activated carbon.

Final solution was checked by AA methods and found to contain negligible amounts of gold.

Submitted by,

Russell B. Dix

RBD/df

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APPENDIX B

STATISTICAL ANALYSIS OF SAMPLE DATA AND SENSITIVITY STUDY

In an attempt to arrive at a more realistic mean value of ore grade based on assay results within the ore zone defined by 1981 rotary drilling, Prof. Pierre Mousset-Jones of the Mackay School of Mines, Reno, Nevada, was asked to recalculate the mean values under several cases, using methods developed by H.S. Sichel (1966) for lognormally distributed gold assays. Using the School of Mines computer and software, M. Klemme, a mining engineering graduate student, developed the data presented in Figure 1.

Case III B, the lognormal mean determined after excluding the two highest assays and the two zero assays, probably represents the best mean value of the cases considered, as far as this method is able to yield an accurate estimate of the mean.

Professor Mousset-Jones suggested that due to probable assay value dependence (i.e., samples are not statistically speaking, truly independent), the indicated confidence limits are slightly optimistic. In otherwords, at the 90 percent confidence level, the upper limit may actually be a somewhat higher value and the lower limit may actually be somewhat lower than the values indicated.

It should be pointed out that although Sichel's method is less sophisticated than modern geostatistical methods, it probably yields a somewhat more realistic mean value of the ore zone than simple arithmetical averaging.

A complete description of the techniques for this study are in "An Introduction to Geostatical Methods of Mineral Evaluation" by J.M. Rendu, Chapter 2, a copy of which is at the end of this report.

STATISTICAL ANALYSIS

A total of 142 samples (103 rotary drillhole and 39 rock chip samples) taken from the designated ore zone (Block A) in 1981, were used in the study. The different cases examined were as follows:

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CASE NUMBER	CONDITIONS
I A	Normal Distribution. All samples included (N = 142).
II A	Normal Distribution. All samples excluding zero values (N = 140).
II B	Lognormal Distribution. All samples excluding zero values (N = 140).
III A	Normal Distribution. All samples excluding two highest assays and zero values (N = 138).
III B	Lognormal Distribution. All samples excluding two highest assays and zero values (N = 138).

The results of these analyses are presented in Figure 1. Figures 2 and 3 list the samples used for the study along with their fire assays. Figure 4 presents the cumulative frequency of the samples. A class size of 0.030 ounces per ton gold was used for the entire study.

Case III B, the worst possible case, indicates that there is a 90 percent chance that the average ore grade lies between the upper limit of 0.1480 ounces per ton gold and the lower limit of 0.0980 ounces per ton, with a 5 percent chance it may lie below 0.0980 ounces per ton.

SENSITIVITY ANALYSIS

Figure 5 presents a sensitivity analysis of the feasibility study to variations of grade, percent recovery and gold prices. Using the minimum ore grade obtained from the statistical analysis (Case III B) and a gold price of $360.00 per ounce, the venture will still have higher returns than costs.

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An alternative sensitivity scenario analysis is also presented on the effects on costs if the same total ounces of gold are recovered as originally expected, at a price of $360.00 per ounce, if due to dilution, an additional 20 percent by weight of "waste" rock (11,000 tons) is diverted to the heaps to be crushed and leached. The venture shows a positive return here also.

Submitted by,

Russell B. Dix

Russell B. Dix

RBD/df

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DRILLHOLE NUMBER	INTERVAL (Feet)	FIRE ASSAY ounces/ton Au
PH-15		0.119
		0.017
PH-39	45- 50	0.046
	50- 55	0.036
	55- 60	0.080
	60- 65	0.086
	65- 70	0.086
	70- 75	0.057
	75- 80	0.074
	80- 85	0.053
	85- 90	0.560
PH-40	20- 25	0.049
	25- 30	0.028
	30- 35	0.049
	35- 40	0.024
	40- 45	0.260
	45- 50	0.051
	50- 55	0.024
PH-43	45- 50	0.041
	50- 55	0.061
	55- 60	0.070
	60- 65	0.054
PH-44	5- 10	0.162
	10- 15	0.126
	15- 20	0.040
	20- 25	0.040
	25- 30	0.030
	30- 35	0.030
PH-45	40- 45	0.081
	45- 50	0.112
	50- 55	0.012
	55- 60	0.033
	60- 65	0.029
	65- 70	0.288
	70- 75	0.053
	75- 80	0.073
	80- 85	0.031
	85- 90	0.085
	90- 95	0.100
	95-100	0.206
PH-47	10- 15	0.063
	15- 20	0.057
	20- 25	0.015
	25- 30	0.026
	30- 35	0.038
	35- 40	0.048
	40- 45	0.109
	45- 50	0.015
PH-48	5- 10	0.032
	10- 15	0.025
	15- 20	0.135
	20- 25	0.031
	25- 30	0.032

