

Report filed with Mining Engineering Division

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MINES
NOVA SCOTIA

YAVA MINES LIMITED

435264

SUBMISSION
OF
MINE-MILL PROPOSAL

TO THE

PROVINCE OF NOVA SCOTIA
DEPARTMENT OF MINES

FOR THE

UNDERGROUND DEVELOPMENT OF, AND PRODUCTION FROM
THE SILVERMINES LEAD PROPERTY
CAPE BRETON COUNTY
NOVA SCOTIA

Date of Submission
September 30th, 1977.

Suite 1500
2 St. Clair Avenue West
Toronto, Ontario, Canada.
M4V 1L5

11F/16B 27-C-83(21)

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I

INTRODUCTION

INTRODUCTION:

This proposal by Yava Mines Limited to commence an underground development program to be followed by room and pillar stope mining on its Silvermines property at Salmon River, and the concentration of lead ores at its existing concentrating facility eight miles distant at Lake Enon, is submitted in confidence to the Department of Mines, Province of Nova Scotia, as required under the Mineral Resources Act Part III.

There has been completed on the Silvermines property 104,000 feet of diamond drilling over a period of 15 years, 1962 - 1977, and several feasibility studies at different tonnages and grades of lead ore but at no time has any underground development, overburden stripping or removal of part of the deposit taken place, with the exception of a small surface sample of 50 tons of mineralized rock taken in 1964 for mill tests.

In 1975 the control of Yava Mines Limited was purchased by Barymin Explorations Limited, and that Company's subsequent review of the diamond drill information indicated the potential viability of an initial 500 ton per day mining operation in the West Zone of the deposit. When in 1977 the opportunity arose to purchase, on favourable terms, an existing concentrator in the close vicinity of the deposit, it was decided to undertake a close patterned diamond drill program over the proposed area to be mined to confirm the grade and uniformity indicated from the previous wider spaced drilling. When results from the drilling were confirmatory, a decision to purchase the concentrator with a view to bringing the property to production was made.

The final drilling results have adequately confirmed the tonnage, grade and continuity of the ore in the area proposed to be mined and financing plans to place the property into production

at a rate of 500 tons of ore per day have reached an advanced stage. These plans contemplate equity financing in an amount of \$1,500,000 to undertake an underground development program, followed by bank debt financing of approximately a further \$2,000,000 to complete equipment purchases and provide working capital.

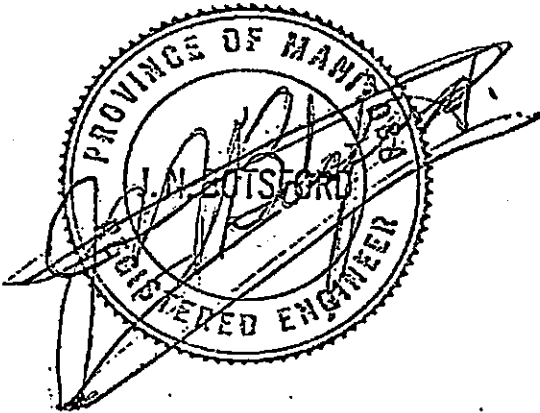
The underground development program is intended to determine the following:

1. Secure an accurate report on rock conditions as regards rock strengths, rock hardness, drilling speeds, ground water conditions and pumping requirements, road conditions for trackless haulage, etc.
2. Determine ore grade correlation with diamond drill results.
3. Undertake modification of the concentrator to handle lead ores from the deposit and make plant test runs to determine the most efficient treatment procedures.

The results of this program will indicate the proper mining and metallurgical procedures to follow and provide accurate information on operating costs and on the final equipment required to place the property on a regular production basis.

From the information available at this time there are sufficient ore reserves indicated for a 500 ton per day mining operation over a period of about seven years within the zone proposed to be mined. There is also indicated possible additional reserves elsewhere in the deposit to provide either many additional years production, or, depending on economic circumstances, an increased rate of production over an adequate life of mine period. The

preliminary mining operation combined with further exploration in the future will provide the best opportunity of realizing the optimum potential of the deposit as a whole.



J. N. Botsford
J. N. Botsford, B.A.Sc., P.Eng.
Engineering Consultant

M. F. Nicholson
M. F. Nicholson
President and Director

II

CORPORATE INFORMATION

CORPORATE INFORMATION:

Yava Mines Limited was incorporated in 1964 under the Companies Act of Canada. It is registered under the Domestic, Dominion and Foreign Corporations Act of Nova Scotia. The address of its Agent in Nova Scotia is:-

Mr. Frank M. Covert,
Bank of Canada Building,
1583 Hollis Street,
Halifax, Nova Scotia.

The address of its Head Office is:-

Suite 1500,
2 St. Clair Avenue West,
Toronto, Ontario. M4V 1L5.

Telephone: (416) 923-6558

Plant: Enon, Nova Scotia
(902) 587-2156

Yava Mines Limited is a wholly owned subsidiary of Barymin Explorations Limited. Barymin originally held an 8.70% interest in Yava Mines through ownership of Gunnex Limited and eventually purchased controlling interest of 57.82% from Phelps Dodge Corporation of Canada, and 33.48% from Talisman Mines Limited.

The Officers and Directors of the Company are:-

M. F. Nicholson	-	Toronto	-	President & Director
R. C. J. Edwards	-	Toronto	-	Vice President & Director
F. A. Schulman	-	Toronto	-	Director
J. N. Botsford	-	Toronto	-	Director
N. B. Ivory	-	Montreal	-	Director

Yava Mines Limited has 100% ownership of the Silvermines Property which consists of 116 mineral claims which cover the lead sulphide deposit.

The Parent Company, Barymin Explorations Limited is an Ontario Corporation, incorporated in 1956, with its Head Office at:-

Suite 1500,
2 St. Clair Avenue West,
Toronto, Ontario. M4V 1L5.

Telephone: (416) 923-6318

The Officers and Directors of the Company are:-

F. A. Schulman	-	Toronto	-	Chairman & Director
R. C. J. Edwards	-	Toronto	-	President & Director
M. F. Nicholson	-	Toronto	-	Secretary-Treasurer & Director
J. N. Botsford	-	Toronto	-	Director
R. H. C. Holman	-	Dublin, Ireland	-	Director
N. B. Ivory	-	Montreal	-	Director
D. H. Stodard	-	Toronto	-	Director

The shares of the Company are listed on the Toronto Stock Exchange.

Capitalization is authorized at 8,000,000 shares of no par value with 5,090,344 shares issued.

Transfer Agents:-

Canada Permanent Trust Company,
20 Eglinton Avenue West,
Toronto, Ontario. M4R 2E2.

Engineering Consulting Services:-

Mr. J. N. Botsford, B.A.Sc., M.E., P.Eng.
is providing full consulting services
to Barymin and Yava Mines in connection
with the development and operation of
the Silvermines lead deposit.

III

GENERAL INFORMATION

- (a) Property Description
- (b) Mine Property History
- (c) Development to Date

GENERAL INFORMATION:(a) Property Description(i) Mine Area

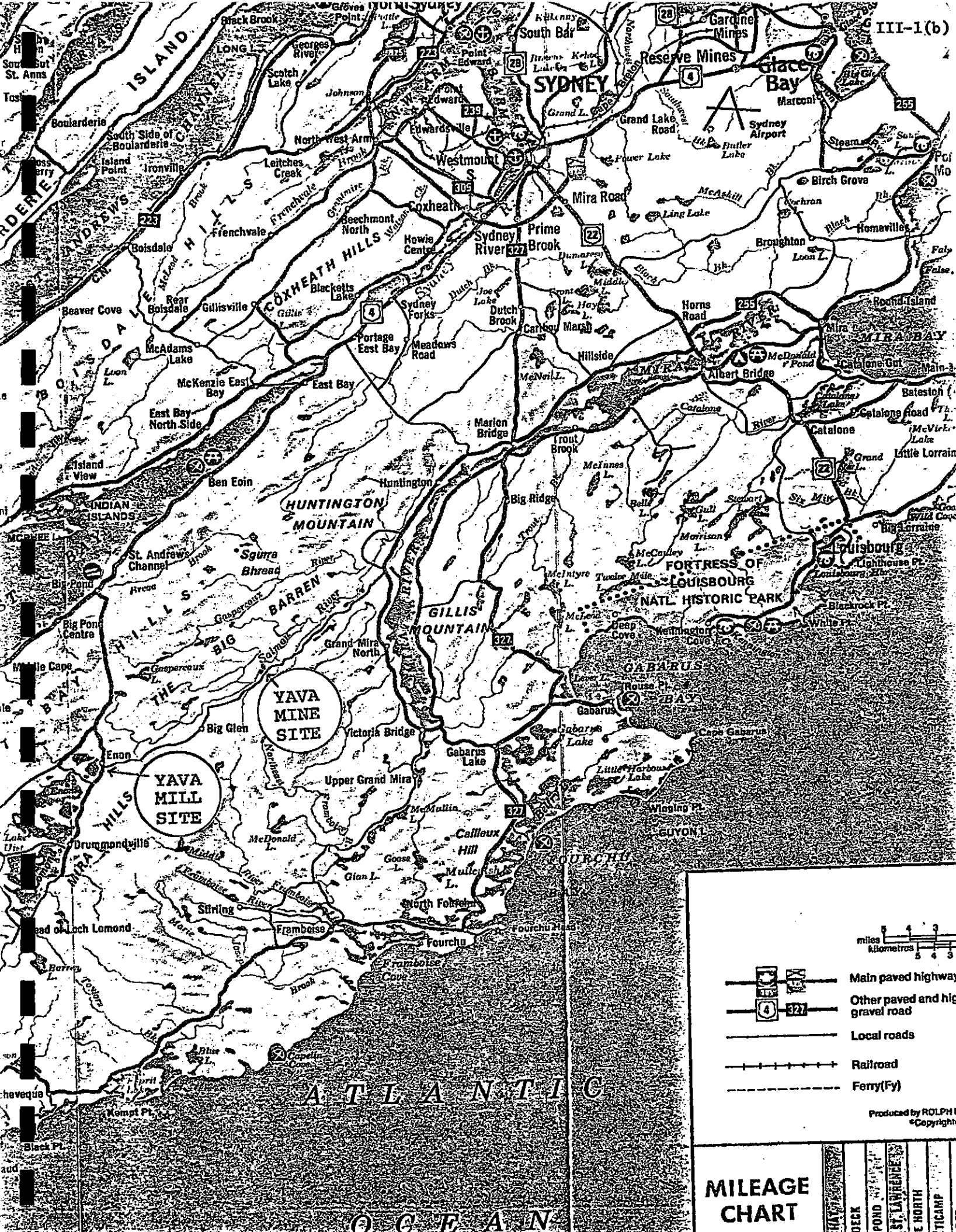
The Silvermines Property of Yava Mines Limited consists of 116 mineral claims located in Cape Breton County, Nova Scotia, approximately 27 miles southwest of Sydney. Sydney is a city of 50,000 population, a major seaport on the Atlantic seaboard, the location of the Dominion Steel and Coal Company plant and the centre of the Cape Breton coal industry. The city is served also by rail and air transportation facilities.

The geographical location of the property at Big Glen, in the Salmon River Basin area is:-

Longitude $45^{\circ} 51'36''$ North

Latitude $60^{\circ} 24'18''$ West

A paved road extends from Sydney for a distance of 17 miles via Highway 327 to Marion Bridge, and then via the Salmon River road, a gravel road, continues for another 10 miles to Silvermines.



Main paved highway
 Other paved and high gravel road
 Local roads
 Railroad
 Ferry (Fy)

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MILEAGE CHART

SYDNEY	1
DECK	1
POND	1
ST. LAWRENCE	1
E. NORTH	1
TICAMP	1

ATLANTIC

OCEAN

(ii) Plant Area

In 1977 Yava Mines Limited purchased a concentrator from Kaiser Celestite Mining Limited at Enon, Loch Lomond, a distance of eight miles on the Salmon River Road beyond Silvermines when approaching from Marion Bridge.

Until the end of 1976 the plant was used to concentrate strontium ore mined in its immediate vicinity and concentrates were trucked by road to the Kaiser Strontium processing plant at Point Edward, Sydney, a distance of approximately 35 miles.

The plant consists of conventional crushing, grinding and flotation equipment which can be used for the concentration of lead ore from the Silvermines deposit. Previous mined out quarries close to the concentrator, into which tailings from the previous operation were piped, provide ample capacity for continued plant operation over many years.

A gravel road extends from the concentrator a distance of about nine miles to No. 4 Highway at Big Pond situated about 20 miles south west of Sydney.

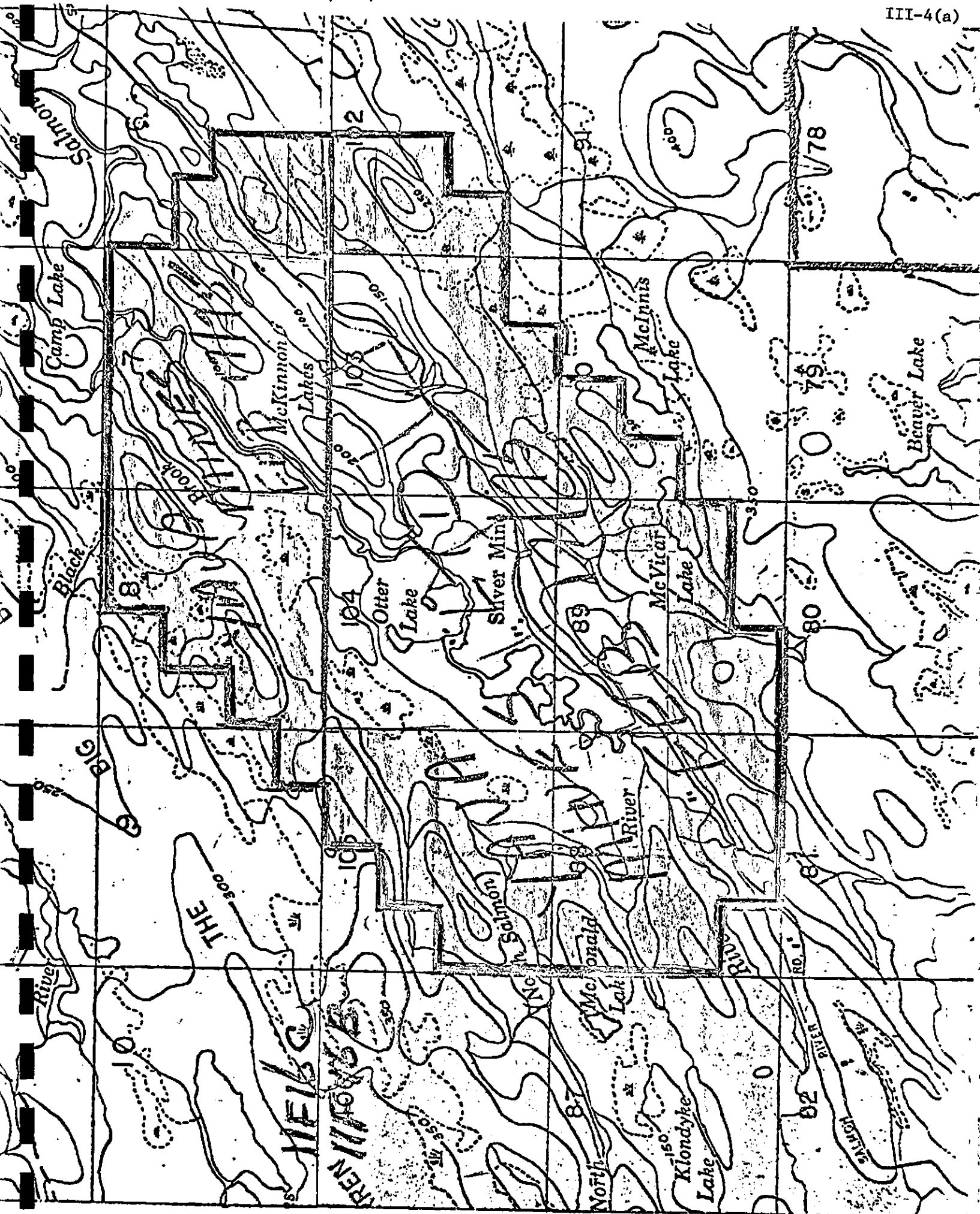
(iii) Mineral Rights

The mining area consists of the following mineral claims:-

<u>Development Licence</u>	<u>Claims</u>	<u>Tract</u>	<u>Reference Map</u>
0018 (70 claims)	ABCE FGHL MNO	88	11F16B
	CDEF JKLT Q	105	11F16B
	A	9	11F16C
	ABCD EFGH JKLT Q	8	11F16C
	JKLM NOPQ	7	11F16C
	CDEF M	6	11F16C
	ELMN O	102	11F16B
	BCDE FGH	103	11F16B
	MNO	90	11F16B
	CDEF GHJQ	89	11F16B

<u>Development Licence</u>	<u>Claims</u>	<u>Tract</u>	<u>Reference Map</u>
0019 (46 claims)	JKPQ	88	11F16B
	ABGH	105	11F16B
	ABCD EFGH JKLM NOPQ	104	11F16B
	KLMN OP	89	11F16B
	JKLM NOPQ	103	11F16B
	ABCD EFGH	7	11F16C

All of the above claims are in good standing,
with annual renewal fees of \$40/claim having
been paid January 14, 1977.



(iv) Surface RightsBig Glen DistrictReference Map C.B. 25 Crown Land Forestry Series,Scale 1: 15,840.

(aa) Crown Lands on which mineral exploration rights granted to Yava Mines Limited by Department of Lands & Forests and on which a mining lease is now sought from the Crown.

(1) Ungranted Lot between Lots 18230 & 15470	200 Acres
(2) Lot 16440	100 Acres
(3) Lot 16439	95 Acres
(4) Crown Land between Lots 16070 & 15470	<u>100 Acres</u>
	<u>495 Acres</u>

(bb) Surface Right Ownership by Yava Mines Limited on which mining lease is now sought from the Crown.

(1) Lot 15470 (Murdoch Morrison)	100 Acres
(2) Lot 16437 (Archibald Morrison)	5.25 Acres
(3) Lot(D. Morrison deceased) between Crown Land lot (aa) (1) above and Norman MacDonald Certificate No. 6	<u>200 Acres</u>
	<u>305.25 Acres</u>

All the above constitute land areas held in connection with the proposed mining operations of Yava Mines Limited.

In addition, Yava Mines holds the following surface rights for purposes of continuing exploration.

(cc) Crown Lands on which exploration rights
granted by Department of Lands & Forests.

(1) Triangle south of 5212	30 Acres
(2) Grant 19561	260 Acres
(3) Petition 12668	100 Acres
(4) Petition 13628	100 Acres
(5) Petition 17294	112 Acres
(6) Petition north of Grant 19657	38 Acres
(7) Ungranted land east of 19657 and west of Certificate 88 Ronald MacKinnon	<u>290 Acres</u>
	<u>930 Acres</u>

(dd) Surface Right Ownership by Yava
Mines Limited.

(1) Grant 16070 Rev. D. Sutherland	50 Acres
(2) Grant 19709 Malcolm McDonald	100 Acres
(3) Lot between Grants 16070 and 19709	<u>100 Acres</u>
	<u>250 Acres</u>

(iv) Surface Rights (Cont'd)

Enon, Loch Lomond District
Reference Map C.B.

(ee) Surface Right Ownership by Yava Mines
Limited incorporating the Concentrator
and tailings pond areas.

(1) Tract No. 209 (Neil McVicar)	200 Acres
(2) Tract 15468 (E. T. Mosely)	46 Acres
(3) Tract 406 (D. R. MacDonald)	<u>100 Acres</u>
	<u>346 Acres</u>

GENERAL INFORMATION: (Cont'd)(b) Mine Property History

The property was acquired by staking in June, 1961, by Talisman Mines Limited. There followed an exploration programme consisting of geochemical and geophysical surveys followed by diamond drilling in 1962 which consisted of 28 holes for a total depth of 9,637 feet.

In mid-1962 Phelps Dodge Corporation of Canada purchased a major interest in the property and continued to extend all phases of the exploration programme, through a wholly owned subsidiary, Winslow Mines Limited. During the period September, 1962 through 1969, there was in excess of \$750,000 expended on geochemical, geophysical, diamond drilling and other forms of exploration. The diamond drilling portion recorded the completion of 219 surface holes for some 81,000 feet of drilling. Also a small surface bulk sample of ore, metallurgical tests and feasibility.

In 1975 Barymin Explorations Limited acquired the controlling interest in Yava Mines Limited by purchasing 997,500 shares from Winslow Mines Limited. Subsequently Barymin purchased the outstanding balance of shares from Talisman Mines Limited and secured 100% ownership of Yava Mines Limited and the property.

(b) Mine Property History (Cont'd)

Additional close spaced surface diamond drilling of 69 holes for some 14,000 feet was completed by Yava in mid-1977 to facilitate the planning of an underground development and production programme.

GENERAL INFORMATION: (Contd')(c) Development to Date

Since early 1962, the contact between the Pennsylvanian sandstone and the Windsor shales has been intensively tested along an 8 mile strike length. The programme has employed geological mapping, geochemistry, geophysics and diamond drilling.

To date, 316 holes have been diamond drilled for a total of 104,658 feet. The holes have explored a strike length of 38,000 feet with a maximum distance of 2,400 feet, between holes. More closely spaced holes were drilled in a rectangular area measuring 12,500 feet along strike by 3,500 feet in the direction of dip.

In the area of more closely spaced drilling, the three zones of higher grade mineralization were drilled at 250 feet intervals on sections 300 feet apart. Between these zones and down dip from them, holes were usually spaced 1,000 - 1,200 feet apart.

During 1969, in an area 500 feet square in the centre of the eastern zone, holes were drilled at 100 foot intervals to prove continuity of a unit higher grade than that of the zone as a whole. The detailed drilling established this continuity.

Likewise, in 1977 an area 1,100 feet down dip by 1,000 along strike was drilled in the west zone at

(c) Development to Date (Cont'd)

100 foot intervals. The results of this diamond drilling supported revised thinking as to the origin of the deposit and likewise proved the continuity of the deposit in this higher grade area. Expansion of the area drilled was limited in a down dip direction by the Salmon River and along strike to the northeast by completion of programme requirements at this time. Drilling results indicated the deposit continued in both directions and was open for further development.

The following is a record of surface diamond drill work completed on the Silvermines property.

<u>Diamond Drilling Completed by or Under Direction of</u>	<u>Time Period</u>	<u>No. of D.D.H.</u>	<u>D.D.H. No's.</u>	<u>Footage</u>
Talisman	Early 1962	28	1-28	9,637
Phelps Dodge (Winslow)	Sept./62- Oct./63	190	29-218	69,412
	1964 & 1969	29	219-247	11,649
Yava	May 11 - July 21/77	69	248-316	13,960
TOTAL	1962 - 1977 (15 years)	316	1-316	104,658

IV

RESOURCES

- (a) Geology
- (b) Mineralogy
- (c) Ore Reserves
- (d) Potential Ore Reserves

RESOURCES:(a) Geology

An extensive mantle of drift, varying in depth from 15 to 75 feet, obscures most of the property. Outcrops are few in number, usually small and widely scattered.

The claims are underlain by Precambrian and Palaeozoic formations, occupying a basin structure elongated along a northeast - southwest axis. The dip of the formation is relatively flat at a more or less uniform 15 degrees to the northwest.

Lead sulphides in the form of galena occur in the bottom 80 feet of a medium to fine grained Pennsylvanian sandstone which was deposited in the Salmon River basin. In the vicinity of the lead deposit the basement rocks consist of Devonian felsite porphyry (rhyolite) overlain by shales and limestones of the Windsor Group. (Mississippian), which are in turn overlaid by the Pennsylvanian sandstones, at the base of which the lead deposit is located.

The mineralized area is divided laterally into 3 zones, known as the East, Central and West Zones, by two basement ridges which trend north. The 3 zones cover a strike length of 11,500 feet. The mineralization occupies the valleys between the ridges with higher grade lead values extending the surface down dip for a length of about 2,500 feet, which at this point is approximately 800 feet below surface.

(a) Geology (Cont'd)

In the West Zone, the ore horizon follows the sandstone-shale contact with the base of the ore horizon not over 10 feet above the shale.

There are some structural irregularities in the sandstone-shale contact; but for the most part the ore in this zone is regular.

In the Central Zone, a sandstone-shale contact forms a ridge. The ore horizon is at the base of the sandstone on top of the ridge but does not follow the sandstone-shale contact into the flanking basin, remaining about 60 - 70 feet above the contact.

In the East Zone, the sandstone-shale contact forms a basin or trough. The ore is at the base of the sandstone toward the edges of the basin; but is above the base in the middle of the trough.

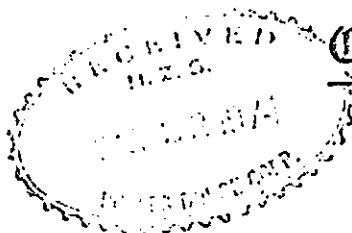
From the drill core, the Pennsylvanian sandstone appears to be a reasonably competent rock; and satisfactory for mining purposes.

RESOURCES: (Cont'd)(b) Mineralogy

Lakefield Research of Canada Limited undertook a programme of bench concentration tests on samples of split core from diamond drilling the Yava property in compliance with a letter of request dated March 25, 1974 from Phelps Dodge Corporation of Canada Limited. The objectives of these tests were to produce a marketable lead concentrate and to determine by analysis the variety of elements in the concentrate on which penalties might be levied by smelters. A bulk lead concentrate was produced in a 3-stage locked cycle test and the concentrates from each cycle were combined to produce a lead concentrate assaying 73% Pb. with a recovery of 90.1% Pb.

The lead concentrate analysis showing elements present follows:-

R.
H.
L.
W.
.....
.....
.....



Certificate of Analysis

Date: May 17, 1974

Received: March 26, 1974

From: Phelps Dodge Corporation of Canada Ltd., Our Reference No. L.R. 1705
Suite 1106-55 Yonge Street,
Toronto, Ontario M5E 1J4.

Invoice No. _____

Samples submitted to us show results as follows:

Pb Concentrate - Test No. 3

<u>Element</u>	<u>Percent</u>
Lead (Pb)	72.85
Zinc (Zn)	2.32
Iron (Fe)	2.58
Nickel (Ni)	0.002
Copper (Cu)	0.013
Bismuth (Bi)	0.002
Carbon present as carbonate (CO ₂)	2.04
Carbon not present as carbonate	1.37
Arsenic (As)	0.008
Antimony (Sb)	0.003
Sulphur (S)	15.3
Silica (SiO ₂)	2.75
Alumina (Al ₂ O ₃)	0.42
Lime (CaO)	0.32
Magnesia (MgO)	0.12
Barium (BaO)	0.004
Silver (Ag)	0.86
	<u>100.007</u>

* ounces per ton

SIGNED _____

MANAGER

A.G. Scobie, P. Eng.,

Mr. D.A. Firth (2)

(b) Mineralogy (Cont'd)

The major sulphide mineral present is galena. Any other metals present are in very minor quantities these being zine and silver.

A semi-quantitative spectrographic analysis was conducted on the diamond drill core and the concentrates with results as follows.

Semi-quantitative Spectrographic Analysis

Element	Head	Conc.	Element	Head	Conc.
Aluminum (Al ₂ O ₃)	H	M	Manganese	.1%	.02%
Antimony	-	-	Magnesium (MgO)	.5%	.1 %
Arsenic	-	-	Molybdenum	-	-
Barium	.03%	.01%	Neodymium (Nd ₂ O ₃)	-	-
Beryllium (BeO)	-	-	Nickel	.001%	.001%
Bismuth	-	.001%	Phosphorus	-	-
Boron	-	-	Silver	.01 oz/t	1 oz/t
Calcium (CaO)	1%	.1%	Silicon (SiO ₂)	H	H
Cadmium	-	.01%	Sodium (Na ₂ O)	.1%	-
Cerium (CeO ₂)	-	-	Strontium	-	-
Chromium	.01%	-	Tantalum (Ta ₂ O ₅)	-	-
Cobalt	-	-	Thorium (ThO ₂)	-	-
Columbium (Cb ₂ O ₅)	-	-	Tin	-	-
Copper	.005%	.01%	Titanium	.2%	.01%
Gallium	-	-	Tungsten	-	-
Germanium	-	-	Uranium (U ₃ O ₈)	-	-
Iron (Fe)	1%	1%	Vanadium	.01%	-
Lanthanum (La ₂ O ₃)	-	-	Yttrium (Y ₂ O ₃)	-	-
Lead	M	H	Zinc	.3%	M
Lithium (Li ₂ O)	-	-	Zirconium (ZrO ₂)	.01%	-

Figures are approximate:

CODE: H - High -10 - 100% approx.
M - Medium - 1 - 10% approx.
L - Low -.1 - 1% approx.

- Not detected - Elements looked for but not found.
X Not looked for
< Less than

(b) Mineralogy (Cont'd)

The Atlantic Industrial Research Institute at Nova Scotia Technical College, Halifax, conducted a series of metallurgical tests under the direction of Mr. M. A.K. Grice, on samples of Silvermines ore from the Yava property supplied from the close spaced diamond drilling programme conducted from May through July, 1977 on the West Zone of the deposit. During the course of these tests a semi-quantitative spectrographic analysis on the lead concentrate (77.6% Pb.) was made, the results of which are recorded on the following page. These results corroborate those secured by Lakefield Research of Canada Limited in 1974.



ATLANTIC INDUSTRIAL RESEARCH INSTITUTE

NOVA SCOTIA TECHNICAL COLLEGE
HALIFAX, NOVA SCOTIA
TELEPHONE 428-8300

Laboratory for the Investigation of Minerals
Atlantic Industrial Research Institute
Nova Scotia Technical College
Halifax, N.S. B3J 2X4

August 31, 1977

Semi-quantitative Spectrographic Analysis

Sample: Lead Conc (77.6% Pb) from floatation tests for Barymin Explorations Limited.

<u>Element</u>	<u>%</u>	<u>Element</u>	<u>%</u>
Al	.2	Mg	.02
As	<.01	Ni	.001
Ba	.01	Ag	.006
Be	<.001	Si	.55
Ca	.03	Sr	<.001
Cd	<.01	Sn	.001
Cr	.001	Ti	.04
Co	.001	U	<.01
Cu	.007	V	<.001
Ga	<.001	Yb	<.001
Fe	1.0	Zn	1.2
Mn	.005		

< less than

Elements not showing were not detected (not present or below sensitivity of instrument)

No analysis for Li, Eu, K, Pd because they are used in buffer.

B. D. Cole

RESOURCES: (Cont'd)(c) Ore Reserves

In October 1974, at the time Phelps Dodge made its latest study on undertaking an open pit mining operation on the Silvermines deposit, the ore reserve position was calculated as follows:-

<u>Zones</u>	<u>Open Pit</u>		<u>Underground Indicated</u>		<u>Underground Possible</u>	
	<u>Tons</u>	<u>% Pb</u>	<u>Tons</u>	<u>% Pb</u>	<u>Tons</u>	<u>% Pb</u>
East	4,625,247	2.665%	2,629,478	3.50%	2,990,078	3.15%
Central	7,424,065	2.158%	2,679,559	3.56%	3,354,019	3.19%
West	3,888,316	2.497%	1,112,658	3.81%	1,580,141	3.75%
Total (Av.)	15,937,628	2.387%	6,421,695	3.58%	7,924,238	3.28%

These figures show that in the West Zone, open pit and drill indicated underground reserves total 5,000,974 tons grading 2.79% Pb. In view of the location of Salmon River over the West Zone and the effect of local flooding, it is impractical to consider open pit mining of this dimension in this zone.

In a review of ore reserves made for Yava Mines Ltd. by D. W. Esson, P.Eng., in October 1976, West Zone reserves, possible and indicated, were calculated at 1,830,000 grading 4.70%, of which drill indicated reserves before dilution were estimated at 871,000 tons grading 5.21% Pb. Mr. Esson included a dilution factor to these reserves and calculated diluted indicated reserves of 1,045,000 tons of 4.69%.

?
Using a 30%
cut off?
20% @ 2.09%

The ore reserve calculations made from the 100' drill programme completed this summer on part of the West Zone

(c) Ore Reserves (Cont'd)

indicate 671,576 tons of 5.08% Pb using 3% cut-off. The area drilled was restricted to part of the up dip of the West Zone south of the Salmon River, which acted as a boundary to drilling on the north. Bearing in mind that the area drilled at close spacing was aimed at confirming the zone of higher grade mineralization indicated previously, and was not intended to drill out all potential ore reserves, there is fair correlation between Mr. Esson's figures and the results of the confirmatory drilling.

The mining proposed in the West Zone will not affect future mining of the Central and East Zones which are respectively 900 feet and 2,000 feet apart and constitute separate deposits for extraction purposes. The proportion of mineralization extracted by mining in the West Zone will be a function of costs of mining and the price of lead. The ore reserves calculated by Mr. Esson represent approximately 62% of the mineralization contained in the open pit and drill indicated reserves calculated by Phelps Dodge. As the mining method proposed will be extended to assay walls, material left beyond the limits of mining will remain available for extraction under conditions of higher lead price so long as underground access is maintained.

Mr. David Otterman, geologist for Barymin Explorations Limited, has filed his report and calculations on the recent diamond drilling on the West Zone, which are reproduced on the following pages. Reference is also made to Maps X - 1 and X - 2.

YAVA MINES LIMITEDWEST ZONEORE RESERVE CALCULATIONSAUGUST, 1977

The present calculations of ore reserves are based on the diamond drilling carried out during the months of May, June and July, 1977. Drilling was confined to the south half of the WEST ZONE and followed a grid pattern of 100 feet by 100 feet. Previously drilled holes were used only to help fill in the grid. The calculations employ the following procedures and assumptions;

- 1) The calculations assume that all intersections are continuous from hole to hole. Distances down dip have been calculated giving each drill hole an area of influence of 10,352.762 sq. ¹⁰⁰ _{cos 15°} x 100 feet (i.e. 100 feet x 103.52762 ft.) except in certain marginal areas where one-half the area is considered to be more appropriate.
- 2) All mineralized widths have been corrected to true widths. An average dip of 15° has been calculated for the footwall. A minimum true width of 12' has been applied except in a few marginal areas where a minimum 8' width was used.
- 3) The ore is assumed to have a specific gravity of 13 cu. ft. per ton. Tonnage values have not been rounded off and should be considered to be accurate only to the nearest 1000 tons.
- 4) Dilution is assumed to be nil.

- 2 -

- 5) Calculations have been made for assay cut-off values of 3.00% Pb, 3.50% Pb and 4.00% Pb and for a true width of 12.0' from the Pennsylvanian/Windsor Contact.
- 6) In areas where weakly mineralized limestone pebble conglomerate lies at the base of the mineralized sandstone, the top of this unit is considered to be the footwall except for the 12.0' true width calculation.
- 7) The results of the four sets of calculations have been reported on a map showing the area of influence of each drill hole and the tonnage and grade exhibited by each hole. A heavy line has been drawn around the area designated by the 3.00% Pb cut-off value.
- 8) The total tonnages and average grade (weighted) of each line were calculated. From these the total tonnage and grade were calculated for the whole zone (see table).

WDO/akv
September 9/77

David W. Otterman

Sample Calculation

Hole # 273

Sample No.	From	to	Width	% Pb.	True Width	Ft. %	Dist. Above Cntct	% Pb Cumul. Weighted Avg.
3948	197.1	201.9	4.8	7.00	4.6	32.20	4.6	7.00
3949	194.2	197.1	2.9	0.27	2.8	0.76	7.4	4.47
3950	189.2	194.2	5.0	5.82(r)	4.8 ✓	27.94	12.2	4.99
3951	181.8	189.2	7.4	3.71(r)	7.1 ✓	26.34 ✓	19.3	4.52
5566	175.2	181.8	6.6	2.16	5.8	12.53	25.1	3.97
5567	169.2	175.2	6.0	1.91	5.8	11.08	30.9	3.59

as on X₁

At cut off at 3.00% Pb.	<u>I</u>	19.3'	4.52% Pb
3.50% Pb	<u>II</u>	19.3'	4.52% Pb
4.00% Pb	<u>III</u>	12.2'	4.99% Pb
12.0'	<u>IV</u>	12.0'	4.98% Pb

$$\text{e.g. } \frac{7.00 \times 4.6' + 0.27 \times 2.8' + 5.82 \times 4.8' + 3.71 \times 7.1'}{4.6' + 2.8' + 4.8' + 7.1'} = \frac{87.24}{19.3} = 4.52\%$$

at 3% or 3.5% cut off.

$$\text{For 12.0' width - } \frac{7.00 \times 4.6' + 0.27 \times 2.8' + 5.82 \times 4.6'}{12.0'} = \frac{59.73}{12.0'} = 4.98\%$$

Tonnage Calculation:

$$\text{e.g. 3.00% Cut off - } \frac{19.3' \times 10,352.762'}{13} = 15,369.9 \text{ tons}$$

$$\text{e.g. 12.0' - } \frac{12.0' \times 10,352.762'}{13} = 9,556.4 \text{ Tons}$$

YAVA MINES LIMITED

WEST ZONE (PARTIAL)

Calculation of Tonnage & Grade at 3.00%, 3.50% & 4.00% Cut-Off

<u>Line</u>	<u>I</u>		<u>II</u>		<u>III</u>		<u>IV</u>	
	<u>3.00%</u>		<u>3.50%</u>		<u>4.00%</u>		<u>12'</u>	
	<u>%</u>	<u>Tons</u>	<u>%</u>	<u>Tons</u>	<u>%</u>	<u>Tons</u>	<u>%</u>	<u>Tons</u>
34W	4.16	20,028.8	4.33	16,883.0	5.37	4,778.2	4.95	9,556.4
33W	5.13	89,073.5	5.45	72,827.6	5.50	70,597.8	5.51	66,894.8
32W	5.88	130,365.3	6.44	108,306.0	6.67	99,625.6	7.30	62,116.6
31W	4.65	123,277.5	4.81	114,198.9	4.85	110,377.3	4.65	86,007.6
30W	4.64	71,991.6	4.68	70,080.3	5.05	52,361.2	4.46	62,116.6
29W	4.62	36,672.7	4.71	35,518.0	4.71	35,518.0	4.09	28,669.2
28W	4.78	45,353.1	4.90	41,530.6	4.98	38,265.4	4.36	33,447.4
27W	5.46	50,728.6	5.46	50,728.6	5.59	43,083.5	5.31	38,225.6
26W	6.21	50,489.7	6.21	50,489.7	6.93	44,835.4	7.16	33,447.4
25W	5.23	53,595.4	5.58	44,357.6	6.15	35,478.0	5.85	33,447.4
Weighted Average	5.17%	671,576.2	5.38%	604,920.3	5.62%	534,920.4	5.39%	453,929.0
Arith. Avge.	(5.08%)		(5.26%)		(5.58%)		(5.36%)	

YAVA MINES LIMITEDDiamond Drilling Program 1977Check Samples

<u>Hole No.</u>	<u>Sample Number</u>	<u>From</u>	<u>To</u>	<u>A.I.R.I. N.S. Lead %</u>	<u>Bondar Clegg Lead %</u>
248	853	26.0	31.0	0.57	0.61
249	859	56.6	62.0	0.82	0.79
250	3818	124.1	127.1	1.09	1.22
251	865	150.0	155.0	1.32	1.38
252	873	152.0	158.2	1.66	1.64
253	874	202.8	206.8	2.00	2.26
254	883	220.5	225.4	1.91	2.08
255	3847	250.0	254.9	2.45	2.54
256	3848	301.1	303.1	2.57	2.66
257	3853	295.0	300.5	2.80	3.00
258	3856	285.0	290.0	2.55	2.78
259	3860	262.6	268.2	3.23	3.66
260	3871	207.0	210.0	4.15	4.76
261	3872	205.8	210.8	4.00	3.82
262	3880	171.1	174.5	4.49	4.82
263	3888	155.0	160.0	3.50	3.72
264	3898	122.0	125.3	4.65	4.26
265	3902	102.7	105.0	4.81	3.58
266	3912	81.0	86.0	4.65	4.56
267	3915	59.7	64.1	2.98	2.90
268	3920	58.6	61.2	4.97	5.76
269	3927	88.5	94.3	6.13	6.48
270	3932	120.8	126.9	5.55	5.56
271	3942	149.0	154.5	5.55	5.90
272	3946	180.8	185.4	6.20	6.22
273	3950	189.2	194.2	5.82	6.92
274	3952	214.7	219.9	6.06	7.62
275	3961	246.0	252.2	8.29	9.06
276	3963	163.0	169.0	7.68	9.64
277	3966	139.6	145.6	6.20	7.88
278	3969	189.6	191.0	9.23	11.0
279	3991	180.1	190.0	4.97	5.76
279-A	5344	202.6	207.5	6.18	6.38
280	3975	183.5	189.5	4.00	5.30
281	3981	152.2	157.3	6.55	7.10
282	3982	106.4	112.4	5.60	7.20
283	3985	70.0	76.4	6.71	8.30
284	3995	38.7	48.0	6.20	8.30
285	5301	31.7	34.0	3.54	5.10

consistently low than

Hole No.	Sample Number	From	To	A.I.R.I.	Bondar
				N.S. Pb. %	Clegg Pb. %
286	5306	98.2	104.2	7.00	8.12
287	5310	201.1	207.6	4.75	5.00
288	5312	238.9	244.9	6.06	6.50
289	5315	184.0	190.9	4.56	5.22
290	5318	220.9	228.5	4.56	5.16
291	5320	246.1	252.1	7.19	7.38
292	5326	254.3	260.0	4.75	4.88
293	5332	259.7	262.9	4.44	4.72
294	5357	274.0	274.5	4.00	3.72
295	5362	275.6	281.6	4.17	4.00
296	5350	257.4	263.4	3.97	4.04
297	5353	250.7	251.5	2.73	2.78
298	5337	205.8	209.9	5.69	6.20
299	5338	222.6	224.6	1.22	1.38
300	5368	180.0	186.0	2.34	2.54
301	5373	248.9	256.5	5.00	5.06
302	5375	273.6	279.6	5.56	6.36
303	5380	290.1	292.3	11.90	12.6
304	5382	279.2	285.2	9.78	10.2
305	5388	262.1	268.1	8.22	8.72
306	5396	228.0	235.0	0.61	0.54
307	5400	190.0	196.1	0.91	0.60
308	5505	158.7	163.0	0.42	0.58
309	5511	103.3	109.3	0.88	0.84
310	5516	129.2	135.2	1.50	1.50
311	5518	181.3	185.6	4.00	4.64
312	5527	192.0	198.0	2.38	2.30
313	5532	215.0	221.0	1.91	1.80
315	5536	116.8	122.8	2.93	3.16
316	5538	71.2	78.0	4.67	4.84

RESOURCES: (Cont'd)(d) Potential Ore Reserves

The ore reserves developed from the close spaced diamond drilling program carried out in 1977, represent the up-dip part of a zone of mineable reserves extending down dip from the drilled section north of the Salmon River, which acted as a physical barrier to conducting further drilling during the summer months.

The down dip extension of the ore zone is indicated by two diamond drill holes, numbers 142 and 144 drilled in 1963. These holes are 600 feet apart and are respectively 600 feet apart and are respectively 600 and 400 feet horizontal distance beyond the north edge of the ore reserve block. From the continuity of the structure and mineralization these holes strongly infer an extension of the ore reserves down dip to the north; the thickness, width and grade of which remain to be determined. The extension of the zone down dip of these two drill holes also remains to be explored.

It is intended during the winter months, after the Salmon River freezes, to carry out further diamond drilling north of the river to outline these reserves in better detail.

The initial mining project is confined to the West Zone of the Silvermines deposit. Important objectives in this preliminary operation are:

- (a) To obtain a better understanding of the nature of the higher grade zone within the wider area of lead mineralization.

(d) Potential Ore Reserves (Cont'd)

- (b) Derive economic guidelines for mine development in the Central and East Zones.

The ore reserve figures on Page IV - 8 which were developed by Phelps Dodge indicate total reserves of ^{open pit indicated if possible} 30,283,561 tons @ 2.97% short tons ore containing 870,243 short tons of lead metal. Of this, 6,581,115 tons of ore (^{@ 3.02%} 21.7%) are contained in the West Zone, containing 19,874 tons of lead metal (22.8%). For reasons explained separate mining development will be contemplated within each Zone.

In view of environmental problems affecting open pit operations on a large scale in this area and the substantial tonnages of waste rock and overburden which would require to be removed to expose the mineralized horizon, it is doubtful whether open pit operations, except on a confined scale, could be seriously entertained. However, underground mining on a substantial scale can be contemplated in the future and this is a principal factor in mind in conducting the preliminary program in the West Zone.

In the East Zone a close spaced drilling program at 100 feet centres carried out in 1969 indicated "a definite horizon of higher grade mineralization (4% lead and up) extending through the drilled area". This program covered an area of 600 feet square but has not since been followed up by further drilling. The information gained from close pattern drilling in the West Zone indicates that continuous zones of better grade mineralization are likely to be found elsewhere in the deposit. The higher grade section in the

(d) Potential Ore Reserves (Cont'd)

East Zone remains to be further extended and there are promising areas in the Central Zone which require further exploratory drilling. The intention is to undertake further drilling in all these areas with a view to increasing ore reserves. The knowledge of rock conditions and mining costs to be gained from the underground program in the West Zone also makes a significant contribution to the overall exploration and development of the property.

X

V

MINING

- (a) General Procedure
- (b) Trackless Mining Equipment
- (c) Development and Production
- (d) Ventilation
- (e) Pumping

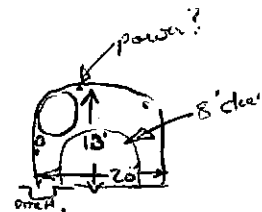
MINING:(a) General Procedure

It is proposed to achieve the underground development and the underground production of ore from the Silvermines deposit by the use of trackless mining equipment.

The average dip of the bedded deposit is 15 degrees in a direction North 41° West. Although it is possible to operate trackless equipment on a road bed sloping at 15 degrees or a grade of 27%, it is felt continuous operation on such a steep gradient imposes excessive strain on equipment with a resultant increase in maintenance costs. Thus it is proposed all main development headings and likewise the long axis of parallel stope rooms be in a direction South 77° West or North 21° East, which will result in an average floor gradient of 7 degrees or 12.25%.

Preliminary plans, subject to minor variations, call for the commencement of the adit decline through over burden at local co-ordinates 48,900 North and 47,500 East and the advance of the decline in a direction $N30^{\circ}$ E at a gradient of minus 7 degrees or 12.25% for a distance of 200 feet. This decline will then turn to a direction $S77^{\circ}$ W and enter the sub-outcrop at approximate local co-ordinates 49,050 North and 47,400 East. This development adit will continue in the same direction at the same gradient for at least another 1,000 feet. This development drive will then be turned in a clockwise direction 124 degrees to a bearing $N21^{\circ}$ E and will continue at the same decline gradient of minus 12.25% for as much as an additional

1,000 feet. Depending upon ground water conditions, a preliminary sump will be established below the floor of the main adit at a point 200 feet or 300 feet inside the portal on the south side or left hand side of the adit. Likewise, within 200 feet or 300 feet of the portal additional development headings can be driven on the north side of the adit, same being established at predetermined locations since these development headings will form the main development heading in a stope room. All development headings in both main adit and stope drives will be driven with the floor of the heading at the bottom of the Pennsylvanian sandstones and at the contact with the Windsor formation of shales and limestones. Thus all headings of adits and stope development drives will be in ore.



The proposed size of the adits and any stope development drives will be 12 feet high by 20 feet wide with a slight crown in the roof to a maximum height of 13 feet above floor level. The 42" ventilation duct, 6" high pressure air pipeline, 2" feed water pipeline and 4" or 6" discharge water pipeline, will be suspended high on the south wall and any pipeline crossovers to the north side of the adit must be kept tight to the roof. Power feed cables will be suspended from the upper left roof corner well above and away from any pipelines. The ditch for water drainage will be on the south side of the main adit driven in the direction of S77°W and opposite to the production stopes, whereas the ventilation duct, pipelines and drainage ditch must be on the east side of the adit driven at N21°E.

In this proposed room and pillar method of ore recovery

the widths and lengths of rooms and pillars will depend on the rock strengths and underground conditions. Until such conditions are known, following considerable underground development work, it is suggested the room and pillar pattern be 50 feet wide by 250 feet long and that end pillars between the wall of the development adit and the end of the room be 25 feet. Likewise the size of the entrance into the room from the development adit be kept to a maximum size of 10 feet wide by 12 feet high. Once inside the room dimensions, then the development adit, driven on the centre line of the room, can be expanded to full size of 20 feet wide by 12 feet high and this can be slashed to an eventual room width of 50 feet. Likewise the stope can be mined to the full economic height of the ore, either in a single lift or in a double or triple lift, depending on the height involved. Since backfilling procedure is planned, which will eventually result in a 90% or better recovery of ore, this mining in lifts can be accomplished on various thicknesses of backfill. A panel length of 250 feet, less thickness of end pillars, can be modified once the ground conditions have been established.

Roof support and ground control will be effected by the use of rock bolts and possibly wire mesh, with the spacing of rock bolts and the use of mesh dependent on ground conditions.

A secondary entrance, to be used for permanent introduction of ventilating air and a second exit, is tentatively located at local co-ordinates 49,700 North and 47,500 East.

MINING (Cont'd)(b) Equipment

The trackless mining equipment proposed for use in this underground development and production programme is as follows:

(i) Drilling

One Jarvis Clark Jumbo Drill Rig, MJM21, with 3 booms and 3 Series 125 Tampella pneumatic rock drills on 12 foot slides, all mounted on a four-wheel hydrostatic drive, centre articulated carrier unit powered by a Deutz Diesel F6L-912W engine with a mine service rating of 78 h.p. at 2300 r.p.m. Operating weight 34,800 lbs.

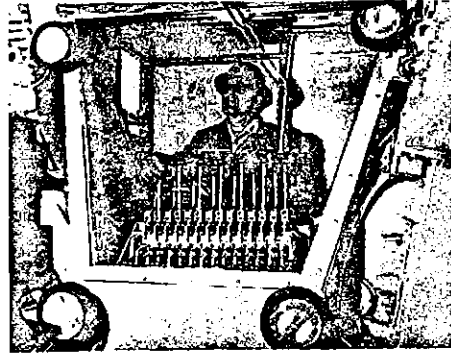
This one new unit will be the main drilling rig in the mine.



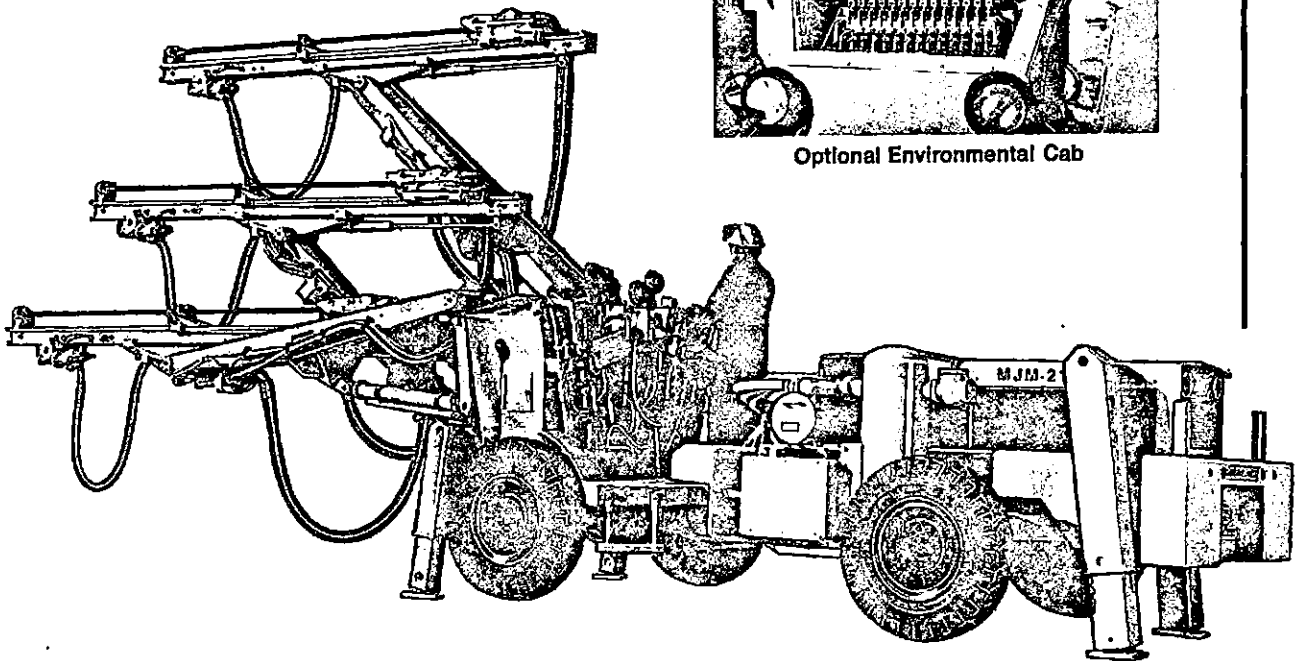
Jumbos

MJM-21

3 Boom Paramatic



Optional Environmental Cab



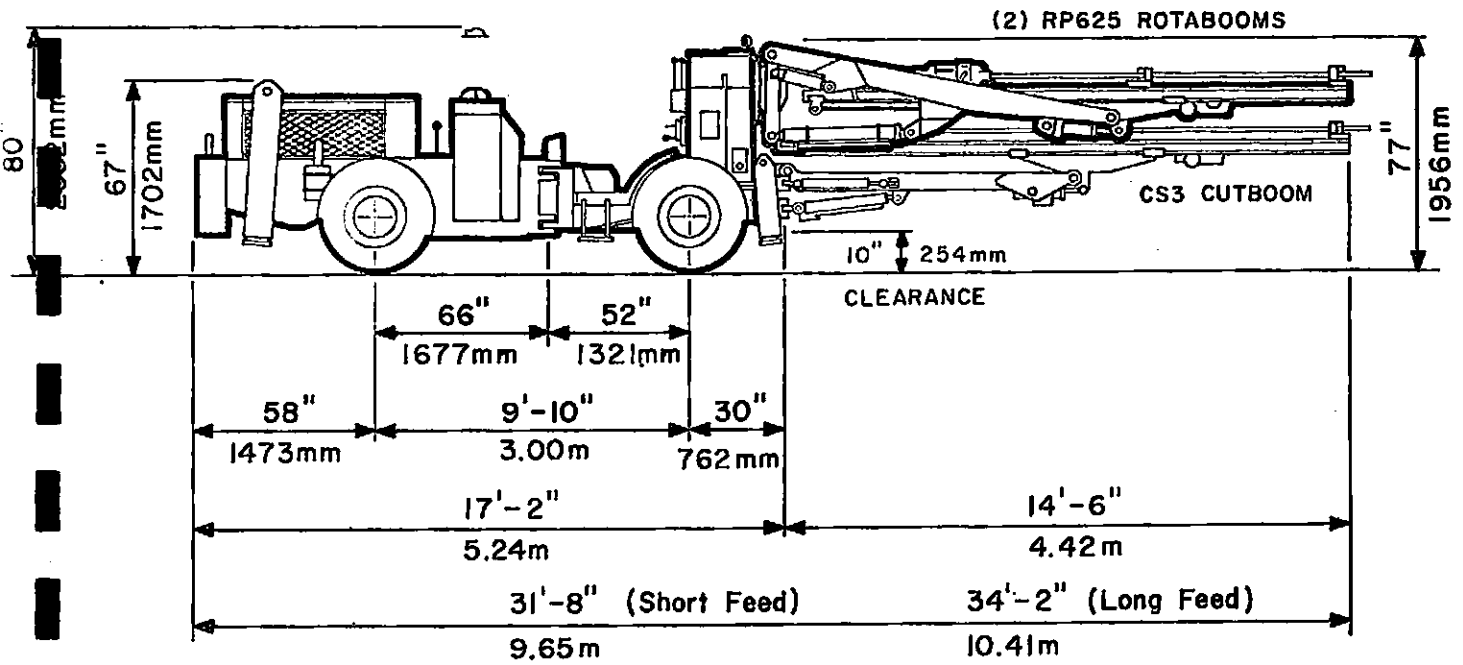
FEATURES

Jarvis Clark developed the ParaMatic Jumbo a decade ago and since that time they have gained almost universal acceptance in underground mining and tunnelling. The MJM-21 has been by far the most popular Jumbo and is the real workhorse for trackless development. The basic features that have made the MJM-21 such a success are parallel drilling of small holes accurately for long rounds; feed automatics that reduce the amount of operator attention; center articulated carrier for working on tight curves; four wheel drive for negotiating steep ramps; low maintenance due to simplified construction and, above all, absolutely proven performance based on millions of feet of drilling experience.