DRILLHOLE NUMBER	INTERVAL (Feet)	FIRE ASSAY ounces/ton Au
PH-49	15- 20	0.042
	20- 25	0.014
	25- 30	0.014
PH-51	40- 45	0.096
	45- 50	0.380
	50- 55	0.272
	55- 60	0.164
	60- 65	0.064
	65- 70	0.122
	70- 75	0.234
	75- 80	0.140
	80- 85	0.312
	85- 90	0.100
	90- 95	0.086
	95-100	0.240
	100-105	0.142
	105-110	0.191
	110-115	0.408
	115-120	0.902
PH-52	5- 10	0.077
	10- 15	0.098
	15- 20	0.081
	20- 25	0.017
	25- 30	0.163
	30- 35	0.317
	35- 40	0.133
	40- 45	0.064
	45- 50	0.126
	50- 55	0.038
PH-53	50- 55	0.060
	55- 60	0.048
	60- 65	0.174
	65- 70	0.010
	70- 75	0.012
	75- 80	0.451
	80- 85	0.145
PH-60	45- 50	0.520
	50- 55	0.044
	55- 60	0.156
	60- 65	0.400
DDH-2	65- 70	0.124
	70- 78	0.258
	78- 82	0.038
	82- 87	0.122
	117-122	0.435
	122-127	0.040
	127-131	0.028
	131-136	0.368
	136-142	0.366

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FIGURE 3

OUTCROP CHIP SAMPLES USED FOR STATISTICAL ANALYSIS

ORE BLOCK A SECTION	LINE	FIRE ASSAY Au oz/ton
1	Northern	.062
		.095
		.132
		.029
1	Center	.038
		.545
		.286
		.167
		.063
1	Southern	.029
		.037
		.049
		.148
		.075
		.057
		.022
		.013
		.008
		.003
2	Adit	.000
		.000
		.155
		.097
3	Northern	.108
		.401
		.088
3	Center	.180
3	Southern	.119
		.032
		.180
		.053
		.020
4	Northern	.061
		.159
4	Southern	.057
		.072
		.695
		.142
		.047

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FIGURE 4

CUMULATIVE FREQUENCY

DISTRIBUTION OF SAMPLE VALUES

(Class Interval = 0.03)

CELL NUMBER	CLASS UPPER VALUE	NUMBER OF SAMPLES	CUMULATIVE SAMPLES	CUMULATIVE FREQUENCY (%)
1	0.030	10	10	7.0
2	0.060	36	46	32.4
3	0.090	29	75	52.8
4	0.120	15	90	63.4
5	0.150	12	102	71.8
6	0.180	12	114	80.3
7	0.210	5	119	83.8
8	0.240	1	120	84.5
9	0.270	2	122	85.9
10	0.300	3	125	88.0
11	0.330	3	128	90.1
12	0.360	1	129	90.8
13	0.390	2	131	92.2
14	0.420	3	134	94.4
15	0.450	1	135	95.1
16	0.480	2	137	96.5
17	0.510	0	137	96.5
18	0.540	1	138	97.2
19	0.570	1	139	97.9
20	0.600	1	140	98.6
21-23	0.690	0	140	98.6
24	0.720	1	141	99.3
25-30	0.900	0	141	99.3
31	0.930	1	142	100.0

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FIGURE 5

SENSITIVITY OF AURORA FEASIBILITY STUDY TO
VARIATIONS OF GRADE, PERCENT RECOVERY AND GOLD PRICES

At .098 oz/ton Au (Case III b Lower 90% Confidence Limit) for Ore Block A and .045 oz/ton Au for Ore Block B at 65% Recovery

.098 oz/ton x 41,000 tons = 4018 oz
.045 oz/ton x 15,000 tons = 675 oz

TOTAL 4693 oz

4693 oz x .65 = 3050 oz recoverable gold

3050 oz @ $400.00 =	$1,220,180.00	@ $360.00 = $1,098,000.00
	- 1,091,000.00	- 1,091,000.00
PROFIT	$ 129,180.00	$ 7,000.00