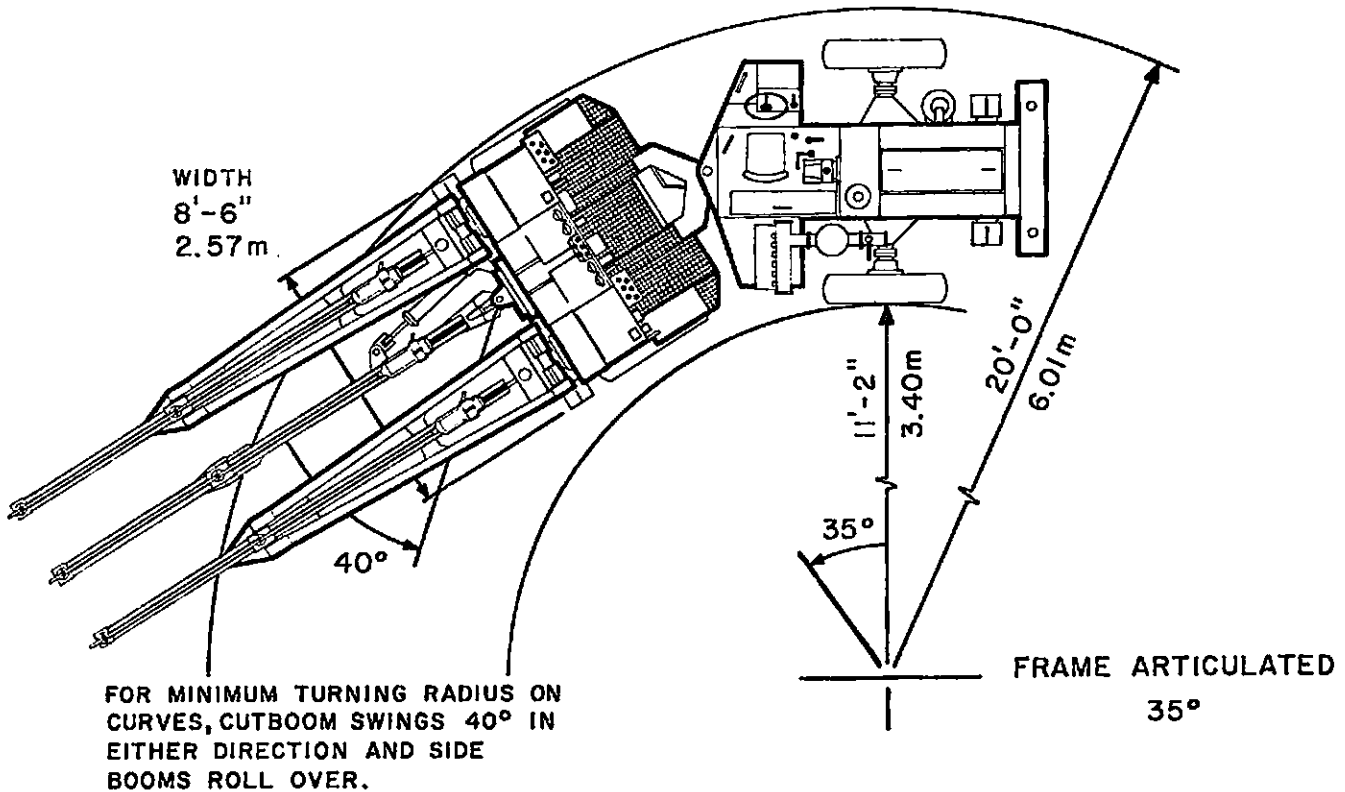
The MJM-21 three boom pneumatic Jumbo features our series 125 valveless, independent rotation, short stroke, high BPM rock drills specifically designed to drill small holes (1½" or less) FAST.

"MJM-21 the proven standard"

3 Boom Parallel Drilling Mobile Jumbo



2 RP625 Rotabooms 1 CS3 Cutboom



Specifications

MJM-21 V-4(c)

CARRIER

The basic carrier used on the model MJM-21 three boom jumbo is essentially the same as that used on the JUT-43 utility vehicle as detailed on spec. JC-163/5/78. It is a four wheel drive, center articulated carrier powered by a Deutz model F6L-912W engine with a mine service rating of 78 HP.

TRANSMISSION AND TRANSFER CASE

Four speed, fully synchro-mesh mechanical transmission with separate constant mesh reverser and transfer case.

AXLES, WHEELS AND TIRES

Front and Rear: Spiral bevel differential, full floating, planetary wheel end drive with no spin differentials. Demountable rims, tube type, nylon, hard rock lug.

Tire Size, front and rear 10:00 x 20, 14 Ply

BRAKES

Service: Two drum brakes mounted independently on the transfer case are operated from one section of the master cylinder and work through drive shafts controlling each axle.

Hydraulically applied disc brake on input to rear axle operated from the second section of the master cylinder and controlling both axles.

Parking: Hand operated band type.

STEERING

Hydraulic power orbitrol type steering for bi-directional operation. Machine center articulated with 2 hardened steel pins and self-aligning ball bushings.

OSCILLATION

Front Axle Oscillation $\pm 10^\circ$ for a total of 20° .

TANK CAPACITIES

Fuel 28 U.S. Gal. (106 L)

Hydraulic — Carrier 15 U.S. Gal. (56 L)

Hydraulic — Boom (RP625) 15 U.S. Gal. (60 L)

BOOMS

Two RP625 RotaBooms

Each RP625 is independently mounted on its own base and has self-contained controls. It has full 360 degree hydraulic power rotation and covers a diameter of 20' 6" (6.25 M) with a blind spot of only 7" (180 mm).

Each RP625 has an integral hydraulic tank which is inter-connected and one boom is equipped with an air operated hydraulic pump complete with filter which operates all three booms.

One CS-3 Cutboom

This boom is mounted in between the RP625 RotaBooms and moves 30" (760 mm) laterally, vertically 96" (2438 mm) and swings 42 degrees to either side of center.

Lock valves are standard on all cylinders.

Optional Universal Head

Where increased coverage is required the CS-3 Cutboom can be fitted with a universal head, which has parallelism only in the normal Cutboom coverage area.

Parallelism

Standard RP625 RotaBoom and CS-3 Cutbooms have parallelism as standard features. When moving from hole to hole parallelism is automatically maintained. When the angle is set for the top holes, side holes and lifters it is automatically maintained.

The operator can set the booms to not use the parallel feature if desired.

CENTRAL LUBRICATOR

One model KVL-45/6 six port central lubricator with individually adjustable ports and inlet filter is standard.

Volume 1.6 cu. ft. (45 L)
Air Pressure (Maximum) 150 psi (10 atm)
Air Distribution Pipe (6) 1 1/4" (32 mm)
Weight 132 lbs. (60 KG)

DUAL LIGHTING SYSTEM

One marine type 55 amp alternator is driven by the diesel engine and one by a vane type air motor, either one of which charges the battery and runs the lights.

CHAIN FEEDS

Three model KS50A/40 chain feeds for 12' 6" (4 M) long steel and two centralizers are standard. Longer or shorter feeds can be provided.

Variable feed pressure control and spring recoil mechanism provide optimum drilling speed and maximum chain life.

OPTIONAL FEED AUTOMATICS

When the drill has completed its drilling cycle, it is automatically shut off, water flushing changes to air flushing and the feed reverses.

ROCK DRILLS

Three series 125 model C40 (for use with 1" hex collared steel) or L40 (for use with striking bar), independent rotation short stroke, high BPM pneumatic rock drills.

MAJOR COMPONENT BREAKDOWN

Front Frame 34" x 54" (864 mm) x (1372 mm)
Rear Frame 27" x 35" (685 mm) x (889 mm)
RP625 Boom
Assembly 36" x 47" (914 mm) x (1194 mm)
CS-3 Cutboom
Assembly 30" x 36" (762 mm) x (914 mm)

The entire unit is made to breakdown to go through a 5' (1.5 M) x 5' (1.5 M) opening without cutting and re-welding.

WEIGHTS

Shipping 34,400 lbs. (15,604 KG)
Operating 34,800 lbs. (15,785 KG)

STANDARD EQUIPMENT

Batteries — 2-95 amp heavy duty

Air driven alternator — 55 amp

— Oxy-catalyst exhaust system

Low oil pressure/high engine temperature audio visual alarm system

Headlights — 6 front, 2 rear

Two 16" (405 mm) levelling jacks (boom end)

Two 30" (762 mm) levelling jacks (engine end)

AC3 air cleaner with Sarco strainer

4" main air shut off, bull hose rack

Gauges:

Engine hour meter, ammeter

Engine temperature, engine oil pressure

Water pressure, air pressure

OPTIONAL EQUIPMENT

Roll over protection for operator

PTX scrubber or fume diluter

Cutboom universal head

Feed automatics

Optional rock drills and feeds

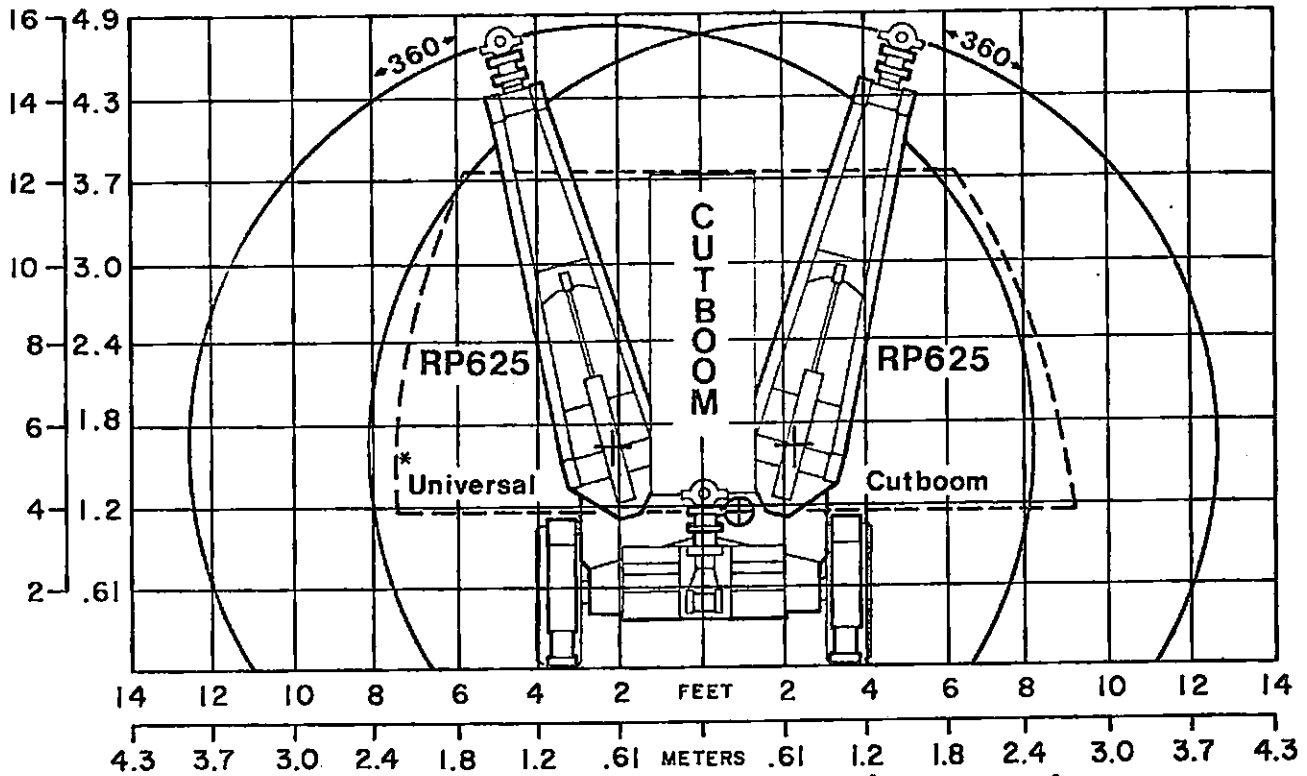
Emergency spring applied disc brake

Soundproof cab (model MJM-21A)

60" stroke jacks (boom end)

The manufacturer reserves the right to change the specifications or design of this machine at any time without notice.

Boom Coverage



* Universal Cutboom Is Optional Equipment

Performance

* VEHICLE SPEEDS — LEVEL PERFORMANCE

GEAR	1	2	3	4
MPH	1.2	2.0	3.3	5.1
KM/H	2.0	3.2	5.3	8.2
MAXIMUM RECOMMENDED GRADIENT				25%

BOOM COVERAGE

TWO RP625 ROTABOOMS	
EFFECTIVE DIAMETER	20' 6" (6.25 M)
BLIND SPOT	7" (208 mm)
ONE CS-3 CUTBOOM	
EFFECTIVE WIDTH	30" (762 mm)
EFFECTIVE HEIGHT	96" (2438 mm)
CS-3 WITH UNIVERSAL HEAD	
EFFECTIVE AVERAGE WIDTH	14' (4.2 M)
EFFECTIVE HEIGHT	8' 0" (2.4 M)
MAXIMUM HEADING SIZE	240 SQ. FT. (22 M ²)

* Vehicle speeds as stated are based on reasonable roadbeds, 60% adhesion factor and are not necessarily a guarantee of performance.

For further information contact:



JARVIS CLARK CO. LTD.

BOX 1110, NORTH BAY, ONTARIO, CANADA P1B 8K4
(705) 474-5520 TELEX 067-76154

Branches:

SUDBURY • TIMMINS • ELLIOT LAKE • THOMPSON
VANCOUVER • BATHURST • NAVAN, IRELAND

(b) Equipment (Cont'd)

Two Gardner-Denver Henderson Type Jumbos,
JMT-A2 with 3 booms and 3 Series 125 Tampella
pneumatic rock drills on 12 foot slides, all
mounted with travel, boom and drill controls
for one-man operation on a chassis with heavy
duty caterpillar type tracks driven by two
25 h.p. air tramming motors, piston type.

These two reconditioned units with new rock
drill machines will be secondary drilling
rigs in the mine.

Two Gardner-Denver Air Trac Drill Units,
Model ATD 3100, each complete with one PR123J
(4½") rock drill, RMH-10 feed and LO30A
3-gallon line oiler. Effective boom length
6' - 8", overall track length 8'-2", maximum
horizontal drill height 8'-10", maximum drill
pattern 11'-10". Rock drill has independent
rotation motors, 4½" bore, 38½" overall length,
1½" air hose to percussion motor, 1" air hose
to rotation motor, and is fitted for use with
1600V series drill steel and 1.3/4" round splined
drill shank. Unit Serial Numbers 4980 and 4986.

Unit Weights - Air Trac	-	7100 lbs.
10' Feed	-	770
PR123J Drill	-	280
Feed Extension 3'	-	<u>150</u>
		<u>8300</u>

These two reconditioned units will be used on
surface rock drilling work and may be used as
tertiary drilling rigs in the mine.

(b) Equipment (Cont'd)

Two Jackleg Type Rock Drills - complete with line oilers.

These new rock drill units are for special rock drill work.

Two Stoper Rock Drills - complete with line oilers. These new stoper rock drills are for drilling vertical or inclined up holes, such as in raise work and for rock bolt installation.

One Pitman Pelican Aerial Rig, used for scaling and roof bolting, Pitman Model PL28 E.H. with:

- 28 foot boom and turret assembly.
- hydraulic manifold assembly.
- non-insulated hose carrier assembly.
- special low frame with externally mounted hydraulic reservoir.
- BA-29 all steel basket.
- special hydraulic bucket levelling system.
- special hydraulic "A" frame fully enclosed outrigger assembly with check valves and two spool valves.
- special lifting attachment for mounting on Pelican boom to accommodate a stoper rock drill.

All the above is mounted on the carrier portion of a reconditioned Joy Drillmobile on 4 wheels with pneumatic tires and tubes, 8.25 x 15, 12 ply nylon hardrock lug, with two 15 h.p. Pistonair trammig motors for propulsion and with independent air motor and generator for

(b) Equipment (Cont'd)

lights. Approximate weight of complete unit 3800 lbs. This reconditioned unit can be hauled to the general work area by diesel engine driven equipment and is self-propelled by air motors in the work area adjacent to high pressure compressed air lines.

(ii) Loading

One Diesel Jarco Scoop LHD Unit, JS-500,
4-wheel drive, 5 cu.yd. capacity, driven by
a Deutz F8L-718 diesel engine with a mine
service rating of 180 h.p. at 2300 r.p.m.
Wheels and tires are demountable rim, tube
type with nylon hard rock wide lug type, size
18:00 x 25, 24 ply front and rear. Brakes are
air over hydraulic disc on all four wheels with
separate systems front and rear and a separate
drive line disc parking brake. Steering is by
hydraulic power stick steering for bi-directional
operation.

Operating weight (empty) 46,750 lbs.

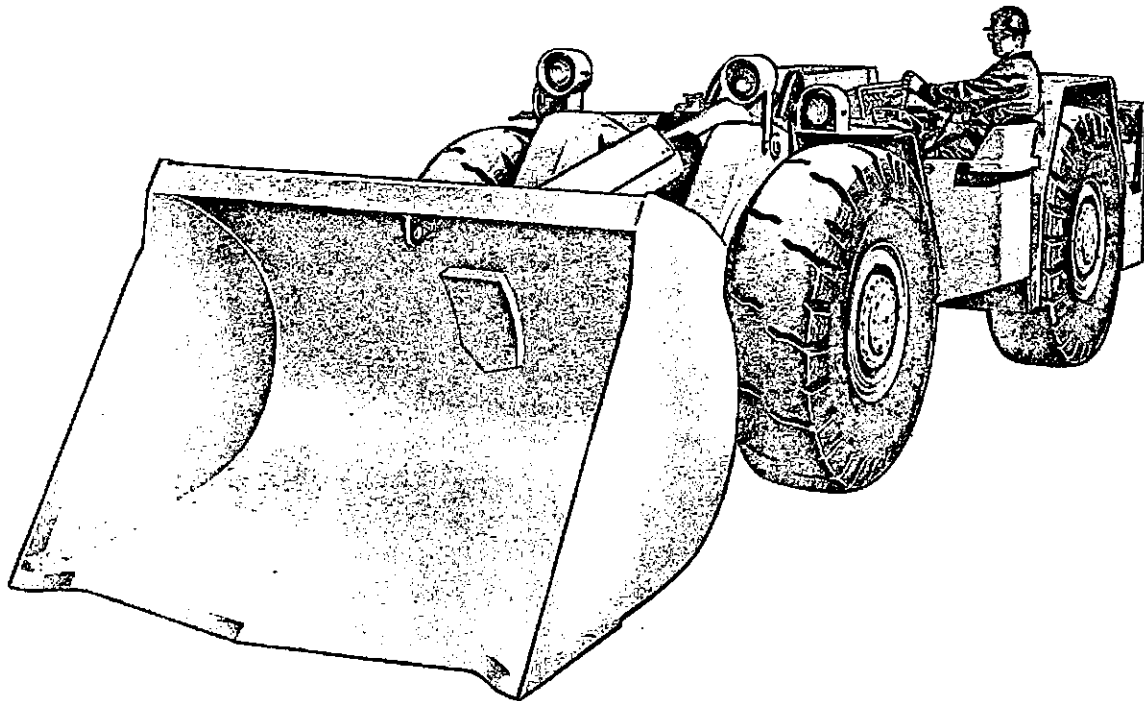
This one new unit will be the main loading unit
in the mine. As development and production
progresses it will be supported by a second
similar unit.



JarcoScoops

JS-500

5 Yard Diesel



FEATURES

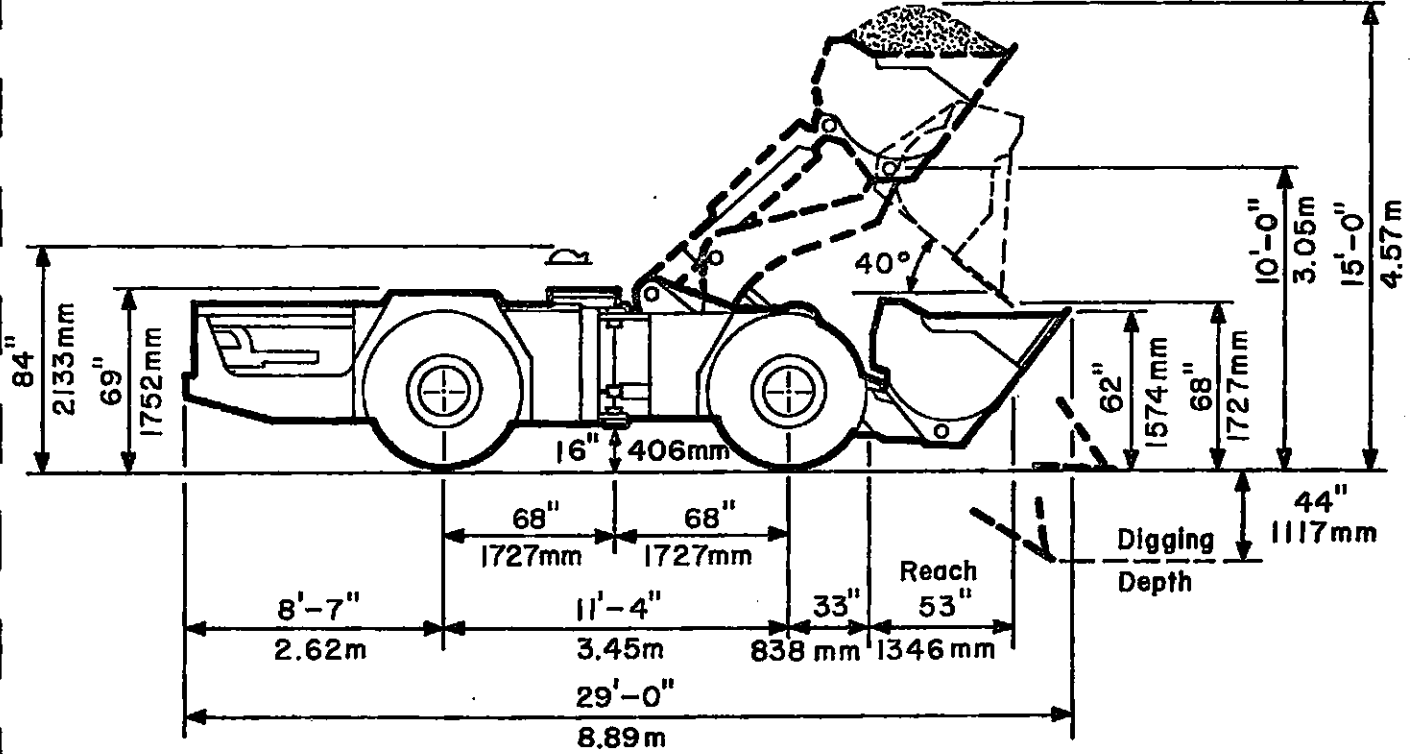
We worked closely with The International Nickel Company of Canada developing the JS-500. They gave us the benefit of their operating experience on hundreds of LHD machines over a six year period. We developed a prototype which was tested for over a year and modified before the first production machines were made.

The JS-500 "JarcoScoop" has been thoroughly tested and has proven to be a first class LHD machine.

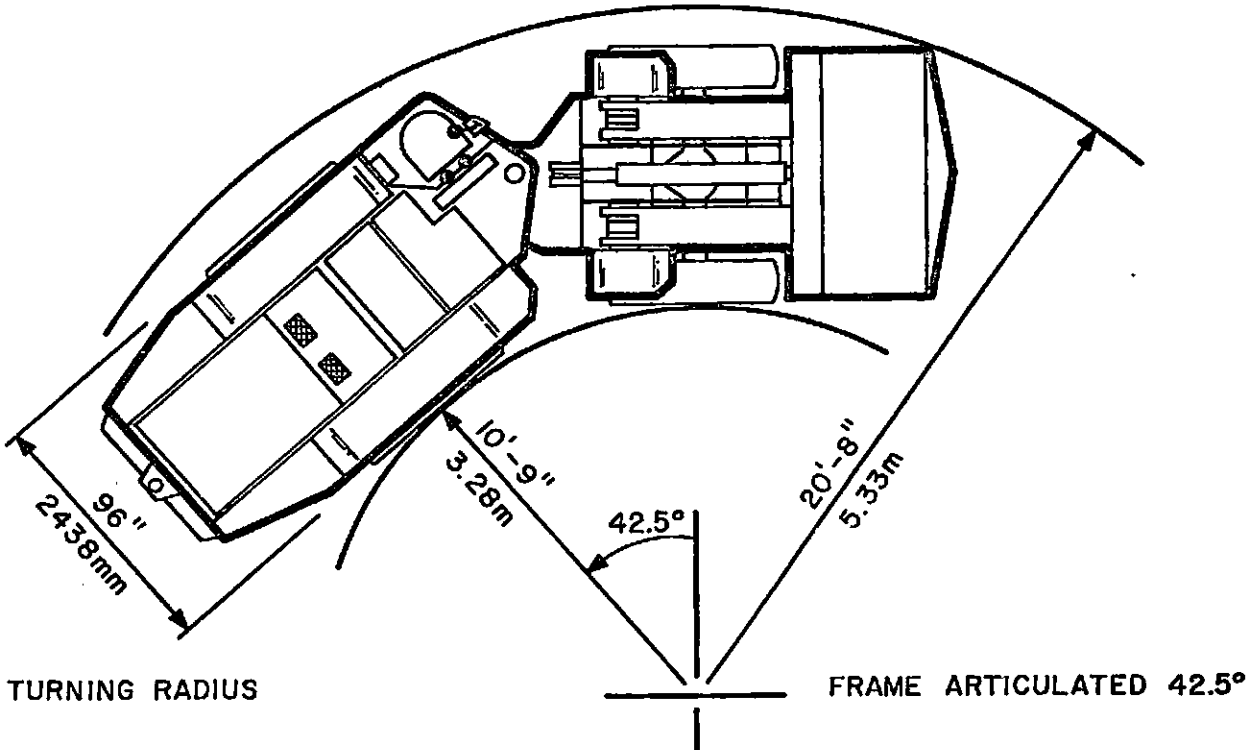
Dimensionally, the JS-500 is essentially the same as other 5 yard (3.8 M³) LHD machines. Also, wherever possible we have retained proven components, (engine, torque converter, transmission, front axle), for ease of servicing. But that is where the similarity ends. The JS-500 has 50 major improvements. These include a much stronger frame, heavier rear axle, disc brakes on all four wheels, improved emergency braking system, larger operator's compartment, improved operator controls, bolt on hydraulic tanks, breakdown construction and many more.

All the 50 improvements add up to a more productive, easier to operate, lower maintenance LHD machine.

4 Wheel Drive, 5 Cu. Yd. Diesel Scoop



180 HP *MS RATING



Specifications

ENGINE

Make and Model Deutz Diesel, F8L-714
 Maximum Power @ 2300 RPM 195 HP (198 PS)
 *Mine Service Rating @ 2300 RPM .. 180 HP (183 PS)
 Maximum Torque @ 1400 RPM 500 ft. lbs. (69 MKG)
 Bore and Stroke 4.75" (120 mm) x 5.5" (140 mm)
 Displacement 771 cu. in. (12.6 litres)
 Number of Cylinders V8
 Cooling Air
 Electrical System 12 Volts

TORQUE CONVERTER

Heavy duty industrial type, single stage with built in retardation. Stall ratio 2.9:1.

TRANSMISSION

Full power shift 4 Speeds, Forward and Reverse
 Ratios 5.31:1 — 2.71:1 — 1.40:1 — 0.71:1

AXLES

Front and Rear: Spiral bevel differential, full floating, planetary wheel end drive with no spin differentials.
 Ratio: Differential 6.333:1
 Planetary 4.125:1
 Total Reduction 26.125:1

WHEELS AND TIRES

Demountable rims, tube type, nylon, hard rock, wide lug, deep tread, with underground mine service compound.
 Tire size, front and rear 18:00 x 25, 24 Ply

BRAKES

Service: Air over hydraulic disc on all four wheels. Separate systems front and rear, with maxi spring applied actuators on each master cylinder for emergency and parking.
 Brake Size 18" x 1/2" (457 mm x 13 mm)
 Parking: Spring applied drive line disc brake, automatically applied by low air or low clutch pressure, may also be manually applied.

STEERING

Hydraulic power stick steering for bi-directional operation. Machine center articulated with 2 hardened steel pins and self aligning ball bushings.
 Turning Angle 42.5° In each direction for a total of 85°.

OSCILLATION

Rear Axle Oscillation ±9° for a total of 18°, with heavy duty 12" (305 mm) diameter bronze bushings, and thrust washers.
 One wheel can drop or rise a total of 12" (305 mm) with all wheels remaining on the ground for maximum traction.

HYDRAULIC SYSTEM

Cylinders: Double acting with chromed stems
 Steering cylinder (1), diameter 5" (127 mm)
 Hoist cylinders (2), diameter 6" (152 mm)
 Dump cylinder (1), diameter 9" (229 mm)
 Filters:
 Return Line 20 Micron
 Suction Line with 18 magnetic rods 100 Mesh
 Pumps: Heavy Duty Gear Type
 Steering 30 USGPM (114 L/M) @ 2300 RPM
 Dump and Hoist 90 USGPM (342 L/M) @ 2300 RPM

TANK CAPACITIES

Fuel 65 U.S. Gal. (246 L)
 Hydraulic 81 U.S. Gal. (306 L)

SEATING

Operator side seated for maximum visibility and bi-directional tramming.

COMPONENT BREAKDOWN

Bucket less lip and side cutters 46 1/2" x 60" (1181 mm) x (1524 mm)
 Boom 54" x 56" (1372 mm) x (1422 mm)
 Front Frame (less fenders) . 54" x 61 1/2" (1372 mm) x (1562 mm)
 Rear Frame (less fenders) . 46" x 60" (1168 mm) x (1524 mm)
 Axles (2) 18" x 24" (457 mm) x (610 mm)
 Wheels and Tires (4) 21" x 67" (533 mm) x (1702 mm)

The entire unit is made to breakdown to go through a 5' (1.5 M) x 5' (1.5 M) opening without cutting or re-welding necessary.

WEIGHTS

Shipping 45,450 lbs. (20,818 KG)
 Operating (Empty) 46,750 lbs. (21,206 KG)

STANDARD EQUIPMENT

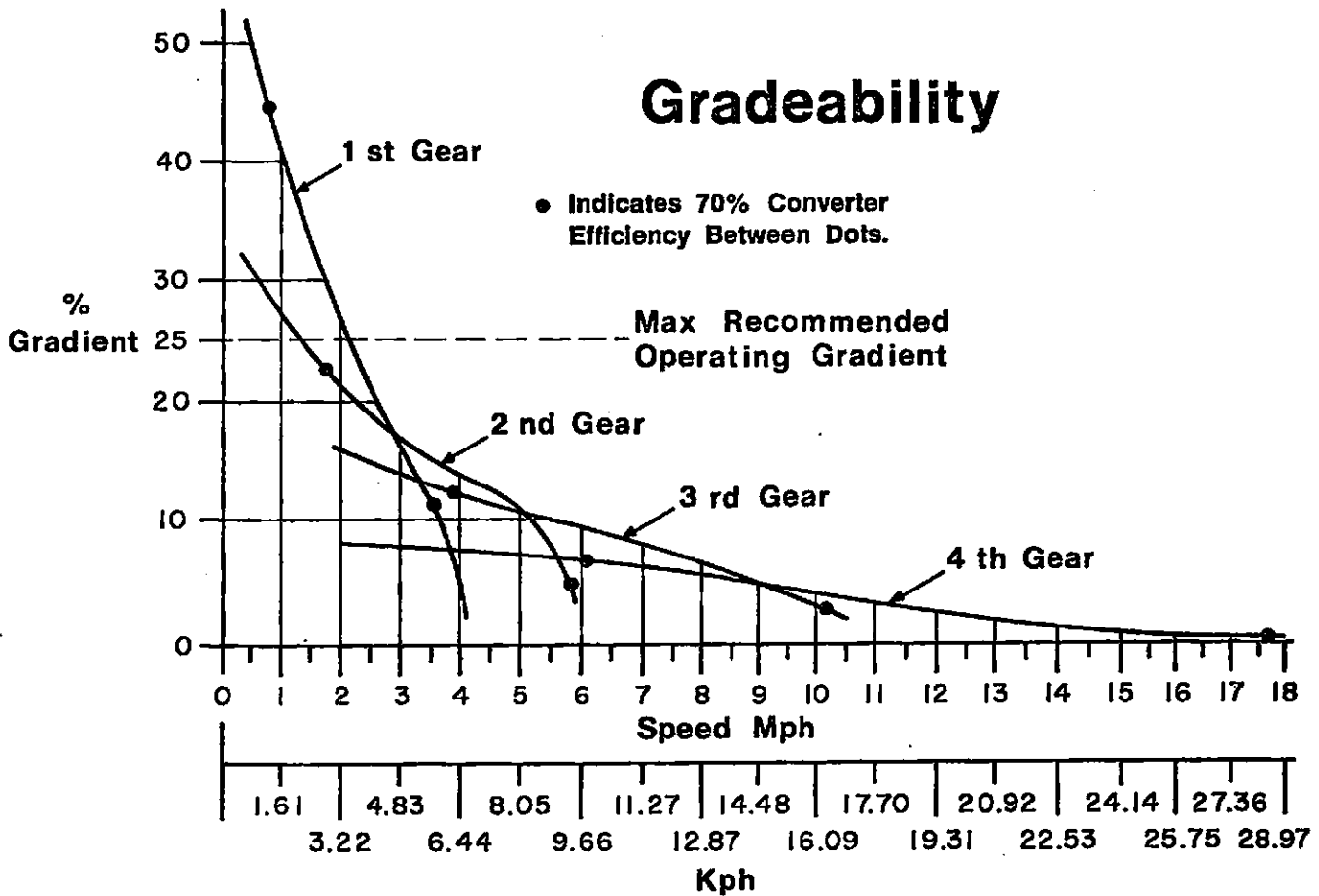
- Independent front and rear service brakes with over-stroke indicators
- Independent front and rear emergency brakes
- Emergency/parking disc brake with overstroke indicator
- Monostick bucket controls
- Auxiliary engine oil filter
- Auxiliary fuel filter
- Hydraulic power steering
- Air starting
- Neutral start only
- PTX scrubber and cowl muffler
- Dual dry type air cleaners
- Battery with Isolation switch
- Alternator, 85 amp, 12 volt
- Lights, 5 front, 4 rear
- Converter oil coolers (2)
- Foam padded operator's seat with adjustable suspension
- Low oil pressure/high engine temperature audio visual alarm system
- Center hinge lock arm
- Engine skid plate
- Abrasive resistant bucket lip and side cutters
- Gauges:
 - 4 air gauges with low pressure indicators
 - Voltmeter
 - Converter temperature gauge
 - Engine hour meter
 - Engine oil pressure gauge
 - Engine temperature gauges

OPTIONAL EQUIPMENT

- Roll over protection
- Optional buckets
- Bucket teeth
- Optional tires
- Seat belt
- Oxy-catalyst scrubbers
- Fume diluter
- Water scrubber

The manufacturer reserves the right to change the specifications or design of this machine at any time without notice.

Gradeability



Capacities & Performance

- * CAPACITY (SAE RATINGS)
 - STATIC TIPPING LOAD — STRAIGHT AHEAD 34,600 LBS. (15,700 KG)
 - BREAKOUT FORCE 36,000 LBS. (16,329 KG)
 - TRAMMING CAPACITY 15,000 LBS. (6,804 KG)
 - * BUCKET CAPACITY (STANDARD SAE RATING)
 - SEMI-NOMINAL HEAP 5 CU. YDS. (3.8 M³)
 - STRUCK 4.5 CU. YDS. (3.4 M³)
 - RAISING TIME 6 SEC.
 - LOWERING TIME 5 SEC.
 - VEHICLE SPEEDS LOADED — LEVEL PERFORMANCE

GEAR	1	2	3	4
MPH	4.1	5.8	10.6	18.0
KM/H	6.5	9.3	17.0	28.8
- * This JarcoScoop has a rating of 7.5 short (6.8 metric) tons under normal operating conditions. Bucket volumetric capacity is related to tonnage capacity by the SAE semi-nominally heaped rating based on material weighing 3,000 lbs./cubic yards (1,780 kg/cubic metre).



JARVIS CLARK CO. LTD.
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 (705) 474-5520 TELEX 027-76154

Branches:
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 VANCOUVER • BATHURST • NAVAN, IRELAND

Specification No. JC-159/12/75

For further information contact:

(b) Equipment (Cont'd)(iii) Hauling

One 26-ton End Drump Truck, JDT 426, 4 wheel drive, driven by a Deutz F12L-714 diesel engine with a mine service rating of 270 h.p. at 2300 r.p.m. Wheels and tires are demountable rim, tube type with nylon hard rock wide lug type, size 21:00 x 25, 28 ply front and rear. Brakes are air over hydraulic shoe type on all four wheels, with separate systems front and rear and a separate drive line disc parking brake. Steering is by hydraulic power stick steering for bi-directional operation.

Operating weight (empty) 46,750 lbs.

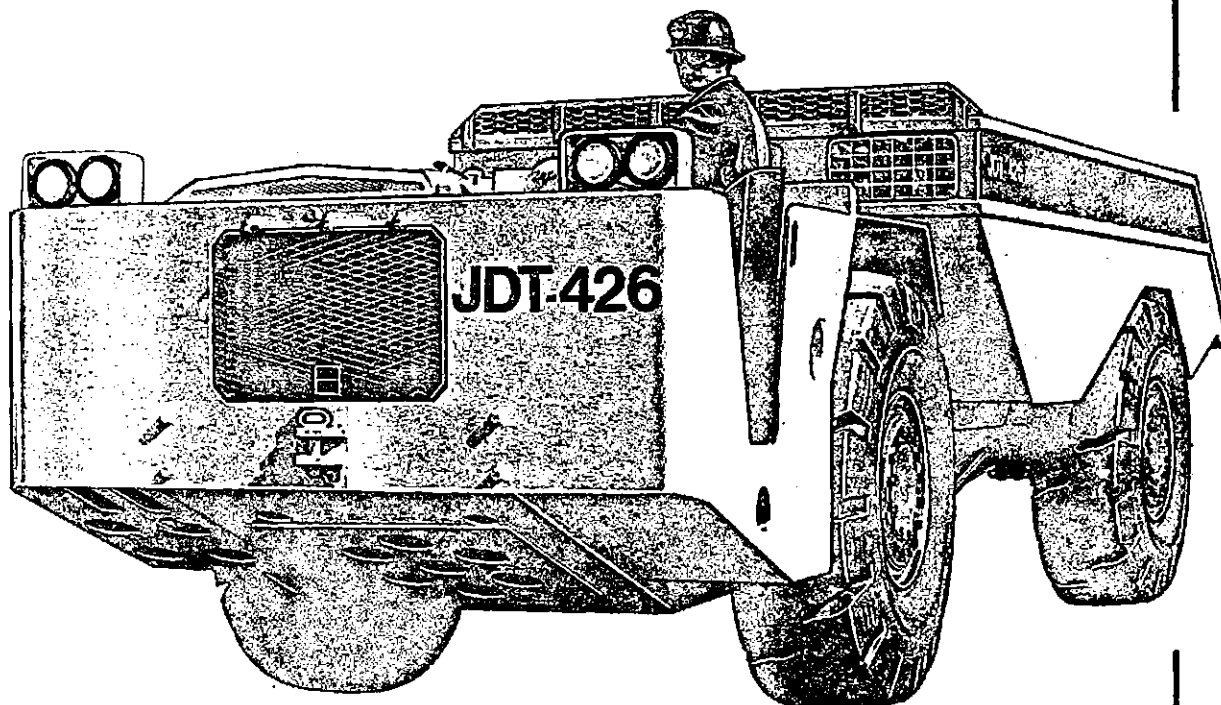
This one new unit will be the main unit trucking development and production ore from the mine to surface.

**JARVIS
CLARK**

Trucks

JDT 426

26 Ton End Dump



FEATURES

Jarvis Clark's JDT-426 end dump truck permits over the end or side loading with a 5 yard LHD machine or 3.5 loader. No special dumps are required for discharge which requires only 17' (5.2 M) headroom. JDT-426 trucks have been thoroughly proven on ramps, in tunnels and a full range of mine haulage jobs.

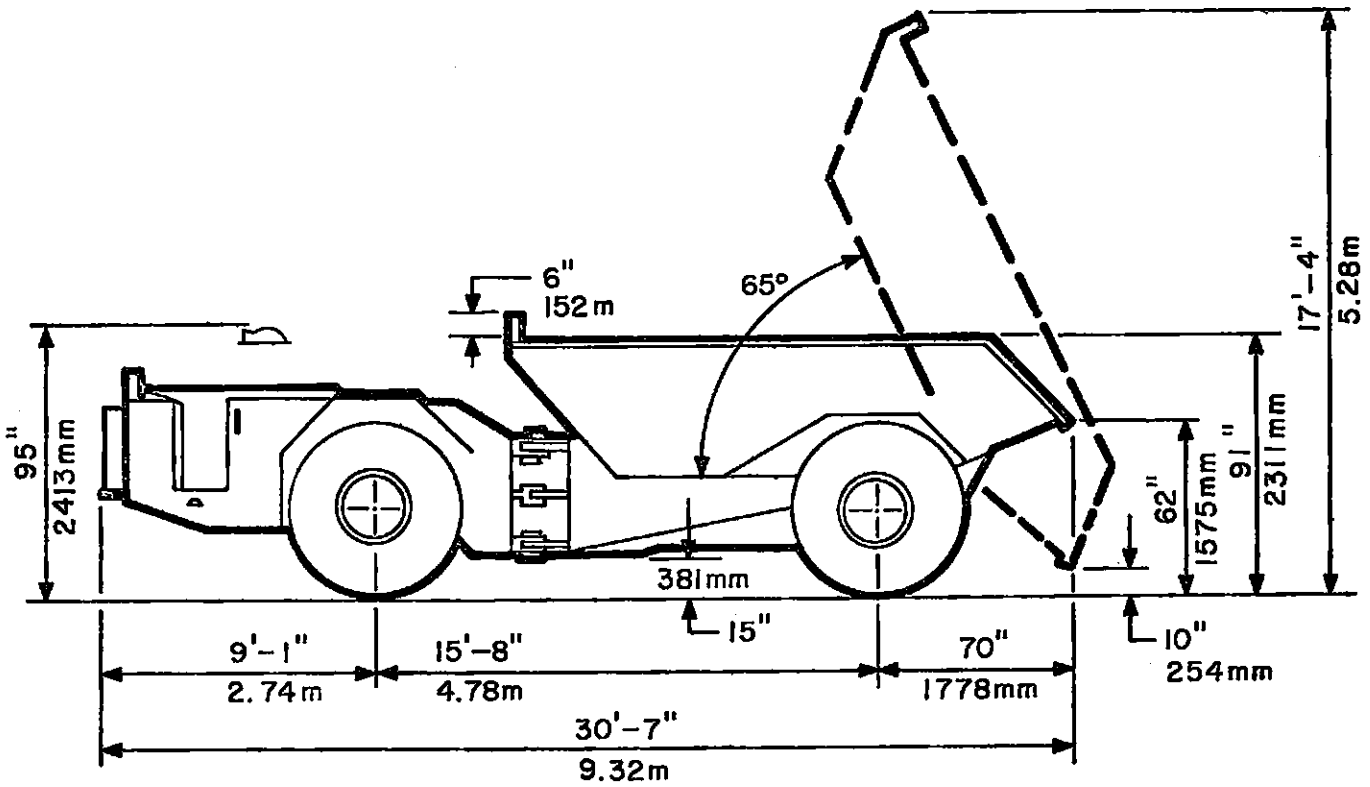
Operation of the JDT-426 is completely bi-directional without requiring the operator to move position. He is side seated with all controls and instruments located conveniently. Center pin steering and front axle oscillation assures complete maneuverability and control.

Powered by a 12-cylinder Deutz low emission, air cooled engine, the JDT-426 is equipped with an engine mounted torque converter and remotely mounted full power shift 4-speed transmission. Special planetary axles with shoe type brakes complete the rugged power train.

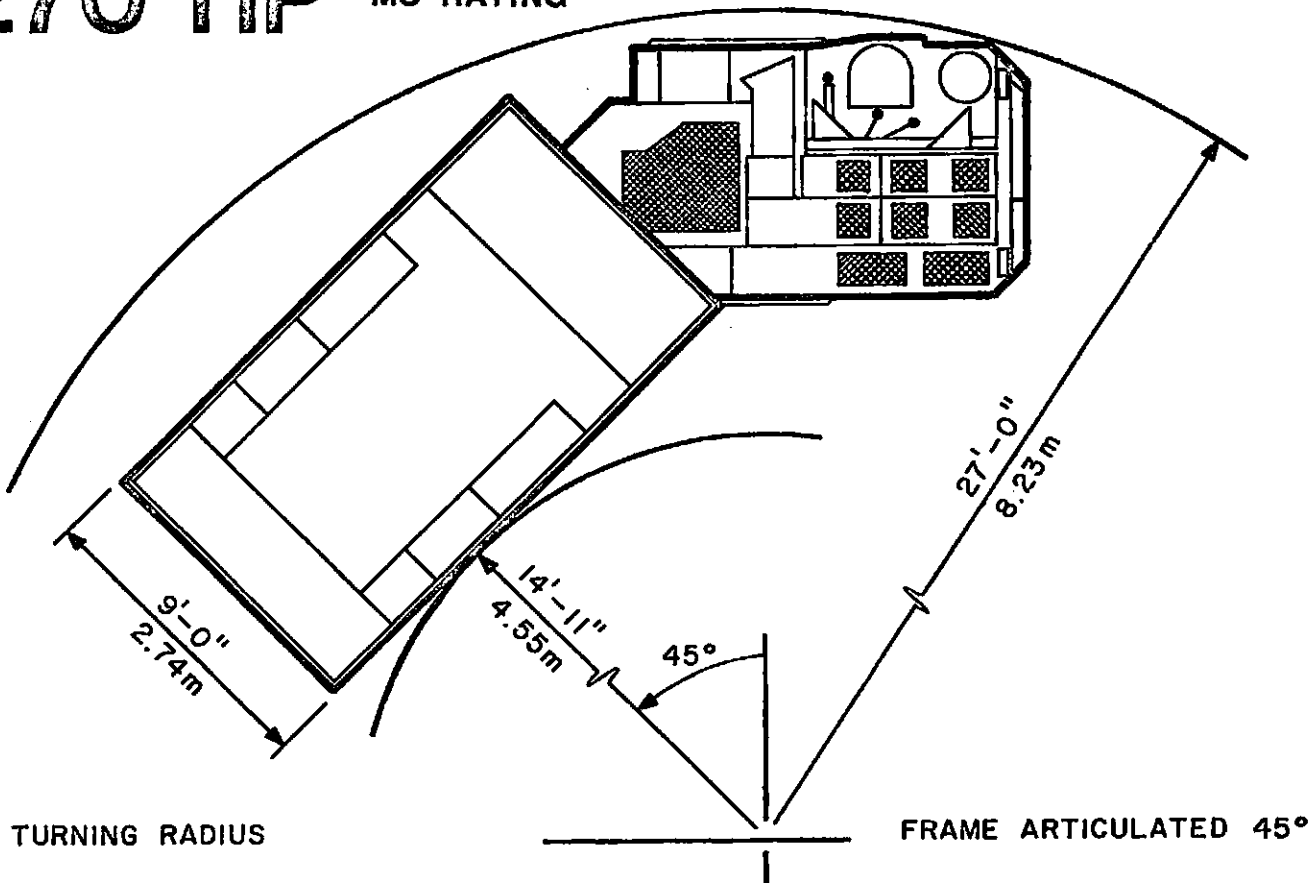
Dual service braking systems plus an independent emergency disc type brake provide the most effective braking now available.

Proven performance and dependability are the most outstanding features of the JDT-426.

4 Wheel Drive, 26 Ton End Dump Truck



270 HP *MS RATING



Specifications

JDT 42^{v-9(c)}

ENGINE

Make and Model Deutz Diesel, F12L-714
Maximum Power @ 2300 RPM 300 HP (304 PS)
Mine Service Rating @ 2300 RPM .. 270 HP (274 PS)
Maximum Torque @ 1250 RPM 760 ft. lbs. (105 MKG)
Bore and Stroke 4.75" (120 mm) x 5.5" (140 mm)
Displacement 1160 cu. in. (19.0 litres)
Number of Cylinders V12
Cooling Air
Electrical System 12 Volts

TORQUE CONVERTER

Heavy duty industrial type, single stage with built in retardation. Stall ratio 3.09:1.

TRANSMISSION

Full Power Shift 4 Speeds, Forward and Reverse
Ratios 5.33:1 — 2.74:1 — 1.40:1 — 0.72:1

AXLES

Front and Rear: Spiral bevel differential, full floating, planetary wheel end drive.
Ratio: Differential 6.333:1
Planetary 6.000:1
Total Reduction 37.998:1

WHEELS AND TIRES

Demountable rims, tube type, nylon, hard rock, wide lug, deep tread, with underground mine service compound.
Tire size, front and rear 21:00 x 25, 28 Ply

BRAKES

Service: Air over hydraulic shoe type on all four wheels. Separate systems front and rear, with lock on actuators on each master cylinder.
Brake size 20" x 7" (508 mm x 178 mm)
Parking: Spring applied drive line disc brake, automatically actuated by low air or low clutch pressure, or may be manually applied.

STEERING

Hydraulic power stick steering for bi-directional operation. Machine center articulated, with 2 hardened steel pins and self aligning ball bushings.
Turning Angle .. 45° in each direction for a total of 90°.

OSCILLATION

Front Axle Oscillation ±10° for a total of 20°, with heavy duty 12" (304 mm) diameter bronze bushings, and thrust washers.
One wheel can drop or rise a total of 15" (381 mm) with all wheels remaining on ground for maximum traction.

HYDRAULIC SYSTEM

Cylinders: Double acting with chromed stems.
Steering cylinder (1), diameter 5" (127 mm)
Hoist cylinder (2), 2 stage telescopic, diameters 7¼" (184 mm) 6¼" (159 mm)
Filters:
Return Line 25 Micron
Suction Line 100 Mesh
Pump: Heavy Duty Gear Type,
Steering 35 GPM (132 L/M) — Dumping 83 GPM (314 L/M) @ 2300 RPM

TANK CAPACITIES

Fuel 100 U.S. Gal. (378 L)
Hydraulic 75 U.S. Gal. (283 L)

SEATING

Operator side seated for maximum visibility and bi-directional tramming.

COMPONENT BREAKDOWN

Front Frame 58" x 58" (1473 mm) x (1473 mm)
R.H. Fender 53" x 52" (1346 mm) x (1320 mm)
L.H. Fender 53" x 54" (1346 mm) x (1371 mm)
Rear Frame 48" x 53" (1219 mm) x (1346 mm)
Box (Split) 58" x 54" (1473 mm) x (1371 mm)
Entire unit is made to breakdown to go through a 5' (1.5 m) x 5' (1.5 m) opening with cutting and re-welding necessary only on the box.

WEIGHTS

Shipping 50,400 lbs. (22,860 KG)
Operating (Empty) 50,900 lbs. (23,088 KG)

STANDARD EQUIPMENT

Independent front and rear service brakes
Emergency disc brake
Parking brake
Auxiliary engine oil filter
Auxiliary fuel filter
Transmission oil filter
Hydraulic power steering
Air starting
Neutral start only
Oxy-catalyst exhaust system
Dual dry type air cleaners
Battery with isolation switch
Alternator
Lights, 4 front, 2 rear
Converter oil coolers (2)
Foam padded operator's seat with adjustable suspension
Low oil pressure/high engine temperature audio visual alarm system
Center hinge lock arm
Dump box service support tubes

Gauges:

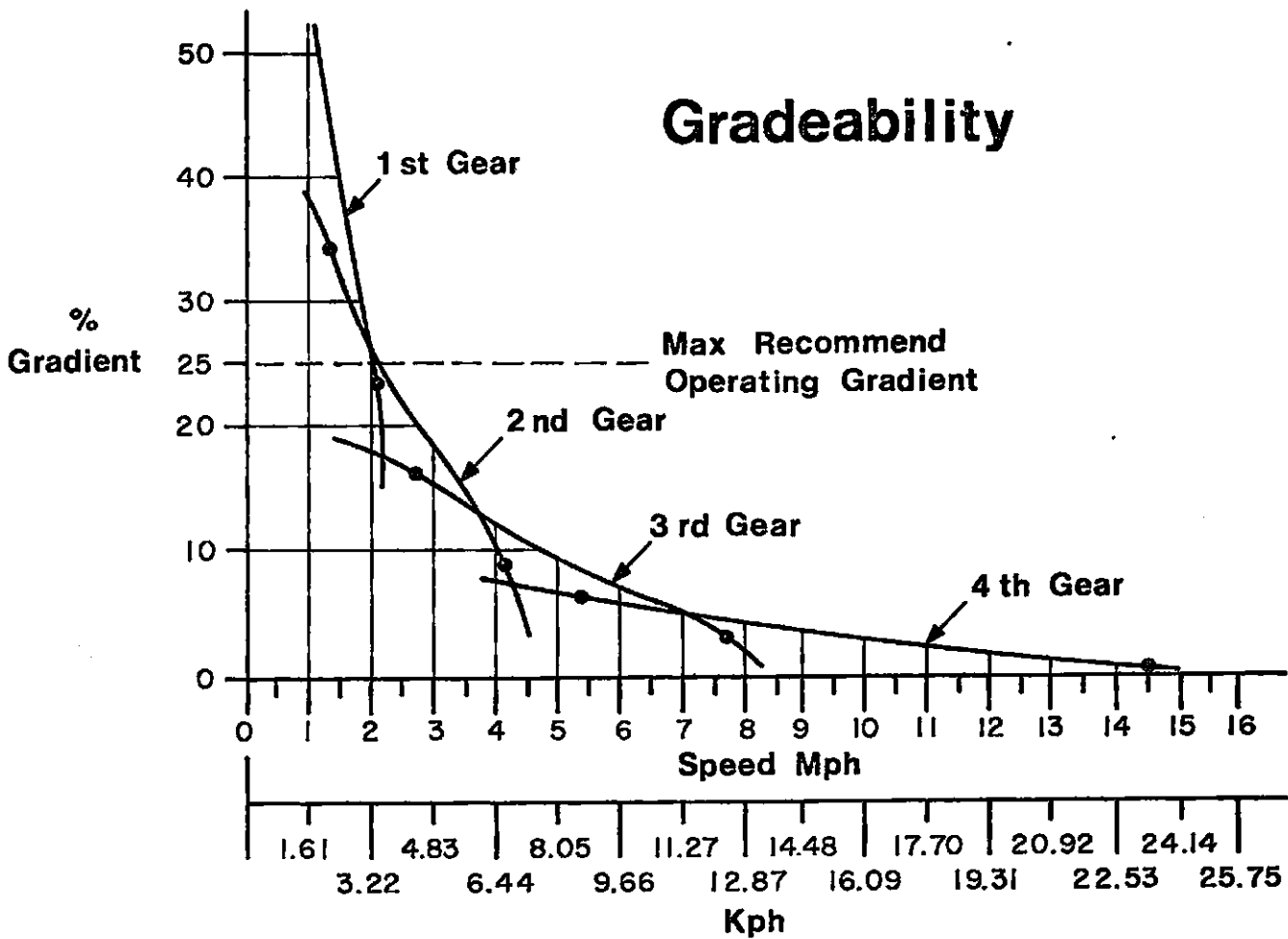
3 air gauges
Ammeter
Converter pressure gauge
Converter temperature gauge
Engine hourmeter
Engine oil pressure gauge
Engine temperature gauge
Dual engine temperature gauges

OPTIONAL EQUIPMENT

Cab with heater and defroster
Roll over protection
Emergency auxiliary steering system
Fume diluter exhaust system
PTX scrubber
Water scrubber
Optional boxes and special liners
24 volt electric starting system

The manufacturer reserves the right to change the specifications or design of this machine at any time without notice.

Gradeability



Capacities & Performance

CAPACITY *

MAXIMUM 26 TONS (23.6 T)

VOLUME RATING (SAE RATINGS) *

NOMINAL HEAP 19 CU. YDS. (14.5 M³)

SEMI-NOMINAL HEAP 17 CU. YDS. (13.0 M³)

STRUCK 15 CU. YDS. (11.5 M³)

DISCHARGE

MAXIMUM DUMPING CLEARANCE @ 65° 17' 4" (5.28 M)

DUMPING TIME 15 SECONDS

VEHICLE SPEEDS LOADED — LEVEL PERFORMANCE

GEAR	1	2	3	4
MPH	2.5	4.7	8.7	16.0
KM/H	4.0	7.5	14.0	25.6

BASED ON 70% CONVERTER EFFICIENCY

* This truck has a rating of 26 short (23.6 metric) tons under normal operating conditions. Volume capacity is related to tonnage capacity by the SAE semi-nominally heaped rating based on material weighing 3,000 lbs./cubic yard (1,360 kg/cubic metre).



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Branches:

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VANCOUVER • BATHURST • NAVAN, IRELAND

Specification No. JC-153/9/75

For further information contact:

(b) Equipment (Cont'd)(iii) Hauling (Cont'd)

One Euclid Dump Truck, Model 91FD-R18, with chute body, payload capacity 36,000 lbs., driven by a General Motors Series 6-71 diesel engine, two cycle, six cylinders, 4½" bore, 5" stroke, 426 cubic inch displacement, 218 h.p. at 2100 r.p.m. Engine exhaust is connected to a Gaspe type wet aluminum scrubber.

Tires - 2 front 14:00 x 25, 21 ply

- 4 rear 16:00 x 25, 24 ply

Brakes are two-shoe internal expanding air operated. Steering is by hydraulic booster cam and lever type.

Operating weight (empty) 30,800 lbs.

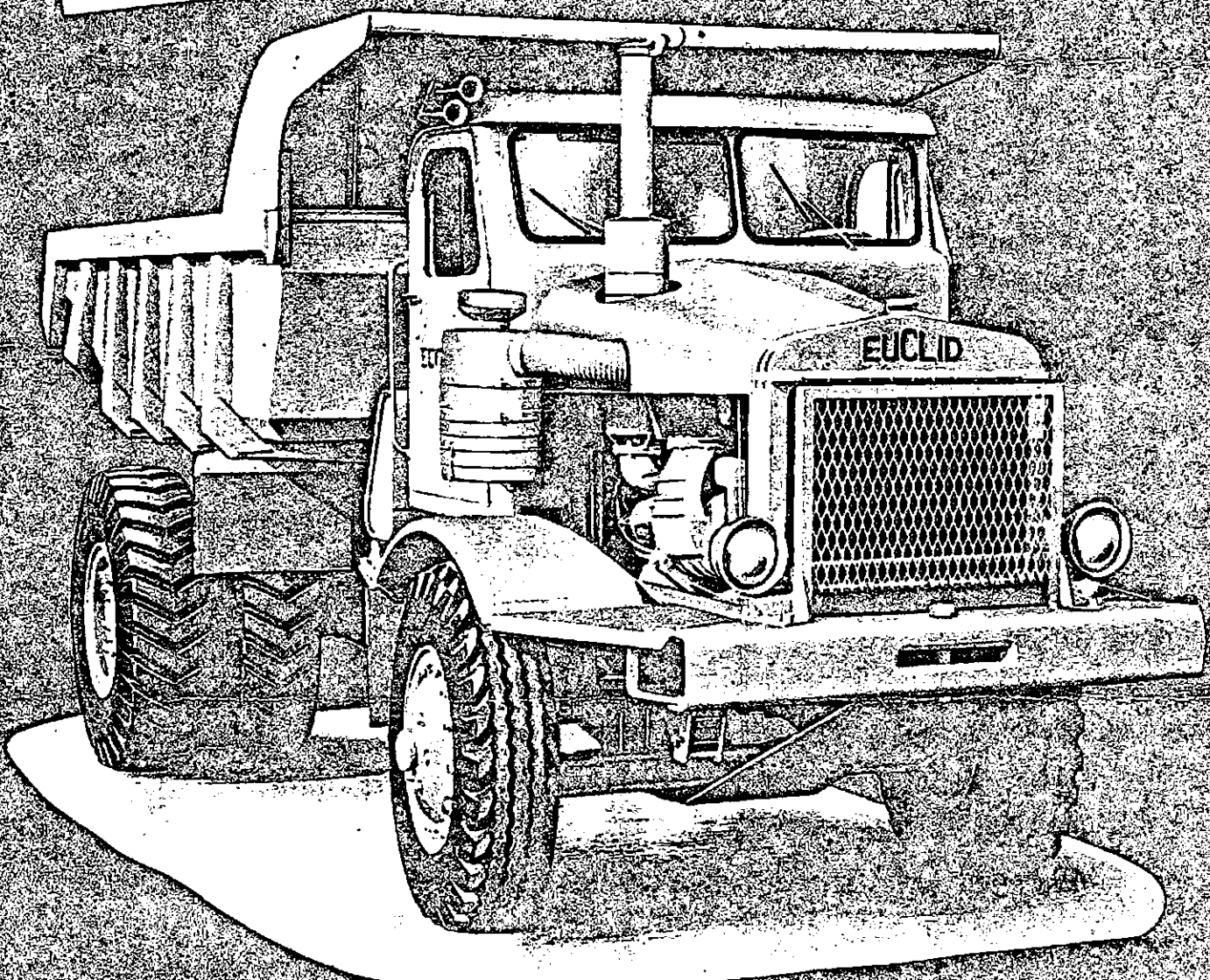
This reconditioned truck will be a secondary haulage unit for trucking development and production ore from the mine to surface.