At .116 oz/ton Au @ $360.00 Gold and 65% Recovery

.116 oz/ton x 41,000 tons = 4756 oz   5506 oz x .65 = 3579 oz recovered
.05 oz/ton x 15,000 tons = 750 oz

TOTAL 5506 oz

3579 oz x $360.00 = $1,288,440.00 $ 1,288,440.00
- 1,091,000.00

PROFIT $ 197,440.00

At .116 oz/ton Au @ $360.00 Gold, 65% Recovery and an additional 20% of Waste diverted to the heaps

.116 oz/ton x 41,000 tons = 4756 oz   5506 oz x .65 = 3579 oz recovered
.050 oz/ton x 15,000 tons = 750 oz
.000 oz/ton x 11,000 tons = 0 oz

TOTAL 5506 oz

3579 oz x $400.00 =	$1,431,600.00	@ $360.00 = $1,288,440.00
	- 1,162,500.00	- 1,162,500.00
PROFIT	$ 269,100.00	$ 125,940.00

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At 70% Recovery vs/65% Recovery, Same Grade

41,000 tons @ .116 oz/ton =	4756 oz	5506 oz x .70 = 3854 oz recovered	
15,000 tons @ .05 oz/ton =	750 oz		
TOTAL	5506 oz		
3854 oz x $400.00 =	$1,541,680.00	3854 oz @ $360.00 =	$1,387,440.00
	- 1,091,000.00		- 1,091,000.00
PROFIT	$ 450,680.00		$ 296,440.00

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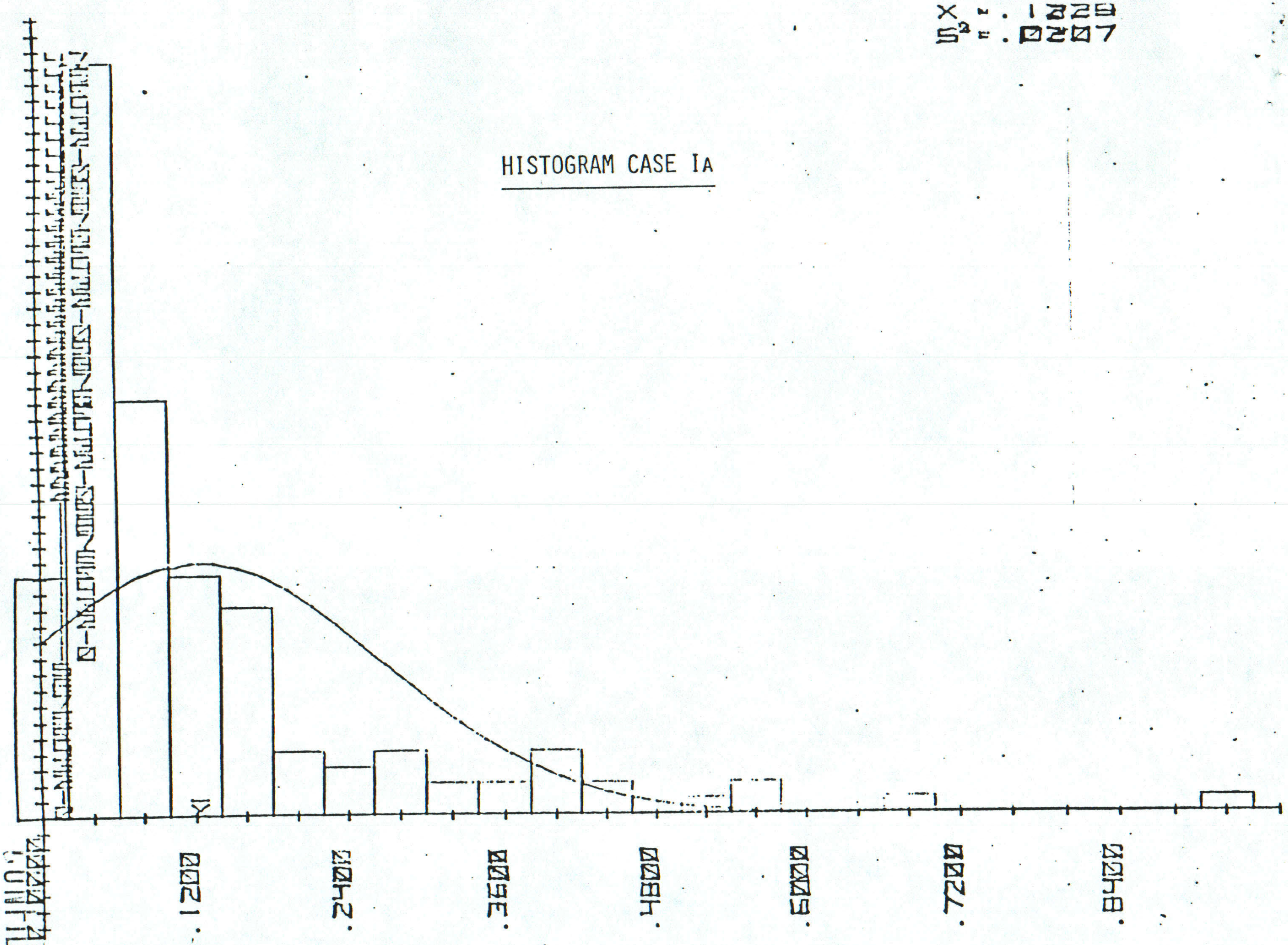
Ia