Euclid

R-18

18-TON

REAR DUMP



BUILT FOR LOW YARDAGE COSTS

MINIMUM MAINTENANCE

Model R-18 Rear-Dump Euclid with Spring Mounted Drive Axle

Capacity:

Struck Measure 10½ cu. yd.
 Heap load at 3 to 1 slope 12½ cu. yd.
 Heap SAE rating at 1:1 slope 16½ cu. yd.
 Payload weight capacity 36,000 pounds

Engine: General Motors Series 6-71 Diesel engine; two cycle; six cylinders; bore—4¼"; stroke—5"; displacement—426 cubic inches; rated 218 horsepower at 2100 rpm; maximum torque 570 pound feet at 1600 r.p.m.

Transmission: Fuller Model 5F 1220 with 5 forward and one reverse gear selections. Speed range loaded 2.6 to 26.7.

Clutch: Lipe-Rollway single plate, spring loaded, dry disc, non-adjustable toggle type clutch with 15½" diameter woven-facings. Ball bearing type clutch release bearing. Air assist operation.

Drive Axle: Euclid Model 60 RA heavy duty, full floating, single reduction carrier and planetary reduction in each wheel; total reduction 19.66:1.

Brakes: Two-shoe internal expanding air operated brakes 17¼" x 4" x ½" on front and 20" x 6" x ¾" on drive wheels. Total brake lining area—786 square inches; total brake drum area—1188 square inches. Four-shoe 14" sector type ventilated emergency brake on driveline. Bendix-Westinghouse air compressor and controls.

Front Axle: Free-floating drop-forged, heat-treated "I" beam center section axle. Spindles are also heat-treated forgings. Cast steel wheels with demountable rims mounted on tapered roller bearings.

Cab and Seat: Seventy-seven inch width completely enclosed streamlined cab, offset to the left. Shatter-proof safety glass throughout. Euclid bucket type driver's seat with Naugahyde covered air foam rubber cushion and mounted on steel base.

Body: Euclid chute type body with inside corners rounded and sides tapered from front to rear permitting elevated body to shed load freely. All welded, sandwich type double bottom with 1½" oak planking between ¾" liner plate and ¼" bottom plate, reinforcing full length of bottom and chute.

Optional—Quarry body has flared sides 1¾" oak planking between ½" liner and ¾" bottom plate; ½" front and side plates. Struck capacity 11 cu. yds., 13 cu. yds. at 3:1 slope, 19½ cu. yds. at 1:1 slope.

Hydraulic Equipment: Euclid three-stage, double acting telescopic hoist used to raise and lower the body. Oil pressure for hoist produced by Euclid gear type packingless pump with shafts mounted on double row ball bearings.

Tire Equipment: (Tubeless)

Standard
 Front 13.00 x 25, 18 ply rating—Rear 16.00 x 25, 24 ply rating.
 Optional
 Front 12.00 x 25, 16 ply rating—Rear 14.00 x 25, 20 ply rating.

Springs: Semi-elliptic, free-floating, thrust mounted on curved spring pads for variable load centers. Front, 13 leaves, 4" wide x ½" thick. Rear, 10 leaves, 6" wide x ¾" thick.

Steering Gear: Cam and lever type with roller bearing mounted studs, bushings in trunion and anti-friction bearing in tube. Four spoke 20" diameter rim type steering wheel. Euclid hydraulic booster reduces manual steering effort.

Frame: Rigid, wide flange, fabricated beam with torque stiffeners and integral front bumper. Body supported by top flange of frame rail and fabricated cross beam.

Dimensions:

Wheelbase 12'-11"
 Length, overall 24'- 7½"
 Width, overall 10'- 2"
 Height, overall 10'- 5"
 Loading height, maximum 8'- 4"
 Clearance, front axle 1'- 3"
 Clearance under drive axle 1'- 6"
 Turning circle on front wheel track 51'- 3"
 Body length, inside top 14'- 1½"
 Body width, front 7'-11"
 Body width, rear 8'- 5"
 Dumping angle, from horizontal 68°

Weights:

	Standard	Quarry
Chassis with hoist.....	22,400 lbs.	22,400 lbs.
Body.....	8,400 lbs.	12,500 lbs.
Net weight, empty.....	30,800 lbs.	34,900 lbs.

Distribution—Net Weight:

Front axle.....	11,900 lbs.	12,700 lbs.
Drive axle.....	18,900 lbs.	22,200 lbs.

Gross weight with 36,000 lb. payload

66,800 lbs.	70,900 lbs.
-------------	-------------

Distribution—With payload:

Front axle.....	16,500 lbs.	17,700 lbs.
Drive axle.....	50,300 lbs.	53,200 lbs.

(Refer to detailed specifications for complete data)

BLACKWOOD HODGE EQUIPMENT LTD.

(Formerly Innes Equipment Limited)

10 Suntract Road
 Toronto 15, Ontario
 CH 4-2531

Court Street S. at Cornwall
 Port Arthur, Ontario

EUCLID DIVISION

General Motors Corporation • CLEVELAND 17, OHIO



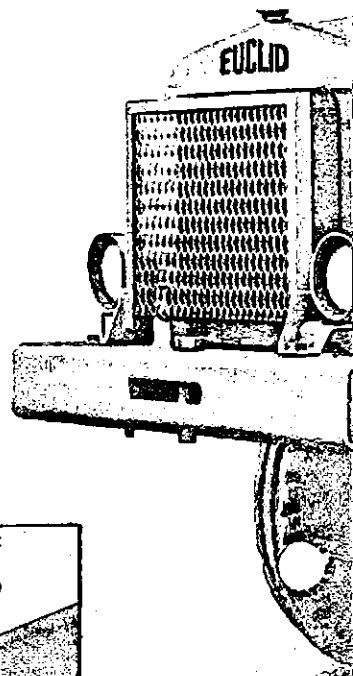
A NEW HIGH IN HAULAGE

18-Ton "Eucs" have been the Choice for "Off-the-Highway" work in Mines, Quarries and Heavy Construction FOR OVER 20 YEARS!

36,000 LB. PAYLOAD
(16.00 x 25 TIRES)
30,000 LB. PAYLOAD
(14.00 x 24 TIRES)

218 HP
STANDARD
OR QUARRY
BODY

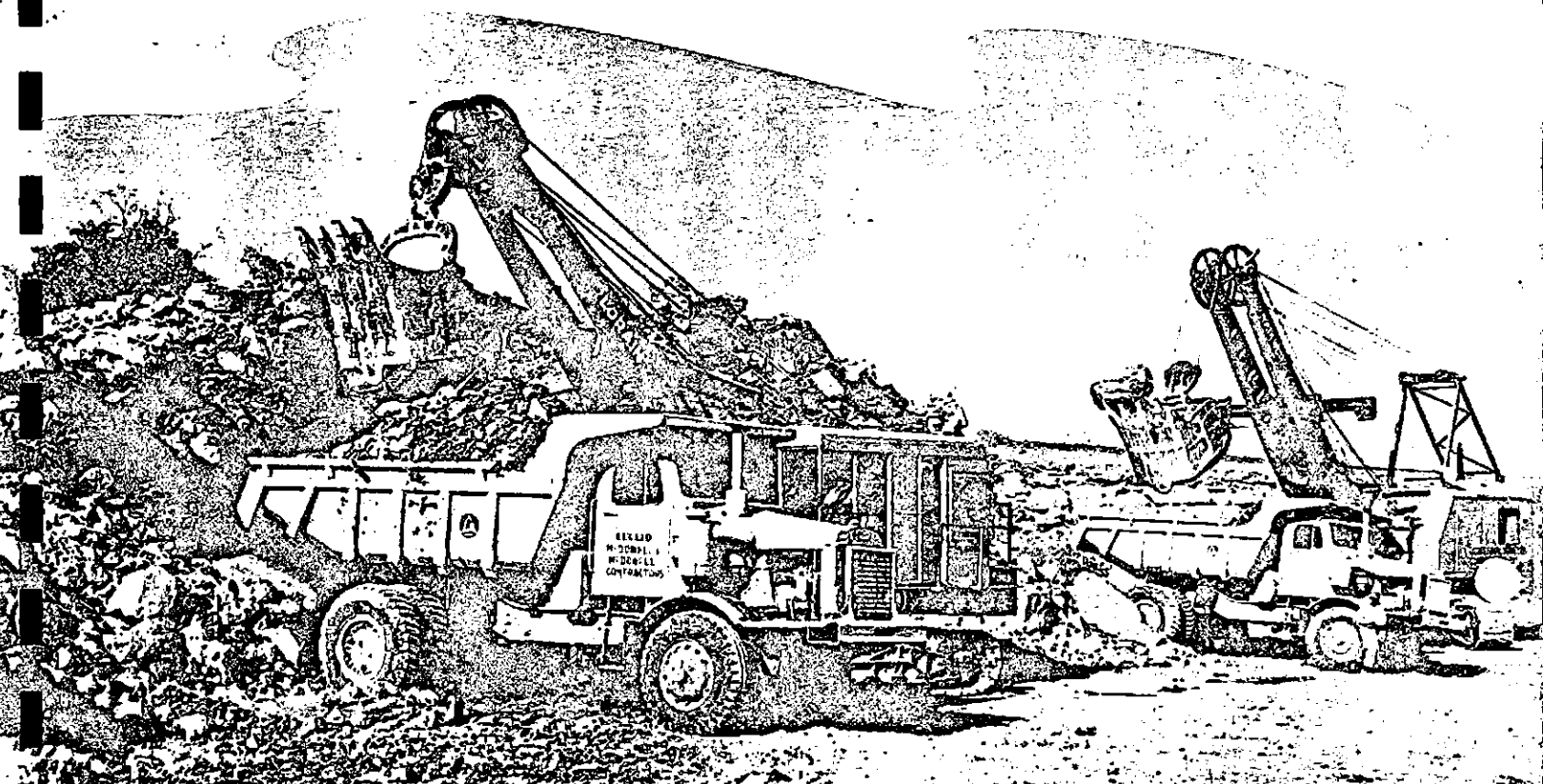
LOADED
SPEEDS FROM
2.6 TO 26.7 MPH



PERFORMANCE DATA

The chart below shows the maximum speeds with 16.00 x 25 tires and the maximum grade ability for each forward gear. For more complete data, if needed for hauling production estimates, refer to detailed specifications that are available upon request.

	5th GEAR	4th GEAR	3rd GEAR	2nd GEAR	1st GEAR
SPEED	26.7 mph	17.0 mph	9.7 mph	5.1 mph	2.6 mph
MAXIMUM GRADE LOADED	1.7% GRADE	3.8% GRADE	8.3% GRADE	17.6% GRADE	36.4% GRADE



(b) Equipment (Cont'd)

NOTE - All diesel equipment is fitted with 12 volt electrical systems, air starters and hydraulic booster steering or hydraulic stick steering for bi-directional equipment.

All equipment will be serviced at a fuel depot on surface in the vicinity of the surface shop and maintenance and repairs will be completed in the surface shop and garage.

All diesel equipment is fitted with oxy-catalyst exhaust systems, or P.T.X scrubbers with a Gaspé type water scrubber on the Euclid truck and with exhaust cooler-diluter units on engine exhausts for the elimination of noxious gases and the reduction in temperature of exhaust gases.

*Automatic fire suppression
on diesel equipment?*

MINING(c) Development and Production(i) Control Factors(aa) Work Period - 5 days/week (Monday thru Friday)

2 shifts/day - day shift

8:00 A.M. - 4:00 P.M.

- night shift

7:00 P.M. - 3:00 A.M.

Statutory Holidays - 9 per year.

New Years Day

Good Friday

Victoria Day

Dominion Day

Labour Day

Thanksgiving Day

Remembrance Day

Christmas Day

Boxing Day

52 weeks x 5 days = 260 - 9 = 251 days = 502 shifts.(bb) Mill Production Rate

365 days/year less 16 days shutdown for repair

= 349 days/year (95.6% operating time)

less 9 statutory holidays = 340 days operating.

500 tons/day @ 5% moisture = 525 wet tons/day

500 x 340 = 170,000 dry tons/year

525 x 340 = 178,500 wet tons/year

(cc) Mine Production Rate

170,000 dry tons/year/251 mine days.

= 677 dry tons/mine working day.

= 340 dry tons/mine working shift.

MINING(c) Development and Production (Cont'd)(ii) Details(aa) Drill and Blast

Headings 12 feet high by 20 feet wide drilled to a depth of 12 feet with a resulting break and net advance of 10 feet will achieve underground development and production. An equivalent or greater amount of ore will be broken in the stope rooms by slashing after the primary heading has been advanced.

Heading $12' \times 20' \times 10' = \frac{2400}{13}$ cu. ft. = 185 tons ore.

At 677 tons/mine working day a conservative program would be 4 rounds per day (2 on days, 2 on nights)
 $4 \times 185 = 740$ tons/day.

As contingencies - allow only 95% in ore = 700 tons/day.

- allow only 95% attendance :
 by mining crews = 670 tons/day.

Optimistic?

We estimate a miner-driller operating an MJM-21 Jumbo, with a helper for supplying steel, installing pipe and ventilation tube and loading holes, will be able to drill and blast in excess of two headings per shift providing sufficient working faces are made available.

(bb) Load

The loading and clean up of broken ore from two separate headings can be handled readily in an 8 hour shift with 6 hours of actual operation by a JS - 500 Jarco Scoop. The operator will always have a full heading to work on while the opposite heading is being drilled off. Also the operator will have time to use the Scoop to help with pipe and ventube installation.

(cc) Haul

Ore haulage using the J.D.T. 426 End Dump Truck is calculated on an average haul distance, one way, of 1500 feet. The semi-nominal heap capacity of the truck is 17 cu. yds. and the semi-nominal heap capacity of the Scoop bucket is 5 cu. yds. Thus $\frac{17}{5} = 4$ Scoop buckets per truck load.

The load - travel - dump - return haul cycle is calculated as follows: -

	<u>Minutes</u>
Load -	4
Travel - loaded, up grade in gear 1 at 2.5 m.p.h.	7
Dump -	1
Return - empty, downgrade 1,500 feet in gear 3 at 8.7 m.p.h.	2
Position for load	<u>1</u>
Total Cycle	15

This is equal to 4 round trips per hour. We will use 6 operating hours in an 8 hour shift = 24 trips/shift.

$$= \frac{24 \times 17 \times 27}{21.6666} = 508 \text{ tons per shift.}$$

$$= 1000 \text{ tons per day.}$$

(dd) Labour

Labour involved per shift:-

- 1 Shift Boss
- 1 Jumbo Miner-Driller
- 1 Jumbo Helper
- 1 Jarco Scoop Operator
- 1 Jarco Truck Operator
- 1 Spare Miner-Driller
- 1 Mechanic (surface/
underground)
- 1 Spare Operator

There would be additional supervision and mechanics on surface on day shift for maintenance and repairs.

MINING

(d) Ventilation

Proposed ventilating equipment will consist of:-

- 1 - 50,000 c.f.m. vane axial fan with variable - pitch blades.
- 1 - 5,000,000 b.t.u./hour direct fired propane heater.
- 1 - 20,000 c.f.m. axivane booster fan.
- 1 - 10,000 c.f.m. axial flow aerofoil booster fan.
- 42 inch diameter ventilation duct for main ventilating air distribution
- 24 inch diameter ventilation duct for secondary ventilating air distribution.

The basic ventilation requirements are:-

A basic supply of ventilating air to development or stope headings with a velocity of 25 f.p.m. Two working areas operating at the same time would require $2 \times 12 \times 20 \times 25 = 12,000$ c.f.m.

Stopes will be flashed & therefore more area than 12 x 20

Diesel equipment requirements	H.P.
1 - MJM 21 Jumbo drill rig (This diesel engine will operate part time only) (The alternate G-D Henderson Jumbos are pneumatic)	78
1 - JS-500 Jarco Scoop	180
1 - JDT-426 Truck (The alternate 9LFD-R18 truck is 218 h.p.)	270

Utilisation Factor - 66% $\frac{528}{75} = 350$ h.p.

100% not for 4 hours/day all machinery would work all the time = 528

*allow 100 cfm per Tpd running rate
675 Tpd - running rate
 $\frac{528}{75}$*

*39600 cfm
 $\frac{67500}{107100}$ cfm for Tpd of bare minimum*

Ventilation requirement 350×100 c.f.m. = 35,000 c.f.m.

Total ventilation requirement

say $\frac{47,000}{50,000}$ c.f.m.

Average c.f.m./ton mined = $\frac{50,000}{677} = 74$

Average c.f.m./man-day shift = $\frac{50,000}{10} = 5,000$

* say 75 cfm required / ft² diesel assume all diesel
operates simultaneously then $75 \times 528 = 39,600$ cfm.
designed for 50,000 cfm (ok)

50,000 cfm does not provide much access and the
drilling should be driven as soon as
possible - no more than 220' from portal.

MINING(e) Pumping

In 1963 Phelps Dodge Corporation of Canada Limited engaged H.Q. Golder and Associates Ltd., Consulting Civil Engineers from Toronto, to aid them in assessing the magnitude of the water seepage problem that might be encountered in the possible mine workings at the Silvermine property in Nova Scotia. They were advised the mine workings would be located in the lower 30 to 50 feet of a sandstone formation which is up to 200 or more feet in thickness.

In order to estimate the rate at which groundwater might flow into the mine workings it was necessary to determine the permeability of the sandstone formation and the general groundwater conditions in the area. These determinations were accomplished by completing a laboratory test on a sample of the sandstone in the form of a diamond drill core from diamond drill hole No. 61 at a depth of 209 feet, and from results of simple pumping tests and pressure packer tests conducted in the field on certain diamond drill holes. *where?*

The results of the laboratory test on the drill core are summarized in the following table. In addition they noted the sandstone was fine-grained, composed predominantly of quartz, and reacted weakly with hydrochloric acid, indicating a calcareous binder. It was reported there were no visible indications of bedding planes or stratification.

61 is located
some 400' to the
east of the mine
(west) zone

A

(e) Pumping -(Cont'd)

SUMMARY OF PROPERTIES OF SANDSTONE

DDH61 Depth 209'

Air Dry Bulk Density	144.8 lb/cu.ft.
Oven Dry Bulk Density	144.2 lb/cu.ft.
Saturated Bulk Density	151.6 lb/cu.ft.
Specific Gravity of Solids	2.71
Void Ratio	0.135
Porosity	11.9%
Coefficient of Permeability	1 to 3 x 10 ⁻⁶ cm/sec.

It was noted by Golder the permeability values are considered to be very low and are analagous to permeability expected in a silty clay. Thus it may be assumed for practical purposes, that the sandstone itself is impervious. Also it is noted that a spot check of core from various parts of the property indicated the sandstone is uniform and it would not be unreasonable to consider the sample which was tested as typical.

The results of the two types of field tests carried out indicated

- (i) The general patterns of flow or take in all holes are similar.
- (ii) The zones of high take and higher permeability are relatively narrow and tend to be separated by relatively thicker zones of lower take and lower permeability.

In conclusion Golder used the results of all tests to compute a maximum overall average coefficient of permeability of 3 x 10⁻⁴ centimeters per second and this governed his estimates of the upper limit of seepage which might be expected to occur in open stopes at various levels. These estimates covered a very wide range from 0.2 to 3.4 gallons per minute per foot

*is this valid?
According to
Denny Murray
Yes OK →*

(e) Pumping - (Cont'd).

width of open stope depending mainly on general groundwater conditions and slightly on the depth of the workings.

The access to the rock portal and decline adit will be by roadway through an open cut in overburden at a gradient of minus 7° or 12.25%. Eventually this decline entrance may be protected with a large culvert of 18 foot diameter, but in the interests of speed and necessity to establish a start on underground work before cold weather, the open cut procedure will be in effect for this year. Such an open draw on surface will funnel a certain amount of rainfall and run off toward the portal. Thus a sump with a capacity in ~~excess of 100,000~~ ^{157,500 gallons} Imperial gallons (50' x 50' x 10') will be established at the side of the roadway outside the portal as a catch basin, and a submerged pump will discharge the water to the settling ponds. *ample*

Depending on the amount of seepage and groundwater encountered, a small sump in the foot wall, west of the decline will be established 200 or 300 feet inside the portal. The pumps in this sump will discharge up the main discharge water line to the settling ponds.

The main sump will be established near the end of the first leg of the main decline and will have a capacity, depending on water conditions, of ^{in excess of} 150,000 Imperial gallons (60' x 60' x 10') with an access ramp for clean out. *ok*

226,800 gal

(e) Pumping - (Cont'd)

Since underground water conditions can be classed as unknown, provision has been made for pumping 1000 U.S. gallons per minute against a total head of 175 feet, with a similar standby pumping unit on hand. This water will be discharged through a 6 inch discharge water line to the portal and thence to the settling ponds.

VI

CONCENTRATING

- (a) Metallurgical Investigations
- (b) Concentrator Equipment
- (c) Proposed Alterations to Concentrator
- (d) Tailings Disposal

CONCENTRATING(a) Metallurgical Investigations

Metallurgical test work on samples supplied by Phelps Dodge Corporation of Canada Limited from diamond drill core intersections and a 50 ton bulk sample from a surface pit was conducted by Ontario Research Foundation in 1963 and 1964, and by Lakefield Research of Canada Limited in 1974.

Samples from diamond drill core intersections of ore secured during the close spaced drill programme on the West Zone completed in May through July, 1977, were tested by the Atlantic Industrial Research Institute at Nova Scotia Technical College, Halifax under the direction of Mr. M.A.K. Grice.

Results of all this test work indicated no problems should be encountered in producing a marketable lead concentrate assaying 70% or better Pb, 0.6 oz. Ag per ton and 2% Zn with recoveries of 95% of the lead.

The estimated optimum grind was 65% minus 200 mesh.

Reports on the results of these metallurgical investigations follow.

Ontario Research Foundation

43 QUEEN'S PARK CRESCENT EAST

TORONTO 5, CANADA

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Department of Engineering & Metallurgy

Final Report of Investigation

No. O-63338

Last copy

Flotation Tests

Phelps Dodge Corporation
of Canada Ltd.*0228/63*

J. R. JANES

October 28th, 1963

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Final Report of Investigation
No. O-63338

Flotation Tests

Phelps Dodge Corporation
of Canada Ltd.

1. INTRODUCTION

On July 25th, 1963, about 120 lbs. of assay rejects (-3/8") from samples of the Phelps Dodge Corporation's Cape Breton Island lead deposit were submitted for ore dressing tests. These samples (see list in procedures of this report) were combined into one sample designated as P53 Bench Test No. 1.

In view of the grade (3.51% Pb) and the size of the galena (from 150 to 270 mesh - Mines Branch Report IR63-12) gravity concentration was dispensed with and only flotation investigated. All laboratory flotation tests were conducted in a Fagergren cell. These took the ore through conventional roughing, cleaning and recleaning steps. At the conclusion of the investigation, a proposed flowsheet was drawn up showing one method of treating this ore. Progress reports were issued on September 6th and September 24th, 1963. The following is the final report of this investigation.

- 2 -

2. CONCLUSIONS

Preliminary flotation testing of sample P53 Bench No. 1 indicates that a marketable lead concentrate assaying 70.9% Pb, .6 oz. Ag (est.) and 2.25% Zn can be produced. Indicated recoveries are 89.3% of the lead and 81.8% of the zinc. A flowsheet consisting of crushing to -10 mesh, grinding to 74% -200 mesh, rougher and cleaner flotation, regrinding of the cleaner middlings and tailings followed by recleaner flotation is proposed. Recleaner products will then be fed back into the circuit, the concentrate into the cleaner cells and the tailings into the roughing cells. Reagent costs are estimated at a total of \$.08 per ton of crude ore.

3. SUMMARY OF TEST RESULTS

(1) Head Analyses

Pb	3.51%
Zn	.21%
Total Iron	2.55%
S	1.70%
C	1.50%
Ba	.05%

(2) Rougher Flotation Grind: 74% -200 mesh

(3) Rougher Concentrate (Average tests 1, 2, 3)

9.5% Weight	
37.1% Pb	94.1% Recovery
2.03% Zn	87.7% Recovery

(4) Cleaner No. 1 Pb Concentrate (Recleaner Test 2 data)

3.27% Weight	
70.9% Pb	62.6% Recovery
2.25% Zn	31.7% Recovery

(5) Recleaner Combined Concentrate (Recleaner Test 2 data)

3.06% Weight	
30.07% Pb	25.8% Recovery
3.24% Zn	48.34% Recovery

Note: A fraction of the cleaner products was unavailable for regrinding and recleaning. Inclusion of these fractions will raise these recovery figures to 26.8% of the lead and 50.14% of the zinc and the weight figure to 3.27%.

3. SUMMARY OF TEST RESULTS - cont'd

(6) Final Combined Tailings (Rougher and Recleaner)

93.46% Weight	
.42% Pb	10.7% Loss
.04% Zn	18.16% Loss

(7) Indicated Recovery

89.3% of the lead
81.8% of the zinc

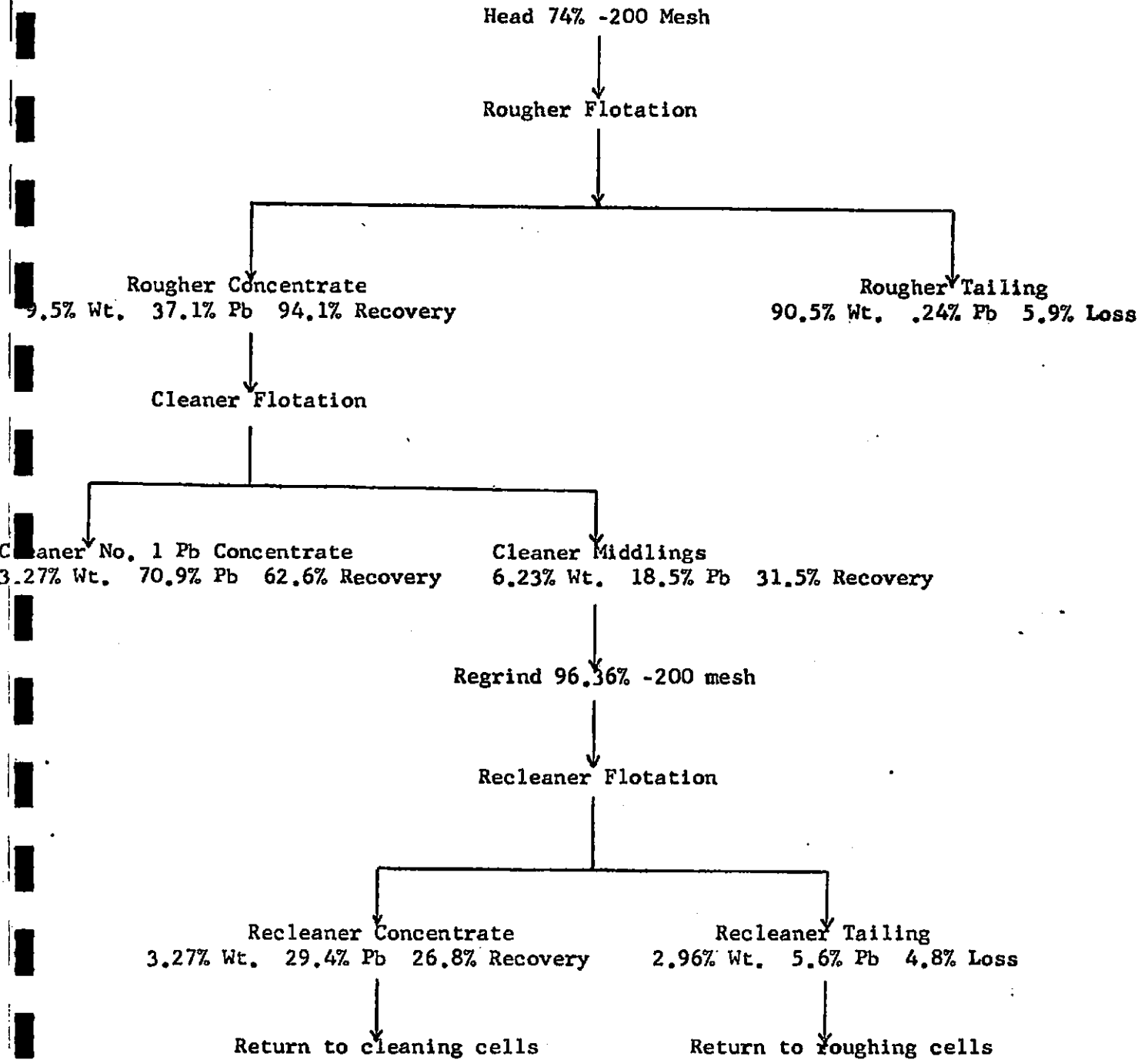
(8) Reagent Costs

Roughing Stage	\$.066 per ton crude
Cleaning Stage	.0086 per ton crude
Recleaning Stage	.0094 per ton crude
Total Cost	\$.084 per ton crude ore

(9) Silver assay test 43, Pb Concentrates 1, 2 and 3

.6 oz. per ton.

4. PROPOSED FLOWSHEET



(Note: Unavailable middling fraction from cleaner flotation included in these results)

5. COMMENTS(a) Recovery

The overall recovery (Recleaner Test 2) is indicated to be 89.3% of the lead and 81.8% of the zinc. Attempts to improve upon this recovery either by withdrawing more froth in the roughing stage or finer grinding before roughing resulted in only a minor improvement in recovery (i.e. from 94.9% to 95.8%).

The final tailing loss of 10.7% of the lead and 18.16% of the zinc is probably a maximum since the recleaner tailings would be returned to the roughing stage or subjected to a scavenging float (i.e. more removed in the recleaner concentrate).

The two recleaner tests show that the size of regrinding has an important effect on the grade and recovery. Test No. 1 - 98.66% -200 mesh (90.8% -325 mesh) was much too fine. Sliming of the lead and gangue consumed reagents and yielded a high recleaner tailing loss (9.56% of the lead). Test No. 2 96.34% -200 mesh (78.54% -325) gave a 4.8% lead loss. Further investigation of this regrind, recleaning stage is warranted and probably will provide improved recoveries.

(b) Reagent Costs

Total estimated reagent costs of \$.084 per ton crude ore are quite low. Further reduction of the reagents in the recleaning stage is possible but this will not materially lower the overall cost. It may be advantageous to increase slightly some of the reagent dosages.

(c) Proposed Flowsheet

The proposed flowsheet shown is only a suggestion. Other possibilities of routing the recleaner products occur and these will doubtless be investigated at a later date. Scavenging of the cleaner tailings was not successful. Therefore, these were reground with the cleaner middlings (as one product) since the tonnages treated probably do not warrant two separate regrind circuits. Recleaner tailings are then returned to the roughing cells for a scavenging float. The recleaner concentrate must be refloated and some of it probably reground. Therefore, these are shown as returning to the cleaning cells.

Investigation No. O-63338

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5. COMMENTS - cont'd(d) Zinc Concentration

Cleaner Test 43 was an attempt at producing a zinc concentrate after removal of the lead. This test was not successful in producing a good zinc concentrate due probably to (a) the fineness of the sphalerite and the nature of its deposition (i.e. finely intermixed with galena and pyrite, replacement of woody structures) - see Mines Branch Report IR63-12; (b) insufficient depression of the zinc during lead flotation. The No. 1 and 2 lead concentrates combined assay 75% Pb, 1.27% Zn and contain 12.46% of the original zinc. This zinc content is believed to be the result of intermixed sphalerite and galena, since sufficient cyanide (.15 lbs per ton) was added to depress the sphalerite. The No. 3 and 4 lead concentrates combined assay 48.39% Pb and 3.78% Zn. Here insufficient zinc depression results in a high zinc content of 61.4% of the original zinc. More cyanide should have been added between lead floats 2 and 3. It should be noted though that too much cyanide will depress the galena also. The No. 1 zinc concentrate assays 20.5% Pb and 5.35% Zn. From the test data and the ore photomicrographs (Mines Branch Report IR63-12) some of the galena will undoubtedly accompany the sphalerite in any zinc concentrate. Some zinc will also occur in the lead concentrate (Pb #1 Conc. test 43, 81.2% Pb, 1.04% Zn). There probably is an in between stage where a zinc concentrate can be produced and further testing is advised.

6. RECOMMENDATIONS

A larger sample (500 lbs) should be tested in a simulated pilot plant circuit to more fully examine:

- (a) flotation variables (i.e. pH, pulp density, grinds, reagent type and dosage)
- (b) proper regrind size for recleaning
- (c) the effect on final lead concentrate grade and recovery of returning the recleaner products to the circuit
- (d) production of a separate zinc concentrate.


6. RECOMMENDATIONS - cont'd

At this time, it may also be advantageous to do a grind liberation study together with polished sections of the various flotation products. In lieu of this, the sections used in the Mines Branch Report IR 63-12 should be carefully examined and the liberation sizes of the fine galena and sphalerite determined.

The above is suggested as an interim to a larger pilot plant test because it is more flexible and yet will give accurate results. Failing this, it is suggested that a pilot plant study of several tons be carried out.

7. SAMPLE DISPOSAL

All of sample P53 Bench No. 1 has been used in this testwork. Test samples will be retained at the pilot plant for a period of six months pending further notice.



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October 28th, 1963.

JRJ:mcd:rh

Ontario Research Foundation

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43 QUEEN'S PARK CRESCENT EAST

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Department of Engineering & Metallurgy

Report of Investigation

No. O-63348

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Flotation Tests

Jan 22/64 Bench Test No. 2

Phelps Dodge Corporation
of Canada Ltd.

J. R. JANES

January 22, 1964

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Flotation Tests

Phelps Dodge Corporation
of Canada Ltd.

1. INTRODUCTION

On November 20, 1963, 4 boxes of 2" diameter drill core weighing 600 lbs. were received from the Cape Breton Island lead property of the Phelps Dodge Corporation of Canada. This was designated sample Bench Test No. 2. After crushing to minus 1 inch, a representative 150 lb. sample was used in the present investigation. In this test work a reground middling fraction was subjected to recleaner flotation. The tailings from this recleaner were then mixed with fresh feed and given a rougher float as proposed in the flowsheet of Report No. 63338. The final tailing loss was then calculated.

2. SUMMARY OF TEST RESULTS

(1) Head Analysis

2.86% Pb 0.06% Zn

(2) Combined Lead Concentrate

70.29% Pb 3.25% Wt 82.39% Recovery

(3) Over-all lead recovery indicated to be 97.6% assuming only tailing loss is to the roughing cells and all lead eventually recoverable in the middlings. ✓

(4) Final Tailing loss

91.74% Wt 0.08% Pb 2.4% Loss

(5) Estimated Reagent Costs \$.08 per ton

3. SUMMARY TEST RESULTS
FLWSHEET

Head 2.86% Pb - 57.4% - 200

Rougher Flotation

Rougher Concentrate
6.51% Wt. 42.8% Pb 97.5% Recovery

Rougher Tail
93.49% Wt .08 Pb 2.5% Loss

Cleaner Flotation

Cleaner Concentrate
3.48% Wt 58.1% Pb 72.31% Recovery

Cleaner Middlings
3.03% Wt 23.1% Pb 25.19% Recovery

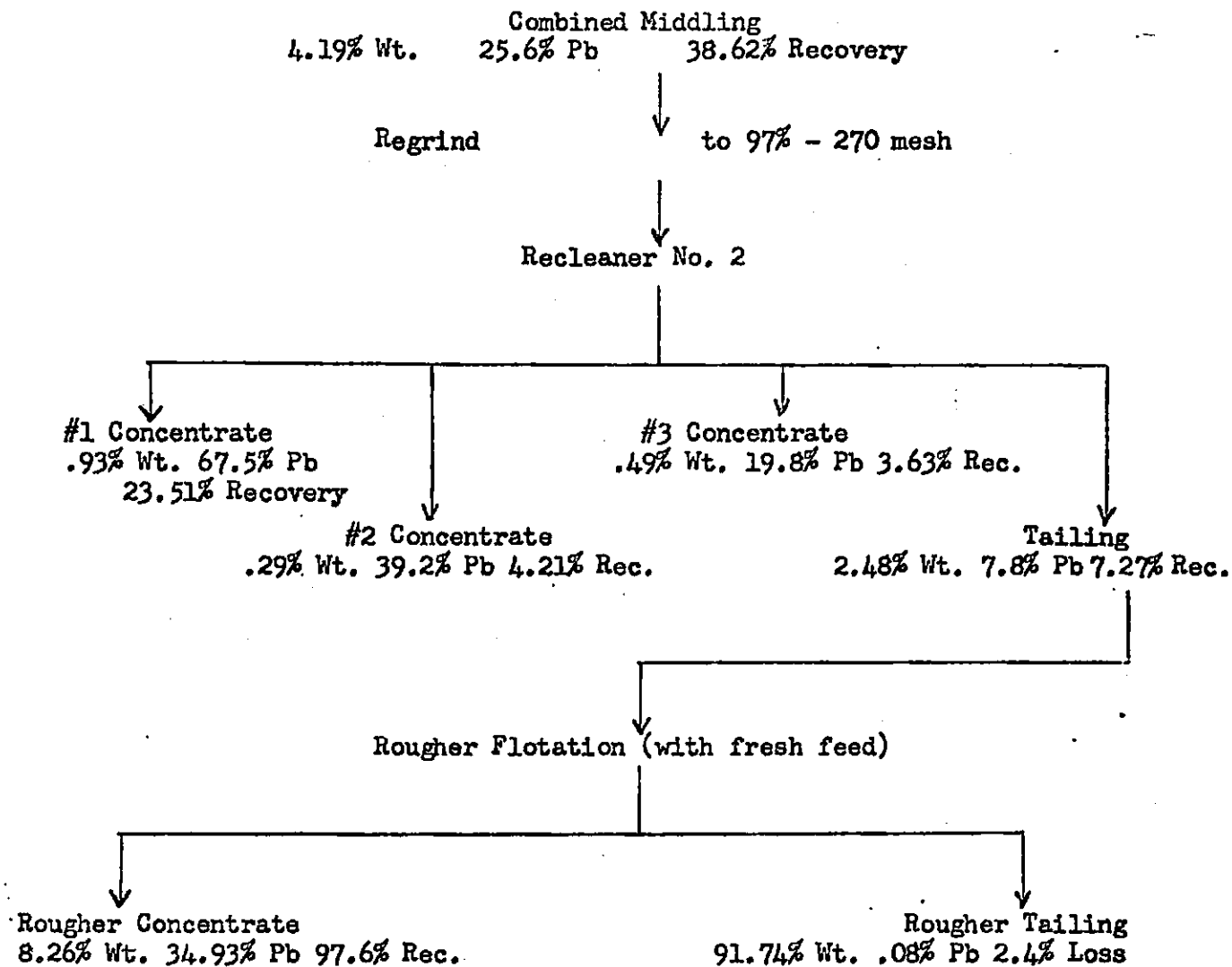
Recleaner #1

Recleaner Concentrate #1 Pb Concentrate
2.32% Wt 71.4% Pb 58.88% Recovery

Recleaner Middling
1.16% Wt 32.6% Pb 13.43% Recovery

Combined Middling
4.19% Wt 25.6% Pb 38.62

(Flowsheet cont'd)



4. COMMENT(a) Sample

Bench Test No. 1 sample (Report 63338) assayed 3.51% Pb and 0.21% Zn. The present sample, Bench Test No. 2, was much lower in grade; 2.86% Pb and 0.06% Zn. This difference produced reductions in the weight recoveries of concentrates (i.e. Rougher Concentrate from 9.5% Wt. to 6.5% Wt.) but did not appear to be a detriment to the over-all concentrating characteristics of the sample, i.e. No. 1 lead concentrate 70.9% Pb - Bench Test, and 70.3% Pb Bench Test 2.

(b) Grinding

Primary grinding was done dry to -100 mesh or 57.4% - 200 mesh. This is somewhat coarser than previously used (74% - 200 mesh by wet means. Initial flotation tests indicated however, better recoveries and higher grades of rougher concentrate at this coarser grind.

(c) Flotation (comment follows flowsheet order)

Rougher flotation produces a clean tailing (0.08% Pb) and a relatively high grade concentrate of 42.8% Pb. Differences in weight recoveries and product assays over Bench Test 1 results can be accounted for by the following;

- (1) Difference in head assay
- (2) Possible coarser grain size of the galena
- (3) Dry grinding to a coarser size
- (4) More effective flotation - (a) 2000 gm. Denver cell used for roughing versus a 600 gm. Fagergren cell, (b) increased reagent dosage to the recleaner cell.

- 6 -

Recleaning of the No. 1 lead concentrate was necessary due to an excess of frother (same quantity as successfully used in Bench Test 1). This step would probably not be necessary if the amount of frother was reduced.

From the cleaner and #1 recleaner flotation a combined middling of 4.19% Wt, 25.6% Pb and 38.6% recovery is obtained. Regrinding to approximately 97% - 270 mesh produces a No. 1 concentrate of 67.5% Pb and 23.51% recovery. The calculated combined lead concentrate is therefore 3.25% Wt, 70.29% Pb and 82.39% recovery.

Recleaning of the middlings also produces 2 high grade fractions (concs #2 and #3) which combined assay 27.0% Pb .78% Wt. and 7.84% recovery and a low grade fraction assaying 7.8% Pb, 2.48% Wt. and 7.27% recovery. The present test returned this low grade fraction to the roughing cells with fresh feed. The final tailing produced was essentially the same as originally obtained i.e. .08% Pb, 2.4% loss as versus 0.08% Pb and 2.5% loss. Weight recovery in the rougher concentrate increased from 6.51% to 8.26% and the grade of rougher concentrate decreased from 42.8% Pb to 34.9% Pb. These differences are sizeable and indicate the necessity for finer re-grinding of the middlings.

The present testing did not retreat the high grade middling fraction of 27% Pb .78% Wt. and 7.84% recovery. This must be returned directly to the regrind mill for finer grinding. The large number of batch tests required to adequately carry out this phase of the problem makes this type of study impractical.

5. CONCLUSIONS

(1) A premium grade concentrate of 70.29% Pb, 3.25% Wt and 82.39% recovery is obtainable without difficulty.

(2) Scavenging of the recleaner tails, which contain 7.27% of the lead in the Roughing cells does not reduce recovery or alter the grade of rougher tailing (.08% Pb).

(3) The indicated recovery is 97.6% assuming no further losses occur in returning the high grade middling fraction to the regrind circuit.


6. RECOMMENDATIONS

(1) A series of batch tests in this case cannot lead to a final figure for the recovery without assuming that there is only one loss and that is to the rougher tailings. Recirculating of the unliberated grains to the regrind mill is a continuous process best evaluated in a pilot plant run of several tons. The indicated recovery of 97.6% is therefore probably too high. A better figure might be 95% to account for further possible losses on retreatment of the middlings.

(2) A polished section study for mineralogy and grain size should be made of a middling composite. The required grind for adequate liberation could be determined and a better idea of the final recovery obtained.

7. SAMPLE DISPOSAL

Test products and about 450 lbs. of Bench Test No. 2 sample remain. These will be placed in storage pending further notice.



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January 22nd, 1964.

JRJ:ms

Ontario Research Foundation

Department of Engineering and Metallurgy

43 QUEEN'S PARK CRESCENT EAST

VI - 20
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Investigation No. 0-64321 2

Pilot Plant Grinding and Flotation
of a Lead Ore from Cape Breton, N.S.

Phelps-Dodge Corporation

H. Z. S.

OCT 5 1964

Sept 28/64

S. ROTHSCHILD

September 28th, 1964.

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ONTARIO RESEARCH FOUNDATION
Department of Engineering and Metallurgy

Report of Investigation No. O-64321

Pilot Plant Grinding and Flotation of a Lead Ore
from Cape Breton, N.S.

Phelps-Dodge Corporation

1. Introduction

A lead ore was submitted in the Fall of 1963 by the Phelps-Dodge Corporation from their Cape Breton, Nova Scotia property to determine its beneficiation characteristics. Project Reports O-63338 and O-63348 described the laboratory results of this study. These drill core samples, containing 3.51% lead, were upgraded by mineral flotation techniques to yield a final concentrate analysing 70.9% lead and having an indicated recovery of +95%; silver content was estimated at 0.6 oz/ton; and zinc, to the value of 2.25%. The estimated optimum grind was approximately 65% -200 mesh.

1964 | A 50 ton sample of representative pit ore was received on July 20, 1964 for further pilot plant testing. This sample graded from 5.98% to 9.40% lead in the analysis of daily test composites, as contrasted to the 3.51% lead in the earlier drill core samples.

2. Objectives

1. To confirm the laboratory batch flotation results on a continuous pilot plant processing 300 to 1000 pounds/hour of pit ore.
2. To provide data for design and cost estimation of a commercial plant based on the test prototype.
3. To utilize the Nordberg 6 foot diameter wet autogenous grinding mill circuit in reducing the feed to size. This would provide basic design and cost data in comparing autogenous vs. conventional crushing, plus ball mill grinding of this ore.

3. Conclusions

1. The pilot plant test data and the original laboratory results could not be compared directly due to the wide range of feed lead content, 3.51% vs. 5.98% - 9.40%. The zinc content of the new sample ranged from 0.11% to 0.36%, increasing with the lead content; in the older sample, from 0.06 to 0.21% zinc.

2. The conventional grinding circuit used could not provide the requisite grind of approximately 65% -200 mesh under the operating conditions. The daily test grinds ranged from 41.8% to 54.0% -200 mesh. The liberation size was estimated to be -150 to +200 mesh, the same as in the original samples.

3. The better final lead concentrates produced contained from 60.8% to 78.3% lead, with indicated recoveries of from 91.4% to 96.6%. Silver content was from 0.98 oz/ton to 1.34 oz/ton; and zinc content, 1.23% to 1.68%.

4. The minimum tailing lead content was 0.24%, with an indicated loss of 3.4% of the total lead contained. This exceeded the anticipated 0.08% lead tailing based on previous laboratory work. Whether the comparison is valid is questionable, see item 1. The lead recovery is reasonable, consistent with the coarse grind.

5. Collateral data showed the regrind of middling to -100 mesh was essential to obtain significant liberation of the galena grains.

6. The Nordberg grinding circuit did not operate continuously, due to mechanical difficulties. Only one short test was possible.

7. Subsequently, autogenous grinding tests were made without the flotation circuit. This work was reported in a letter circulated to both Nordberg Manufacturing Company and the Phelps-Dodge Corporation.

4. Comments

a) Grinding Equipment

The proper employment of the Nordberg circuit at the time of this pilot plant run was not possible due to start-up troubles involved in testing of a new type mill. The only test with the mill delivering flotation feed was

marred by the coarse grind of 48.9% -200 mesh.

The Denver Ball Mill was not flexible enough to deliver flotation feed finer than 54.0% -200 mesh since (1) the pulp density had to be maintained at approximately 25% solids, (2) the ball charge was at the maximum, (3) the classifier unit could not be readily adjusted to deliver a finer feed fraction, and (4) the feed rate was only 275 pounds/hour vs. the 400 + pounds/hour desired for the tests.

b) Grinding, General

The key to the entire problem was the initial grind of feed and that of the regrind. However, that the lead can be concentrated into a suitable product is shown by the data.

As liberation of the lead must be obtained, and a coarse grind did not provide sufficient release of lead grains to provide the desired grade of tailings, any improvement in the high level of lead recovery in the concentrate must be balanced against the cost of further grinding of the feed, and the possible lead losses due to overgrinding and sliming the lead values.

c) Reagents, General

Due to the low feed rate of the tests, the maintenance of low rates of reagent addition created a problem. The original frother employed, AEROFROTH 71, was water-insoluble. Therefore no dilution was possible with water to obtain the low dosage rate required. White diesel oil was used as a diluent. (1 part AEROFROTH 71: 9 parts of D.O.), but this "killed" some of the froth action. Therefore DOWFROTH 250 was finally substituted in a 0.5% water solution, and this proved quite satisfactory.

d) Flowsheet Modifications

1) The grind must be made finer to even have any opportunity to reduce the tailing loss.

2) Middling regrind is essential to the entire process. Circulating middlings through the feed mill is not the best for efficient grinding, but can be done in lieu of a separate middling grinding circuit.

e) De-zincing

This problem was not investigated since the zinc content of this new material was quite low, 0.11% - 0.36%, and the content of the final lead concentrate reached only 1.28% - 1.68% zinc.

f) Circuit Stability

There were pronounced surges in the flotation circuit. This was shown by the difficulty in balancing timed-samples during testing. This must be due to the inherent characteristics of the flotation cells. Long period samples would have probably provided better samples for material balances.

g) Middling Regrind and Final Tailings

The open circuit operation of the flowsheet should have provided a good opportunity for minimum lead losses to the tailing, since no middling "tailing" was available to raise the lead content. This was not the case. Perhaps the circuit balance was not controlled for sufficient time.

Assuming the final tailing was 90% "rougher" tailing and 10% middling "tailing" having assays of 0.08% and 3.0% respectively, then the final tailing would be 0.37% lead. Thus, our results must show a feed-back of these high lead "tailings". This leads back to the problem of grind of the feed.

5. Recommendations

During the completion of the laboratory work and preparation of the report on the pit ore, the Nordberg wet autogeneous grinding mill has been made operational. In addition, a pilot plant sized Hardinge regrind ball mill has been acquired. There also remains a quantity of pit ore in storage. In view of the foregoing, it would seem profitable to make a final run on this material with this supplemented circuit.

6. Sample Disposal

All the remaining pit ore sample and test samples will be retained at the pilot plant for a period of six (6) months pending further notice.

Reference: Notebook No.21, Rexdale E. & M.

Testwork done by:

J. N. Starkey
M. O. Pearce
P. Meredith
J. Biskupski

S. Rothschild

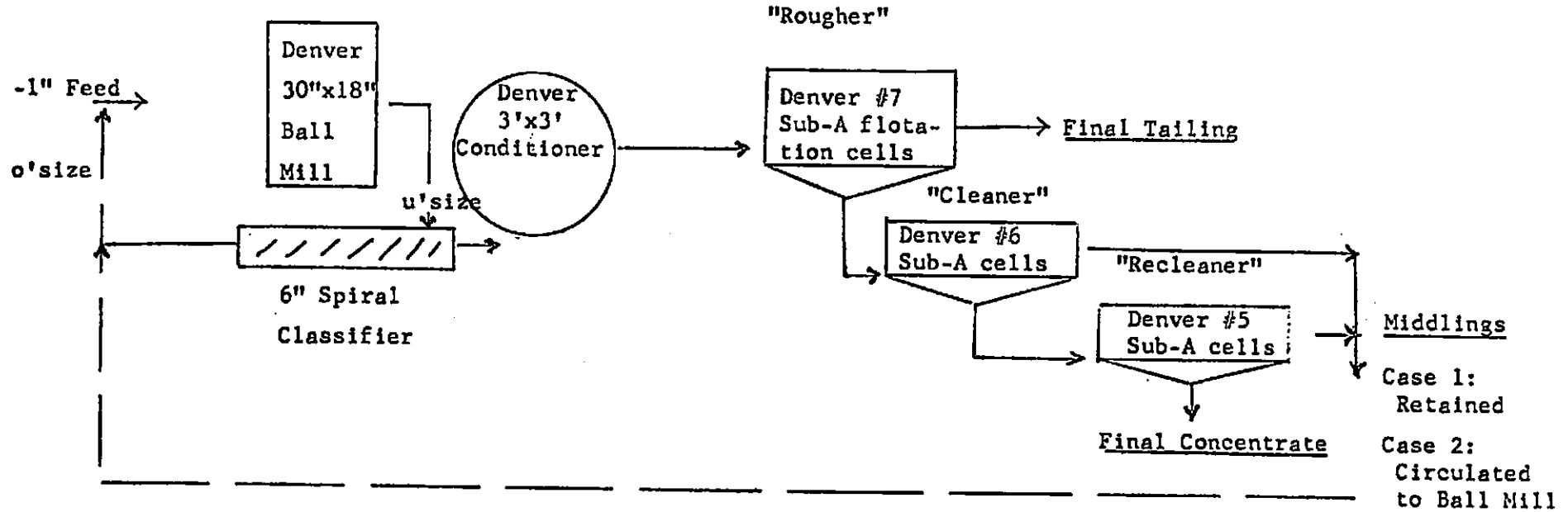
S. Rothschild,
Research Engineer,
Department of Engineering
and Metallurgy.

SR:rh

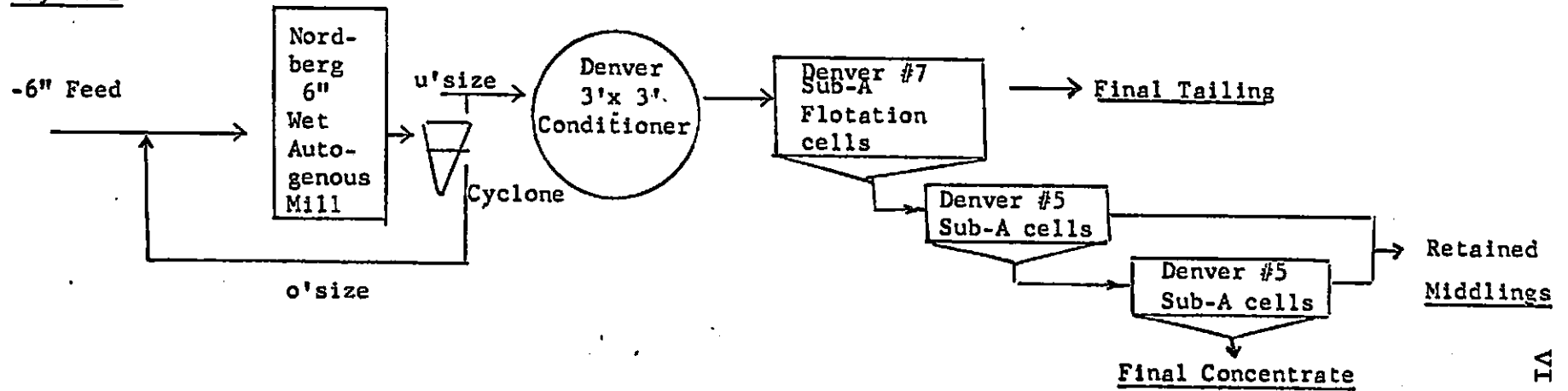
Figure 1.

Pilot Plant Flowsheets

Layout A



Layout B



An Investigation of
THE RECOVERY OF LEAD

from Yava Project samples

submitted by

PHELPS DODGE CORPORATION OF CANADA LIMITED

Progress Report No. 1

Project No. L.R. 1705

Note:

This report refers to the samples as received.

The practice of this Company in issuing reports of this nature is to require the recipient not to publish the report or any part thereof without the written consent of Lakefield Research of Canada Limited.

LAKEFIELD RESEARCH OF CANADA LIMITED

Lakefield, Ontario

May 27, 1974

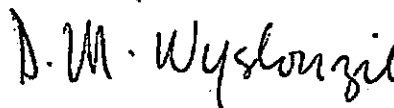
I N T R O D U C T I O N

In a letter dated March 25, 1974 Mr. D.A. Firth of Phelps Dodge Corporation of Canada Limited requested that we should undertake a program of bench tests on samples of split core from Yava Mines Limited, Cape Breton, N.S. The principal objective of the bench tests was to produce a marketable lead concentrate and to analyse this concentrate for a variety of elements on which penalties might be levied by smelters.

LAKEFIELD RESEARCH OF CANADA LIMITED



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



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

Investigation by: K.W. Sarbutt

S U M M A R YHead Sample Assays

Lead (Pb)	3.79 %	3.63 %*
Zinc (Zn)	0.21 %	
Silver (Ag)	0.05 oz/ton	

* Average head assay as calculated from test results.

Flotation

Two preliminary tests were performed to investigate the recovery of a lead concentrate. The grinding and flotation conditions, with the rougher flotation results, are summarised below.

Test No.	Primary Grind Minutes	% -200 mesh	Reagents lb/ton		Froth Time min.	Rougher Concentrate		
			R-242	Z-4		Weight %	Assay % Pb	Distribution % Pb
1	15	45	0.04	0.10	10	11.71	28.96	97.2
2	30	84	0.04	0.10	10	12.13	28.79	97.3

The rougher concentrate from Test No. 1 was reground to 98 % minus 200 mesh and cleaned three times. The lead concentrate amounted to 4.41 % by weight, and assayed 71.2 % Pb with a recovery of 90.0 % Pb.

Summary - Continued

The rougher concentrate from Test No. 2 was cleaned four times without regrinding and the lead cleaner concentrate amounted to 4.64 % by weight, and assayed 69.7 % Pb with a recovery of 90.1 % Pb.