$\bar{X} = .1229$
$S^2 = .0207$

HISTOGRAM CASE IA



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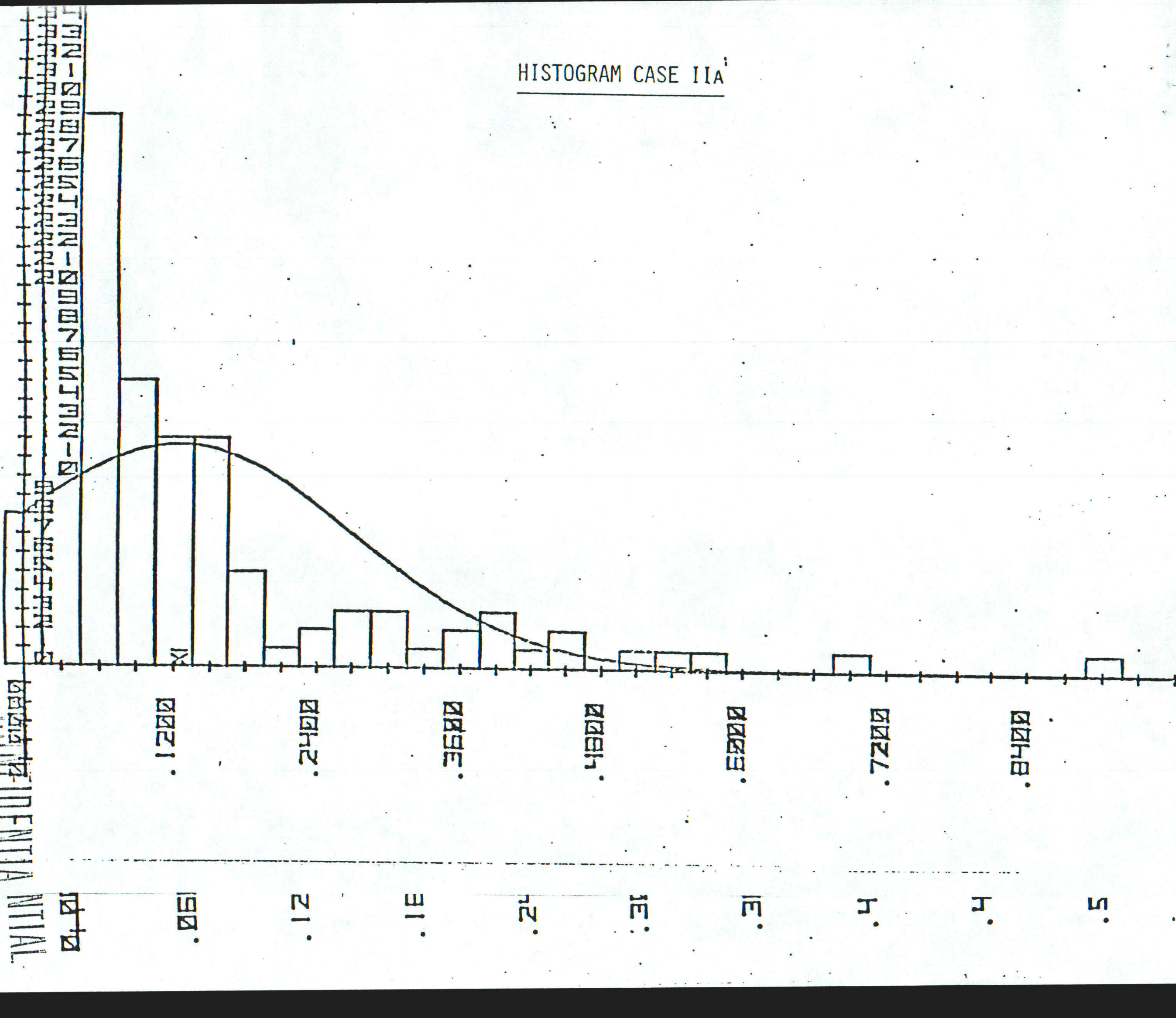
Ia