A bulk lead concentrate was produced in a 3-stage locked cycle test (Test No. 3) using 10 kg charges. The concentrates from each cycle were combined to produce a lead concentrate assaying 73 % Pb with a recovery of 90.1 % Pb.

Lead Concentrate Analysis

The combined lead concentrate from Test No. 3 assayed as follows:

Lead	(Pb)	-	72.85 %
Zinc	(Zn)	-	2.32 %
Iron	(Fe)	-	2.58 %
Nickel	(Ni)	-	0.002 %
Copper	(Cu)	-	0.013 %
Bismuth	(Bi)	-	0.002 %
Carbon present as carbonate	(CO ₂)	-	2.04 %
Carbon not present as carbonate	(C)	-	1.37 %
Arsenic	(As)	-	0.008 %
Antimony	(Sg)	-	0.003 %
Sulphur	(S)	-	15.3 %
Silica	(SiO ₂)	-	2.75 %
Alumina	(Al ₂ O ₃)	-	0.42 %
Lime	(CaO)	-	0.32 %
Magnesia	(MgO)	-	0.12 %
Barium	(BaO)	-	0.004 %
Silver	(Ag)	-	0.86 oz/ton

Summary - Continued

Semi-quantitative Spectrographic Analysis

Element	Head	Conc.	Element	Head	Conc.
Aluminum (Al ₂ O ₃)	H	M	Manganese	.1 %	.02 %
Antimony	-	-	Magnesium (MgO)	.5 %	.1 %
Arsenic	-	-	Molybdenum	-	-
Barium	.03 %	<.01 %	Neodymium (Nd ₂ O ₃)	-	-
Beryllium (BeO)	-	-	Nickel	.001 %	<.001 %
Bismuth	-	<.001 %	Phosphorus	-	-
Boron	-	-	Silver	<.1 oz/t	1 oz/t
Calcium (CaO)	1 %	.1 %	Silicon (SiO ₂)	H	H
Cadmium	-	.01 %	Sodium (Na ₂ O)	.1 %	-
Cerium (CeO ₂)	-	-	Strontium	-	-
Chromium	.01 %	-	Tantalum (Ta ₂ O ₅)	-	-
Cobalt	-	-	Thorium (ThO ₂)	-	-
Columbium (Cb ₂ O ₅)	-	-	Tin	-	-
Copper	.005 %	.01 %	Titanium	.2 %	.01 %
Gallium	-	-	Tungsten	-	-
Germanium	-	-	Uranium (U ₃ O ₈)	-	-
Iron (Fe)	1 %	1 %	Vanadium	<.01 %	-
Lanthanum (La ₂ O ₃)	-	-	Yttrium (Y ₂ O ₃)	-	-
Lead	M	H	Zinc	.3 %	M
Lithium (Li ₂ O)	-	-	Zirconium (ZrO ₂)	<.01 %	-

Figures are approximate:

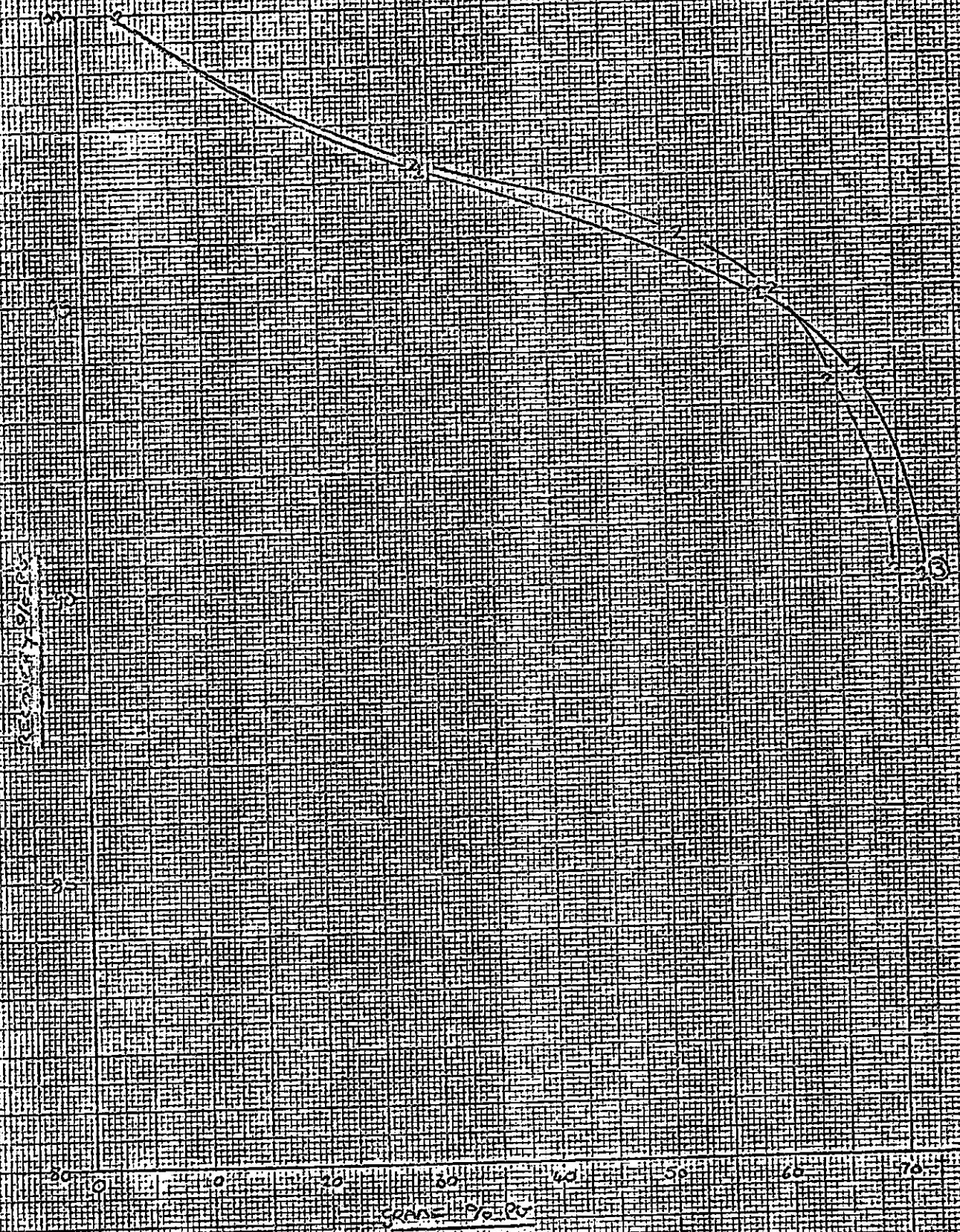
CODE:

H - High - 10 - 100 % approx.
M - Medium - 1 - 10 % approx.
L - Low - .1 - 1 % approx.

- Not detected - Elements looked for but not found
X Not looked for
< Less Than

PU GRADE - RECOVERY CURVES

VI - 31 (a)



ENGINEERING CONSULTANTS
MADE IN CANADA
SQUARE INCH TO 10 MM
G-N

METALLURGICAL TEST RESULTS ON A SAMPLE OF ORE FROM
THE SILVERMINES PROPERTY OF YAVA MINES

M. A. K. Grice
M. A. K. Grice, P. Eng.

3rd Oct. 1977
Date

Testwork

The metallurgical tests were carried out at Nova Scotia Technical College in August 1977.

Origin of Sample Used

A composite sample was made up by combining half the drill core reject from the following holes if the section in question assayed greater than 4% Lead.
Hole No. 278A, 290, 291, 292, 293, 296, 297, 298, 299.

Test Equipment Used

All flotation tests were done using Denver Laboratory Cells. The large unit for Roughing and Scavenging and the small unit for cleaning.

1.0 SUMMARY

1.1 Conclusions

1.11 A good recovery can be obtained by a Primary Grind to around 70 mesh.

1.12 A concentrate regrind to -200 mesh is necessary to give a high grade of concentrate

1.13 It may be possible to prepare a glass sand from the coarse fraction of the Primary Tailing.

1.2 Comments

1.21 Froth Conditions.

The froth was rather sticky, this can be attributed to the Aerofloat 31. In future tests the amount of this reagent will be reduced and the Xenthate increased.

1.22 Grinding Condition

Further tests are required to optimize as course a primary grind as possible.

2.0 Original Flotation Test

The object of this test was to get an idea of how coarse a grind could be attempted and yet hold the recovery.

A grab sample of ore was screened on a 20 mesh screen and the oversize crushed to -20 in the Shatterbox.

The flotation reagents selected were as per Ontario Research Foundation with quantities double because of the higher head grade.

2.1 The following table shows the ore crushed into definite sand and slime fraction.

Mesh	Feed		Tail		Conc.			Estimated Recovery
	% Wt.	% Pb	% Wt.	% Pb	% Wt.	% Pb	% Zn	
30	4.5	5.56	3.4	5.25	-	-	-	0
40	4.3	4.25	2.8	4.67	-	-	-	0
50	8.0	3.56	6.3	2.88	3.7	23.75	0.59	19
70	17.8	2.49	16.0	0.69	12.4	22.50	0.48	83
100	27.9	2.52	30.7	0.11	21.6	26.25	0.47	96
140	11.3	4.50	10.8	0.06	12.3	47.50	0.60	98
200	7.8	7.50	7.4	0.03	11.4	60.59	0.85	99
-200	19.8	5.50	22.6	0.25	38.6	73.33	1.03	95
	-	4.04	-	0.70	-	50.39	-	

	<u>% Pb</u>	<u>Analysis</u>	<u>% Zn</u>
Scavenger Concentrate	6.18		0.10
Cleaner Tail	22.50		0.47

The estimated recovery is $\frac{\text{Analysis of Tail Fraction}}{\text{Analysis of Feed Fraction}} \times 100$

Due to the large tailing weight this is a reasonable approximation.

2.2 Recovery becomes reasonable at around 80[#].

2.3 A regrind on the +200[#] concentrate is needed due to locked particles.

3.0 Four Stage Lock Test

Grinding

Conditioning

Original Grind

1000 gms / 7 rods / 15 min. / 66% Solids

Wet screened on 70 mesh screen

Size Analysis of +70[#] Fraction. The wet screening efficiency was very low - hence the large amount of -70[#].

<u>Mesh</u>	<u>% Wt.</u>	<u>% Lead</u>
+70	7.1	2.44
-70 +80	21.4	0.55
-80 +100	30.4	1.08
-100 +120	16.1	2.69
-120 +200	19.2	6.05
-200	5.8	<u>13.00</u>
	Calc Head	<u>2.96</u>

The plus 70[#] was reground with 7 rods / 15 min / approx. 60% solids (visual estimate of +70[#], 50% of total weight).

Size Analysis after regrind.

<u>Mesh</u>	<u>% Wt.</u>	<u>% Lead</u>
+70	0.0	0.0
-70 +80	0.0	0.0
-80 +100	4.5	0.43
-100 +120	10.3	0.15
-120 +200	50.4	0.22
-200	38.4	<u>5.91</u>
	Calc Head	<u>2.41</u>

The regrind material was combined with the -70[#] fraction for flotation feed.

Size Analysis of Flotation Feed.

<u>Mesh</u>	<u>% Wt.</u>	<u>% Lead</u>
+80	1.0	0.35
-80 +100	2.4	0.48
-100 +120	7.4	0.44
-120 +200	29.2	0.58
-200	60.0	6.46

Flotation

Rougher Scavenger Float

Four Stage Lock Test

Reagents & Times

	<u>Stage</u>			
	<u>1</u>	<u>2</u>	<u>3</u>	<u>4</u>
Conditioning Min.	5	5	5	5
AeroXanthate 303	.1	.1	.1	.1
Aerofloat 31	.1	.1	.1	.1
Aerofroth 77	.05	.05	-	-
Roughing Min.	5	5	5	5
Conditioning Min.	5	5	5	5
AeroXanthate 303	.1	.1	.1	.1
Aerofloat 31	.1	.1	.1	.1
Aerofroth 77	0.05	-	-	-
Scavenging	3	5	5	5

Results

Head Sample 5.36% Lead.

Tailing Analyses

	<u>Wt. gms.</u>	<u>Wt. %</u>	<u>Lead %</u>	<u>Distribution*</u>
1	624	16.82	0.34	2.85
2	660	17.79	0.07	0.92
3	667	17.98	0.13	1.72
4	750	20.22	<u>0.20</u>	<u>2.98</u>
		Average	<u>0.15</u>	<u>1.89</u>

* Based on original weight of 1000 gms.

All rougher concentrates were combined for regrind and cleaning.

Back calculating Rougher Conc Analysis gave 22.23% Lead.

The table above gives a recovery of 98.11% in the roughing/scavengering circuit.

The analysis of the final scavenger Concentration was

<u>% Wt.</u>	<u>% Lead</u>
6.36	0.71

Screen Analysis of the Tailings

	<u>% Wt.</u>	<u>% Pb</u>
+80	0.8	0.35
-80 +100	8.5	0.02
-100 +120	12.7	0.05
-120 200	32.6	0.02
-200	45.4	0.18
	Calc Head	<u>0.10</u>

Microscope examination of the tails showed that some of the +200[#] particles were clean quartz. An attempt will be made to prepare a glass sand (bottle grade) from this fraction.

Concentrate Regrind

The rougher concentrate was wet screened on 200[#] and the +200[#] rod milled to pass 200[#] and recombined with the -200[#] for the cleaner flotation.

No reagents were added to the cleaning circuit.

	<u>% Wt.</u>	<u>% Lead</u>
Cleaner Tail Stage 1	14.01	3.55
Cleaner Tail Stage 2	1.83	22.86
Cleaner Tail Stage 3	0.65	53.85
Final Concentrate	4.34	77.6

After the analyses had been performed the 3 Cleaner Tails were refloated with one drop of Aerofloat 31 and recleaned.

Metallurgical Balance

	<u>% Wt.</u>	<u>% Lead</u>	<u>Distribution</u>
Final Conc.	4.34	77.6	70.4
Cleaner Tails 1	14.01	3.55	10.4
Cleaner Tails 2	1.83	22.86	8.7
Cleaner Tails 3	0.65	53.85	7.3
Scavenger Conc	6.36	0.71	0.9
Tails	72.81	<u>0.15</u>	<u>2.3</u>
Calc. Head		<u>4.78</u>	<u>100.0</u>
Actual Head		<u>5.36</u>	

The larger than normal difference between the Actual and Calculated Head is due to samples extracted from the circuit for analysis.

Refloating Cleaner Tails

The dried cleaner tails were repulped in the cell and 2 drops Aerofloat and 5 stage cleaning done.

	<u>% Wt.</u>	<u>% Lead</u>
Final Conc. .	2.6	57.69
Cleaner Tail 1	39.6	1.80
Cleaner Tail 2	40.8	2.94
Cleaner Tail 3	9.8	11.31
Cleaner Tail 4	4.50	14.71
Cleaner Tail 5	2.7	29.78

CONCENTRATING(b) Concentrator Equipment

The description and information on concentrator equipment on the following pages has been secured from:-

"Equipment Lists - Cost Centre #15 - Pages 1 - 8
Cost Centre #17 - Pages 1 - 2
Cost Centre #20 - Page 1 "

as prepared by Kaiser Celestite Limited on their concentrator plant at Enon, Loch Lomond District.

Thus any reference made at various stages of flotation treatment in the concentrator will refer to the former Kaiser Celestite circuit and flowsheet in place when the concentrator was purchased from Kaiser by Yava, and not to any future Yava lead concentrator circuit.

CONCENTRATING (b) Concentrator Equipment (1) Crushing, Screening, Conveying, Bins

<u>Quantity</u>	<u>Equipment</u>	<u>Year</u>	<u>Manufacturer</u>	<u>Model</u>	<u>Size</u>	<u>Capacity</u>	<u>Weight</u>	<u>Serial No.</u>
1	Ore Hopper	1970						
1	Syntron Vibrating Feeder Drive	1970	FMC Syntron	GRF4-45-2	36"x120"	100TPH	5,050	
1	Jaw Crusher	1969	Missouri-Rogers	2136 HD	21"x36"	40TPH	23,000	
1	Motor	1969	Westinghouse	2136-J-1100		75HP	1,050	1018535
1	24" Belt Conveyor to Magnetic Separator		Stephens-Adamson	225	24"			
1	Motor		U.S. Electric	Corr. Duty Tot. Enclosed	Frame 215T	10HP		
1	Tramp Steel Magnetic Separator Belt	1970	Stephens-Adamson	24-240-P	24"	300 FPM		A12-300
1	Motor	1970				2HP		
1	Vibrating Shaker Screen	1975	Kaiser rebuild Dillon Screen Associate Motors	Double Deck K213T	4'x10'	40TPH	Drive shaft from old screen	
1	Motor					7.5HP		
1	Nordberg Symons Cone Crusher	1928	Nordberg Symons		3' Std.	100TPH	22,000	348
1	Motor		ES Maude Rebuilt			60HP		3227281
1	24" Belt Conveyor to Transfer		Stephens-Adamson	225	24"x124'			
1	Motor		U.S. Electric	Corr. Duty Tot. Enclosed	Frame 213T	7.5HP		
1	21" Belt Conveyor to Fine Ore Surge Bin		Stephens-Adamson	225	21"x97'			
1	Motor		U.S. Electric	Corr. Duty Tot. Enclosed	Frame 213T	7.5HP		
1	Dust Fan	1970	Canada Blower & Forge, St. John, N.B.	DV2340	10,000CFM	1,322RPM		Wheel Type A
1	Motor							
1	Rock Box Vibrator	1975	Savo	BR-2	8 CFM		23	
1	Fine Ore Surge Bin					350 Tons		
1	Fine Ore Surge Bin Vibrator		Savo	BR-6	40 CFM		350	
1	Ore Surge Bin Vibrator		Higginson Equipment sales	5032L	4"		95	
1	24" Belt Conveyor to Ball Mill		Fabricated at Loch Lomond Reeves	S-A225				
1	Motor		Reeves			2HP		
1	Gear Drive		Reeves					
1	Concentrate Loadout Conveyor		Stephens-Adamson	224-20-24"	24"x50'	40TPH		
1	Motor					1HP		
1	Concentrate Storage Pad		Kaiser Celestite	Concrete	40'x60'			

<u>Quantity</u>	<u>Equipment</u>	<u>Year</u>	<u>Manufacturer</u>	<u>Model</u>	<u>Size</u>	<u>Capacity</u>	<u>Weight</u>	<u>Serial No.</u>
1	Ball Mill-(used shell, new liners 1970)	Used 1970	Allis-Chalmers		8'x11'	25TPH		1475
1	Motor	Used 1970	English Electric		Frame 684	300HP		
1	Gear Drive	1961	Falk	130Y1 6180-0126	1750 in 284 out	325HP		
1	Trommel	1974	Keiser Celestite		1/8" opening			
1	spare Trommel	1976	Keiser Celestite		5/16" opening			stock
1	Dorr Cyclone		Dorr	FR	12"	3"Variable Apex		7517-1
1	Dorr Cyclone		Dorr	FR	12"	3"Variable Apex		7517-2

<u>CONCENTRATING</u>		<u>(b) Concentrator Equipment</u>		<u>(iii) Pumping</u>					
<u>Quantity</u>	<u>Equipment</u>	<u>Year</u>	<u>Manufacturer</u>	<u>Model</u>	<u>Size</u>	<u>Capacity</u>	<u>Weight</u>	<u>Serial No.</u>	
1	ASH Cyclone Feed Pump	1970	Allen Sherman Hoff.	B-6-5	6"	700GPM	1,515	12234UH	
1	Motor		U.S. Electric	Tot. Enclosed	Frame 326T	50HP		14392101	
1	ASH Spray Water Pump	1970	Allen Sherman Hoff.	C-6-5	8"	1,200GPM	1,850	12356VII	
1	Motor		U.S. Electric	Tot. Enclosed	Frame 364T	60HP 1770RPM		P4390923	
1	Sala Vertical Slurry Pump	1970	Sala	SPV304-4	4"		1,500	3604-1	
1	Motor		U.S. Electric	Tot. Enclosed	Frame 254T	15HP 1760RPM		4392317	
1	Sala Vertical Slurry Pump		Sala	SPV304-4	4"		1,500	3604-2	
1	Motor		U.S. Electric	Tot. Enclosed	Frame 254T	15HP 1760RPM		4390733	
1	Sala Vertical Slurry Pump		Sala	SPV304-4	4"		1,500	3604-3	
1	Motor		U.S. Electric	Tot. Enclosed	Frame 254T	15HP 1760RPM		4390761	
1	Sala Vertical Slurry Pump	1970	Sala	SPV304-6	6"		1,800	3604-4	
1	Motor		U.S. Electric	Tot. Enclosed	Frame 286T	30HP 1765RPM		P4390755	
1	Recycle Feed Pump	1970	Sala	SPV365-6	6"		1,800	3604-5	
1	Motor		U.S. Electric	Tot. Enclosed	Frame 256T	20HP 1765RPM		P4390448	
1	Recycle Thickener Pump	1970	Allen Sherman Hoff.	A-6-5	3"	250GPM	741		
1	Motor		C.G.E.	IF3385M	Frame 284T	15HP 1175RPM		L11639	
1	Filter Feed Pump		Allis-Chalmers	SRL	2x2x10				
1	Motor		U.S. Electric		Frame 213T	7.5HP 1740RPM		P4396217	
1	Flotation Area Sump Pump with motor		Flygt	CS3100	3"	9HP			
1	Final Tails Pump	1970	Allen Sherman Hoff.	A-6-5	3"	250GPM	726		
1	Motor								
1	Rougher Tails Pump		Allis-Chalmers	SRL	3x3x10			27535	
1	Motor		C.G.E.	215T	101P	1750RPM			
1	Ball Mill Area Sump Pump	1975	Sala	VASAC 150-234-2	3"			7991-1	
1	Motor	1975	Westinghouse	11A1G	Frame 213 T	7.5HP 1740RPM	185	17-66563	

<u>Quantity</u>	<u>Equipment</u>	<u>Year</u>	<u>Manufacturer</u>	<u>Model</u>	<u>Size</u>	<u>Capacity</u>	<u>Height</u>	<u>Serial No.</u>
1	Denver Automatic Sampler		Denver	1-61003-25				
1	Motor		Westinghouse	Frame EBK 143TW	17 1/2"AL	1/2HP		OA-366277
1	Conditioner Tank				12'x20'H			
1	Agitator		Lightning	74-0-15	2 1/2" shaft	84RPM		270959
1	Motor		U.S. Electric		Frame 254T	15HP		P4390734
1	Gear Drive		Lightning	74-0-15	20.6-1 Ratio			
1	Reagent Feeder		Clarkson	E	18"	0-2000 cc/min	10	
1	Motor		Universal Electric	AB2R034	6"	1.04A	3	11KA9197
1	Denver Automatic Sampler							
1	Motor							
1	Denver Automatic Sampler-Tails							
1	Motor							
1	Reagent Head Tank A-845	1970	Kaiser Celestite		4'x8'x5'			
1	Reagent Head Tank, Fatty Acid&Silicate	1970	Kaiser Celestite		4'x4'x3'			
1	Reagent Mix Tank, Silicate	1976	Kaiser Celestite		38"x33"x80"			

<u>CONCENTRATING</u>		<u>(b) Concentrator Equipment</u>		<u>(v) Flotation</u>				
<u>Quantity</u>	<u>Equipment</u>	<u>Year</u>	<u>Manufacturer</u>	<u>Model</u>	<u>Size</u>	<u>Capacity</u>	<u>Weight</u>	<u>Serial No.</u>
10	Flotation Cells-Roughers Nos. 1 thru 10		Denver	DR24				
10	Flotation Cell Agitators for Cells Nos. 1 thru 10							
5	Flotation Cell Agitator Motors (1 Motor for 2 cells & 2 agitators)		General Electric	IF3335N	Frame 256T	20HP 1760RPM		
1	Flotation Cell Agitator Gear Drive		Hamilton Gear		2.92 1P 26AF	10:1		15812
5	Flotation Cells-Scavengers Nos. 11 thru 15		Denver	DR24				
5	Flotation Cell Agitators for cells 11 thru 15,							
1	Flotation Cell Agitator Motor		General Electric	IF1295M	Frame 254T	15HP 1755RPM		
1	Motor		English Electric		Frame 324	10HP 1730RPM		216756
1	Motor		U.S. Electric	P4396072	Frame 254T	15HP 1760 RPM		
5	Flotation Cells-1st Cleaners Nos. 16 thru 20		Denver	DR24				
5	Flotation Cell Agitators for Cells Nos. 16 thru 20							
1	Flotation Cell Agitator Motor		U.S. Electric		Frame 254T	15HP 1760RPM		P4392318
1	Gear Drive		Hamilton Gear		26AF	2.92HP 10:1		
1	Motor		General Electric	IF1295M	Frame 254T	15HP 1755RPM		
8	Flotation Cells-2nd Cleaners Nos. 21, 22, 23, 26, 27, 28. No. 24, (Level Control Weir) No. 25		Denver Denver Denver	DR18 DR18 Sub A18				
8	Flotation Cell Agitators for Cells Nos. 21 thru 28							No forward dir.
1	Flotation Cell Agitator Motor		Westinghouse	Life-LineT Type NSB	Frame 254T	15HP 1760RPM		55-66411
1	Gear Drive		Eberhart Denver		26SB 1.31 1P	10.66:1 1800RPM		550682
1	Motor		English Electric		Frame 324	10HP 1730RPM		233904
1	Motor		Lancashire Dynamo		Frame 254U	7.5HP 1750RPM		26823
1	Motor		English Electric	AD 745-374	Frame 324	10HP 1730RPM		

CONCENTRATING(h) Concentrator Equipment(v) Flotation (Cont'd.)

<u>Quantity</u>	<u>Equipment</u>	<u>Year</u>	<u>Manufacturer</u>	<u>Model</u>	<u>Size</u>	<u>Capacity</u>	<u>Weight</u>	<u>Serial No.</u>
4	Flotation Cells 3rd Cleaners No's 29 thru 32		Denver	Sub A 18				
4	Flotation Cell Agitators for Cells No's 29 thru 32							
1	Flotation Cell Agitator Motor		C.G.E.	9F1225N	Frame 254U	7.5 h.p. 1760 rpm		
1	Motor		Gen. Electric	9F1225N	Frame 254U	7.5 h.p. 1760 rpm		
4	Flotation Cells 4th Cleaners No's 33 thru 36		Denver	Sub A 18				
4	Flotation Cell Agitators for Cells No's 33 thru 36							
1	Flotation Cell Agitator Motor		Gen. Electric	9F1225N	Frame 254E	7.5 h.p. 1760 rpm		
1	Gear Drive		Boston Reductor		9.3/4:1			183831
1	Motor		Gen. Electric	9F1225N	Frame 254U	7.5 h.p. 1760 rpm		

CONCENTRATING(b) Concentrator Equipment(vi) Thickening

<u>Quantity</u>	<u>Equipment</u>	<u>Year</u>	<u>Manufacturer</u>	<u>Model</u>	<u>Size</u>	<u>Capacity</u>	<u>Weight</u>	<u>Serial No.</u>
1	Re-cycle Thickener - Tank - Wood	Used 1940						
1	- Rake Mechaniam	Used 1940	Porreo	Type S	45' ϕ x12'			Wood Stave Tank
1	- Rake Gear Drive	Used 1940	Porreo	D 13931		Centre Column		U54851
1	- Motor		C.G.E.	69100-S12	Frame 225	13,000 \bar{g} 3 h.p.		Chain Drive to Gear Drive
1	Concentrate Thickener - Tank - Steel	1967	Dorr Oliver Long	Type A	20' ϕ x6'			Dol
1	- Rake Mechaniam		Dorr Oliver Long	2 long rakes 2 short	14,000 gal.			7462-1
1	- Rake Gear Drive		Foots & Whitney	16A		Ratio 52:1	700	70-3339
1	- Rake Liftors		Huff - Norton	CH-1801-15	12"			Manual Lift
1	- Motor		U.S. Electric		Frame 143T	3/4 h.p.		P438560

CONCENTRATING(b) Concentrator Equipment(vii) Filtering

<u>Quantity</u>	<u>Equipment</u>	<u>Year</u>	<u>Manufacturer</u>	<u>Model</u>	<u>Size</u>	<u>Capacity</u>	<u>Weight</u>	<u>Serial No.</u>
1	Disc Filter (American Type) Modified to 4 disc by Kaiser		Finco		6' x 6 disc			13396
1	- Filter Drive		Reeves	MDM1773993 GP	223 - 1M 18	1½ h.p. 29.3-8.93 rpm		
1	- Agitator Drive		Reliance Master	427833-ET 182 DC 28A	Ratio 17:1	3 h.p. 100 rpm		

CONCENTRATING(b) Concentrator Equipment(viii) Compressors & Vacuum Pumps

<u>Quantity</u>	<u>Equipment</u>	<u>Year</u>	<u>Manufacturer</u>	<u>Model</u>	<u>Size</u>	<u>Capacity</u>	<u>Weight</u>	<u>Serial No.</u>
1	Spencer Blower- Low Pressure		Spencer	1007-R		1075 cfm 1500 rpm		238347
1	- Motor		U.S. Electric		Frame 184TZ	7.5 h.p. 3450 rpm		18183-947
1	Webster Compressor - High Pressure	1971	W.M.White			100 cfm		
1	- Motor		Webster (Dust) Brook	804-1 DP	Frame K 284T	200 psi 25 h.p. 1730 rpm	2080	S 07442
1	Nash Vacuum Pump		Nash	Hytor	702	675 cfm		7005766
1	- Motor		U.S. Electric		Frame 324T	40 h.p. 1765 rpm		P4396797

CONCENTRATING(b) Concentrator Equipment(ix) Miscellaneous

<u>Quantity</u>	<u>Equipment</u>	<u>Year</u>	<u>Manufacturer</u>	<u>Model</u>	<u>Size</u>	<u>Capacity</u>	<u>Weight</u>	<u>Serial No.</u>
1	Ball Mill Area Sump		Maiser Celestite	Concrete				
1	Flotation Area Sump		Maiser Celestite	Concrete				
1	Rougher Tails Tank							
1	Howe-Richardson Scale		Howe Richardson	5401-8B1		1000		E52716
1	Tank Concentrate Thickener Comp	1972	Maiser Celestite	55" O x 36" h.				
1	Diesel Electric Generator (stand-by)							
	Diesel	Used 1970	Caterpillar	D334	6 cyl.	235 h.p.		
	Generator		Allis-Chalmers		600 v	1800-1350 rpm		920 B 594
						187-512VA		327035
						150 kw.		
1	Space Heater Furnace - oil burning	1970	Industri-Aire	HD-GH40				
1	Space Heater Furnace - oil burning	1970	Industri-Aire	HD-GH50				

CONCENTRATING

(b) Concentrator Equipment

(x) Electrical

<u>Quantity</u>	<u>Equipment</u>	<u>Year</u>	<u>Manufacturer</u>	<u>Model</u>	<u>Size</u>	<u>Capacity</u>	<u>Weight</u>	<u>Serial No.</u>
1	Electric Motor Control Centre Located on ground floor of concentrator in a concrete block room. Push button control stations with control lights above control room on Mezzanine floor.		Cutler-Hammer	Unitrol				

CONCENTRATING(b) Concentrator Equipment(xi) Buildings

<u>Quantity</u>	<u>Equipment</u>	<u>Year</u>	<u>Manufacturer</u>	<u>Model</u>	<u>Size</u>	<u>Capacity</u>	<u>Height</u>	<u>Serial No.</u>
1	Crusher Building Concrete walls and floors, light steel superstructure, asbestos cement cladding				18' x 30' x 22' x 60' high			
1	Main Process Building Prefabricated steel, asbestos cement cladding, insulation with rockwool batts, top 4 feet of wall is translucent plastic. Building is fitted with an inside overhead crane and sodium vapour lighting.		Butler		40' x 120' x 46' high			
1	Overhead Crane in Main Process Building	1970	Demag	P210 LF10 Type 242		4400#		
1	Concentrate Pad Cover Building				40' x 30'			
1	Administration Building with Portaspec Room Addition		Concrete Block		36' x 50' x 9' high			

CONCENTRATING:(c) Proposed Alterations to Concentrator

All of the equipment in the concentrator must be re-commissioned by making a close examination and completing any necessary repairs or adjustments before the concentrator is returned to continuous operation. It appears all of the equipment has been maintained in good condition by the previous operators and the concentrator was left in a clean condition. However, this equipment has been standing idle for one year now and machinery can develop problems when idle as well as when operating. Such a programme will involve the examination of all bearings and lubrication lines, examination of all wearing parts, the checking and drying of all electrical circuits, and the painting of equipment which shows signs of corrosion and deterioration.