IIa

HISTOGRAM CASE IIA



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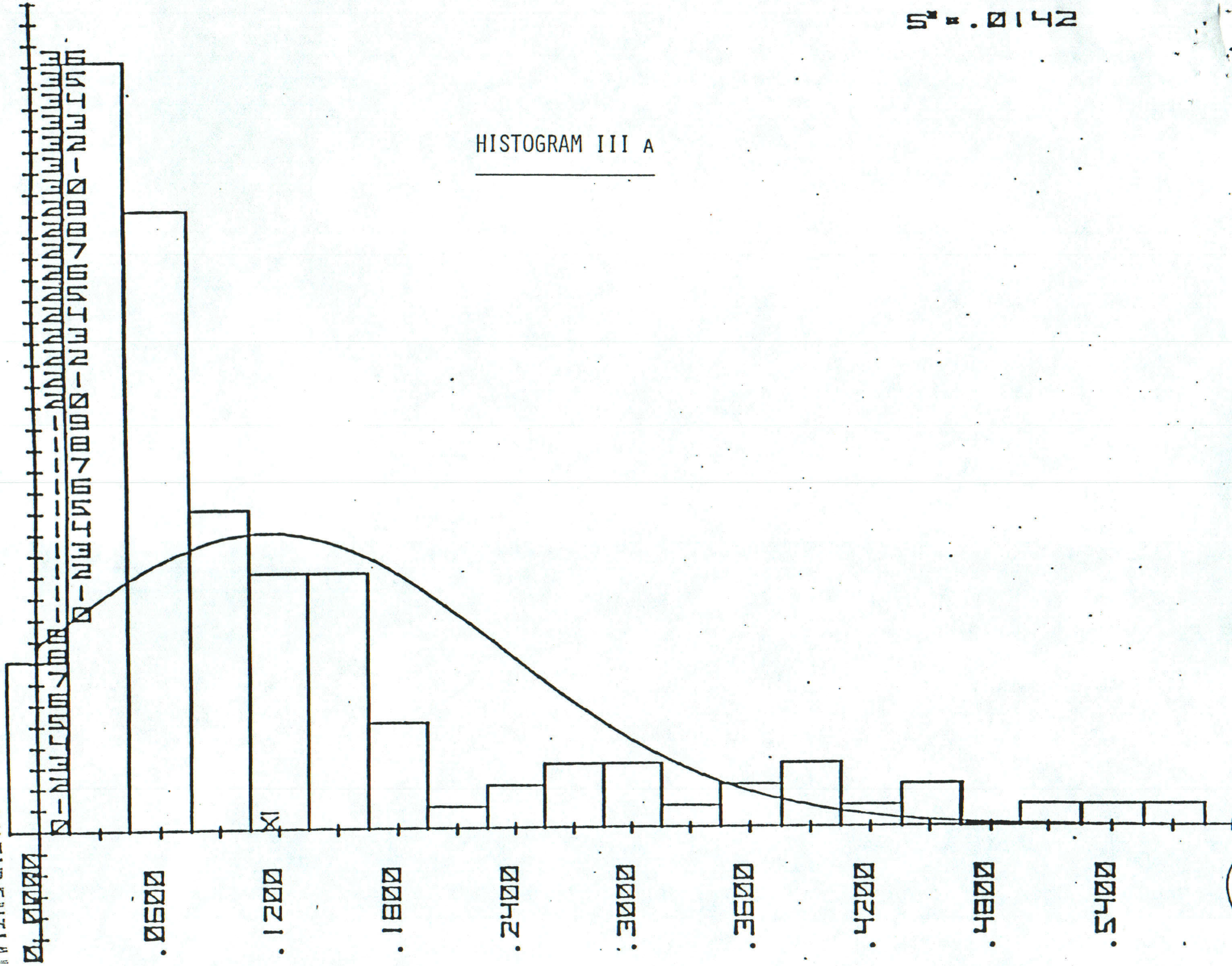
$S^2 = .0142$

HISTOGRAM III A



III a

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LOG-NORMAL FWT CASE (III)