The concentrator flowsheet will have to be changed, since the flotation circuits required for concentration of a lead ore will be different than those for concentrating celestite. The major job here is the fabrication of new launders to and from the flotation cells and the change in connections between the various groups of flotation cells.

Also, it is proposed to house in or enclose the exposed conveyor galleries and the exposed concentrate thickener, since reports from previous operations indicate the conveyor belts are subject to overturn during high winds and the thickener will give better performance when protected from low temperatures. The housing of the concentrate thickener will probably be combined with the erection of a lean-to addition to the south side of the concentrator building on a concrete pad which is presently in place.

(c) Proposed Alterations to Concentrator (Cont'd)

The addition will provide housing for a machine and repair shop and a warehouse, the space for which does not exist at the present time.

New analytical instrumentation for rapid on-site analysis of mine and concentrator samples in the form of an Inax or Portospec X-Ray Spectrometer will be installed in an addition to the administration building, which was constructed by Kaiser Celestite.

It is estimated one supervisor and 9 personnel as mechanical and electrical technicians and labour would be required for commissioning the concentrator equipment for a period of 2 months on an 8-hour per day, 5-day per week basis. Housing and additions would probably be completed under a local contract arrangement.

CONCENTRATING:(d) Tailings Disposal

Mr. M. A. K. Grice, a staff member of the Atlantic Industrial Research Institute at Nova Scotia Technical College, Halifax, was involved in a senior capacity with the installation and early operation of the Kaiser Celestite concentrator. The following report by Mr. Grice on the disposal of tailings and liquid waste was prepared in conjunction with his test work on the concentration of the Yava ore, and is considered an authoritative report on this subject.

A review of the open pits and settling pond facilities indicates sufficient tailings storage space available for at least seven years operation of the concentrator.

Environmental Considerations

Lead Concentration Mill

Enon

Cape Breton

M. A. K. Grice

M.A.K. Grice, P. Eng.

1.0 Introduction

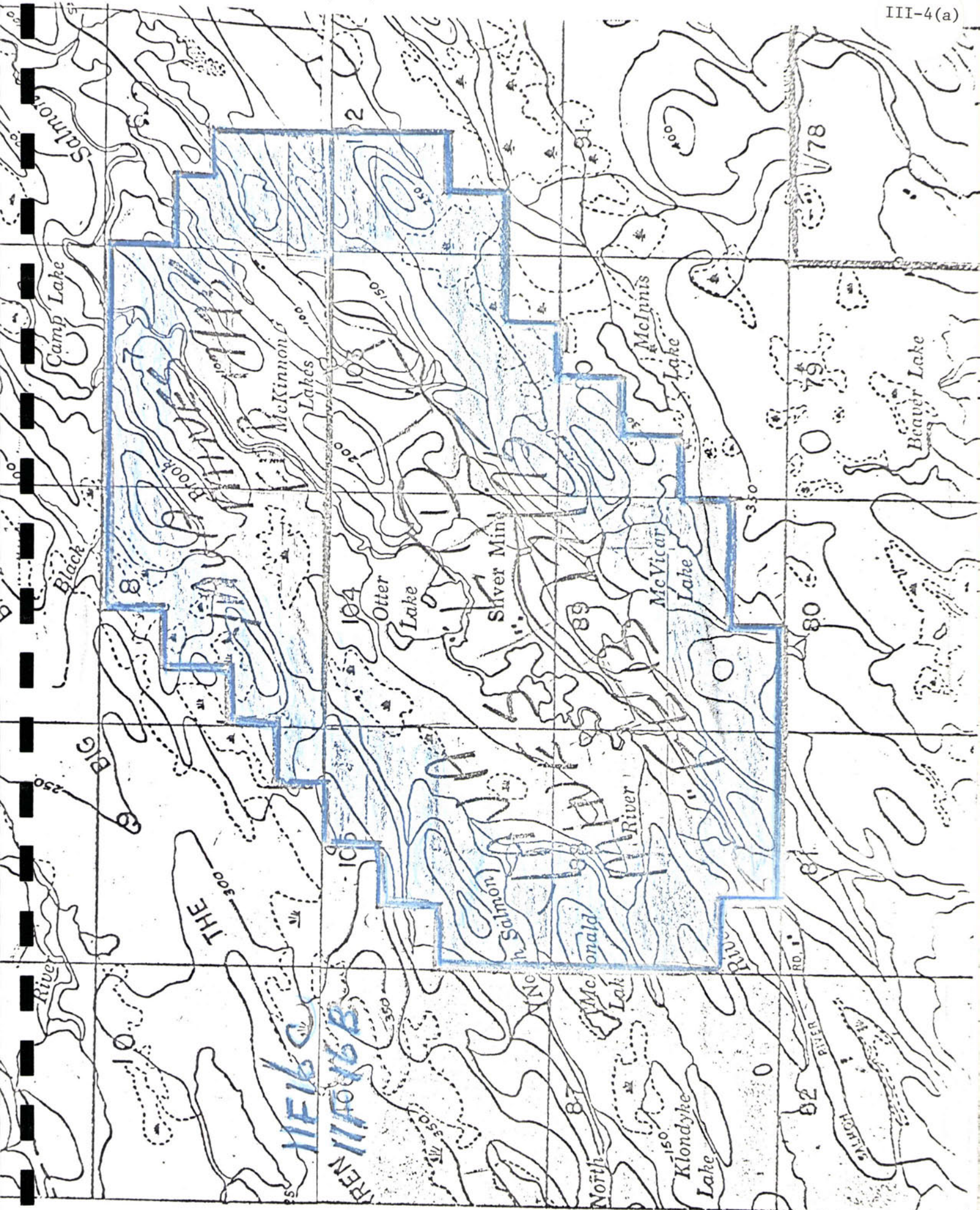
The concentration mill that Barymin intend to use was originally used by the Kaiser Celestite Mining Co. to process celestite ore (SrSO_4). No major alterations are proposed to the milling circuit, only minor alteration to the pipework, and a different suite of chemicals to treat the ore. The water reclaim system remains the same as previously used and is detailed in Section 2.0

2.0 Water Circuit

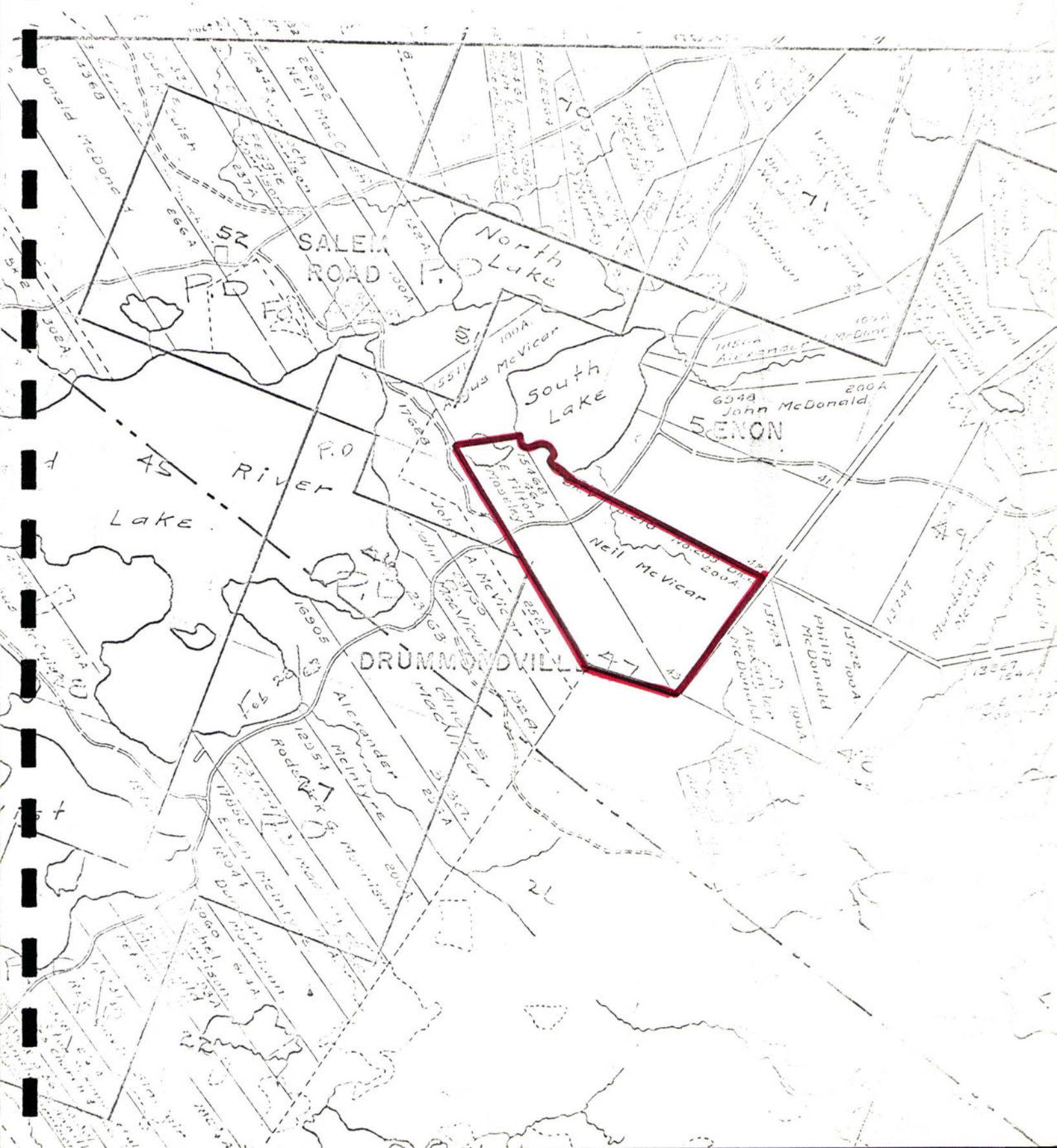
Water additions to the process are made in both the Grinding and Flotation section of the mill.

Because of the single sulphide recovery from the ore no depressants are needed so the water can be recycled back to the process. (In fact reuse will help to reduce the costs.) With the process selected there also is no need to adjust the pH of the water for metallurgical reasons but to keep the level of soluble lead in the water below the accepted maxim of 250 ppb the pH will have to be increased with the use of lime from 6.0 to 7.5

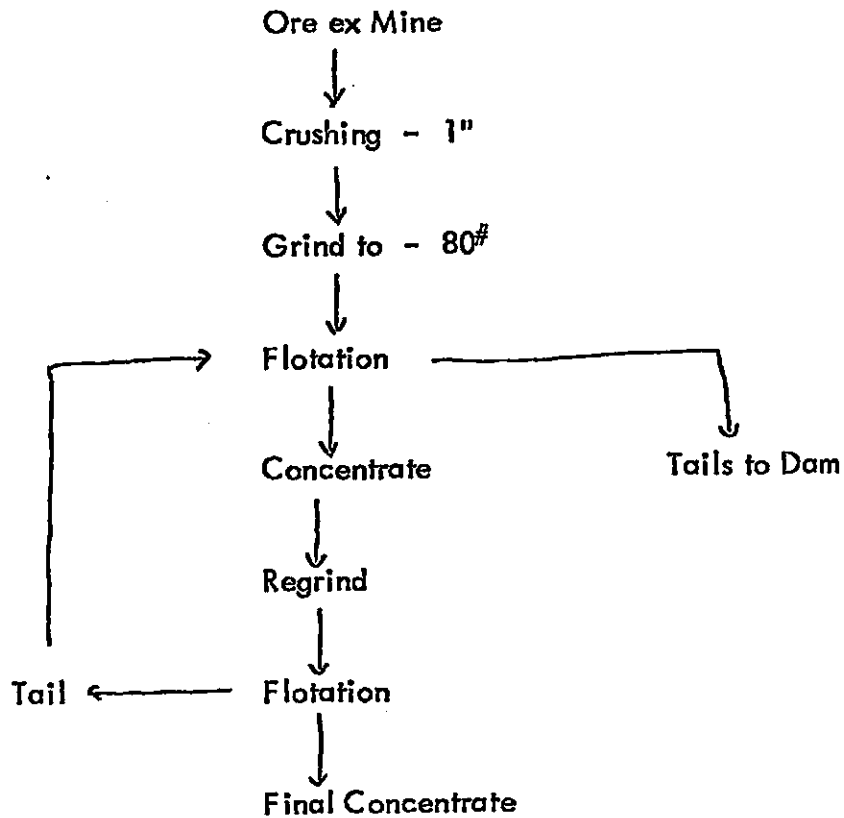
The proposed circuit is shown in Section 2.2.



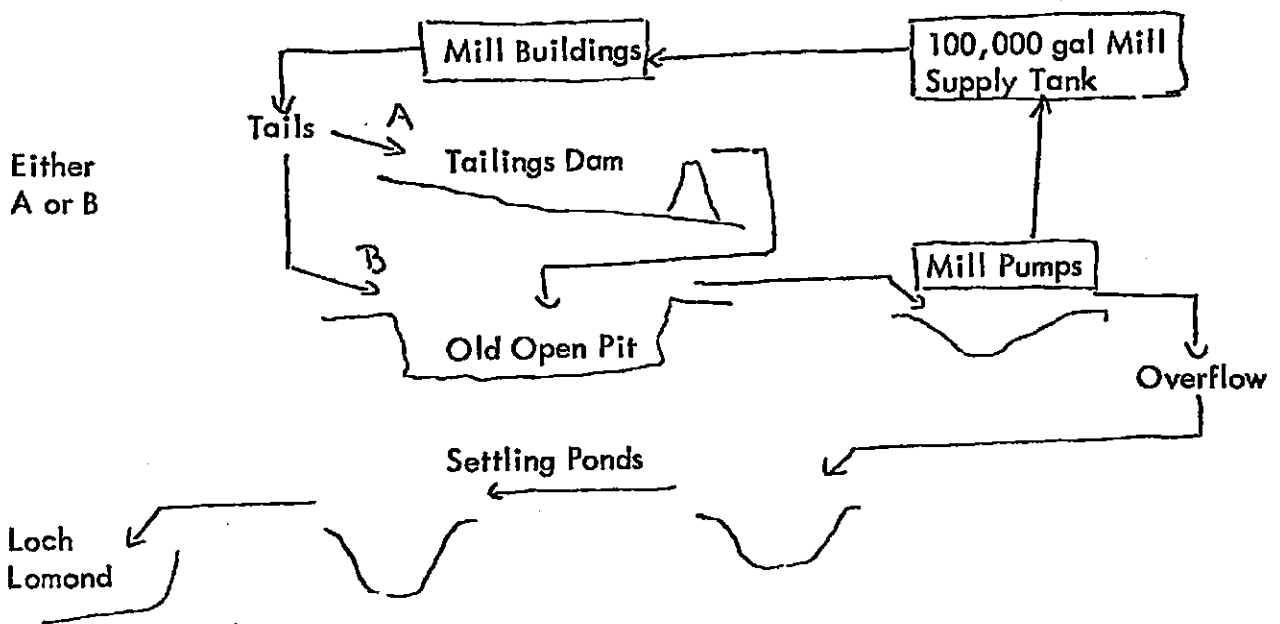
1 in. = 1/2 Mi



2.1 Basic Flowsheet



2.2 Proposed Water Circuit



2.3 Process effects on Water Quality

The flotation tests were conducted using Halifax City Water, which due to problems in supply matched the pH condition of the Loch Lomond/Salmon River watershed

The analyses are given in Table 1. Without lime treatment the amount of soluble lead both after grinding and flotation is unacceptably high, but adjustment of the pH with lime reduces it to an acceptable level.

Lime treatment will be necessary at the mill for environmental reasons and not metallurgical ones.

3.0 Proposed Flotation Chemicals

It is proposed to use the following chemicals in the flotation process. The following are Cyanimid Products.

AeroXanthate 325 (The tests were done with 303, the potassium salt no longer available, 325 is the sodium salt of ethyl Xanthate.

Aerofloat 31 Aryl dithiophosphoric acid.

Aerofroth 77 Straight Chain Higher Alcohol.

Estimated quantities per ton milled

	<u>lbs/ton</u>
Aeroxanthate 325	0.2
Aerofloat 31	0.1
Aerofroth 77	0.02

Other Reagents

Lime is required to adjust the pH to 7.5, no quantity is given because the amount will drop as the pH of the recirculated water increases.

With regard to the toxicity of the reagents the Ontario Dept. of Environment published a paper "Use, Characteristics and Toxicity of Mine-Mill Reagents in Ontario

TABLE 1.

	<u>Before Use</u>	<u>After Grinding</u>		<u>After Flotation</u>	
		<u>As Is</u>	<u>Lime Treated</u>	<u>As Is</u>	<u>Lime Treated</u>
pH	5.7	5.7	7.5	6.0	7.5
Conductivity	110	480	465	275	250
Fe ppm	.43	.75	.05	.80	.06
Mn ppm	.1	.38	.008	.61	.01
p.p.b.					
Cu	480	13	1	9	3
Cd	1	1.5	2	1.5	1.5
Pb	152	500	58	1800	170
Ni	15	65	14	17	17
Zn	370	84	17	200	22
Co	1	3	4	2	2

1972" which includes studies on the above reagents.

4.0 Tailing Acidity Test

This test was to confirm that there is enough carbonate in the tailing to more than neutralize the sulphide remaining in the tailing.

<u>% Total S</u>	<u>% Calcium Carbonate</u>
0.38	8.4

The above analyses is for total sulphur. The sulphide sulphur will be lower than this. Not included is the amount of lime that will be added during the flotation process.

The analyses confirm that there is more than enough lime to neutralize any acid produced from the sulphides in the tailings.

5.0 Other Items

5.1 Domestic Water

A bore hole N.E. of the mill building supplies domestic water to the mill and mill offices.

5.2 Sewage Disposal

A small sewerage plan treats all the effluent from the mill offices, changing rooms and is discharged into the settling ponds.

CYANAMID

CYANAMID OF CANADA LIMITED

2031 KENNEDY ROAD
SCARBOROUGH, ONTARIO, M1P 2M4

October 11, 1977

AREA CODE 416
TELEPHONE 293-8181Nova Scotia Technical College
P.O. Box 1000
Halifax, N.S.Attention: Prof. M.A.K. Grice
Mining and Metallurgical Engineering

Dear Mike:

In addition to the booklet put out by the Ministry of the Environment - "Use, Characteristics and Toxicity of Mine - Mill Reagents in Ontario - 1972", you might find the following article of interest " Toxicity of Selected Sulphydryl Collectors to Rainbow Trout".

If further information is required, please contact us.

Under separate cover you will be receiving some conversion booklets.

Yours very truly

CYANAMID OF CANADA LIMITED

W.H. Thomas
Sales Manager
Mining, Water Treating and
Paper Chemicalsenclosure
mo

Toxicity of Selected Sulfhydryl Collectors to Rainbow Trout

by M. C. Fuerstenau, B. M. Wakawa, R. K. Price, and R. D. Wellik

Median lethal concentrations of pure and commercially available ethyl xanthate, isopropyl xanthate, amyl xanthate, diethyl dithiophosphate, dibutyl dithiophosphate, and isopropyl ethylthionocarbamate for rainbow trout have been established utilizing static bioassays.

Since the discovery of the specificity of xanthate for sulfide minerals, use of this reagent as a collector has increased to the point where the metallurgical industry in this country alone uses approximately 8 million lb annually. In addition to xanthate, considerable quantities of other sulfhydryl collectors, such as thiophosphate and thiocarbamate, are also utilized. Billions of gallons of water are used in conjunction with these collectors, and although most flotation plants reclaim water used in processing, some water containing residual collector is lost to natural receiving waters. It is expected that these chemicals will exhibit toxic effects on aquatic life.

The toxic effects of other flotation reagents have already been demonstrated. For example, Laclerc and Devlaminc¹ have shown the limiting concentrations of sodium oleate, palmitate, and stearate to be 10 to 12 ppm in distilled water and in the case of sodium oleate, 900 to 1000 ppm in hard water.

Schmid and Mann² have reported that a concentration of 5 ppm dodecylbenzene sulfonate is toxic to trout. These results are similar to those reported by Henderson, et al.³ for minnows.

Because of the quantity of sulfhydryl collectors and water used in flotation processing, toxicity data are needed to determine the potential of these reagents for harming fish life. The objective of this investigation was to establish the 96-hr median lethal concentration (LC50-96) of selected sulfhydryl collectors, namely ethyl xanthate, isopropyl xanthate, amyl xanthate, diethyl dithiophosphate, dibutyl dithiophosphate, and isopropyl ethylthionocarbamate to rainbow trout (*Salmo gairdneri*).

Experimental Materials and Procedure

Experiments were conducted with commercially available and purified collectors. Purification of the xanthates was effected by dissolving commercially available potassium xanthate in acetone and precipitating the salt with petroleum ether. The purified xanthate was filtered, washed with petroleum ether, dried under vacuum, and stored in a desiccator. The purity, based on potentiometric titration, was measured to be 99%.

Pure dithiophosphates were obtained by reaction of dithiophosphoric acid and the stoichiometric amount of

potassium hydroxide dissolved in methyl alcohol. This solution was placed in a vacuum desiccator until the methyl alcohol and water formed by reaction were removed as vapor. The remaining material was dissolved in acetone and purified using the same process described for xanthate. Potentiometric titration revealed that this material was 99% pure.

Pure isopropyl ethylthionocarbamate was prepared by vacuum distillation of Z-200 at 103°C and 10.6 mm mercury pressure. The distillate obtained before constant boiling temperature was reached was discarded as this product contained impurities of low molecular weight.

The experimental water used in the investigation was a naturally hard spring water which is a source of water for a trout fish hatchery. The pH of the water was 8.6; other constituents are given in Table 1.

Fingerlings obtained from three different egg sources were used in the study. The fingerlings, which ranged in average weight from 1 to 10 g, were transported to the laboratory in polyethylene bags under an oxygen atmosphere. A 24-hr period was allowed to relieve the fish of the stress incurred during transportation.

After this time period, 20 fish were added to each of the test aquaria. The volume of water was adjusted to maintain a weight to volume ratio of 2 g of wet fish per l of solution. This ratio, suggested by Sprague⁴ for static tests, is more than that required for consumption by trout which is 0.22 cc O₂ per g per hr at 14.7°C or 0.047 cc O₂ per g per hr at 11.6°C as found by Heilbrunn⁵ and van Dam,⁶ respectively. Oxygen saturation was maintained in the control and test aquaria by bubbling air into the water contained in each of the aquaria. Oxygen content was monitored periodically to ensure that saturation was maintained.

Approximately 1 hr after the fish had been transferred to the test aquaria, the collector was added, hav-

Table 1. Composition of Experimental Water

	Ppm
Total Dissolved Solids	330
Volatile Dissolved Solids	102
Total Hardness	310
Carbonate Hardness	203
Noncarbonate Hardness	145
Calcium	63
Magnesium	34
Sodium	2
Potassium	1
Iron	0
Manganese	0.2
Alkalinity (Methyl Orange)	203
Chlorides	2.5
Sulfate	85
Silica	13
Nitrates	0.30
Total Phosphate	0.7
pH	8.6

M. C. FUERSTENAU, Member SME, is Professor and Chairman and B. M. WAKAWA is Research Assistant, Dept. of Metallurgical Engineering, South Dakota School of Mines & Technology, Rapid City. R. K. PRICE is Quality Control Engineer, Caterpillar Tractor Co., Peoria, Ill. R. D. WELLIK is Process Engineer, Alcoa Warwick Operations, Newburgh, Ind. TP 748211. Manuscript, Jun. 20, 1974. Discussion of this paper, submitted in duplicate prior to Mar. 15, 1975, will appear in SME Transactions, June 1975, and in AIME Transactions, 1975, Vol. 258.

ing dissolved previously in an aliquot of experimental water. Water temperature was maintained at $12 \pm 1^\circ\text{C}$ unless temperature was the parameter under investigation.

Experimental Results

Bioassays were conducted first with purified potassium ethyl xanthate. Median lethal concentrations for 96-hr exposures have been established by plotting median survival times, LC50, as a function of the logarithm of the xanthate concentration. The data in Fig. 1 show a well-defined break between the concentrations which did not cause 50% mortality and those concentrations which caused 50% mortality in less than 96 hr. It also shows that source of eggs has no effect on sensitivity to xanthate. Those points with extending arrows at the 96 hr level indicate that LC50 was not attained in 96 hr.

It can also be noted that concentrations of less than 15 ppm potassium ethyl xanthate did not cause 50% mortality in 96 hrs, while concentrations greater than 20 ppm potassium ethyl xanthate show decreasing median survival times as the concentration of xanthate is increased. Concentrations of potassium ethyl xanthate greater than 25 ppm are seen to cause 50% mortality in approximately 20 hr. From the break in the bioassay data, the LC50-96 is seen to lie between 15 and 20 ppm potassium ethyl xanthate.

Experiments were also run with commercially available potassium ethyl xanthate, Z-3, the results of which are presented in Fig. 2. As shown, LC50-96 was found to lie between 15 and 17 ppm.

Median survival times were established next for commercially available sodium ethyl xanthate (Aeroxanthate 325 and Z-4) and sodium isopropyl xanthate (Aeroxanthate 343 and Z-11). As can be noted from Fig. 3 to 6, the median lethal concentration of Aeroxanthate 325 is 14 to 16 ppm, of Z-4 is 13 to 15 ppm, of Aeroxanthate 343 is 18 to 20 ppm, and of Z-11 is 18 to 20 ppm.

Median survival times in the presence of various concentrations of purified potassium amyl xanthate are plotted in Fig. 7. As shown, 50% mortality was not caused by concentrations of less than 70 ppm in 96 hr. The LC50-96 is seen to lie in the range of 70-75 ppm potassium amyl xanthate.

Experiments were also run with a potassium amyl xanthate (Aeroxanthate 350) that is available commer-

cially for industrial use. Median survival time is listed as a function of potassium amyl xanthate concentration in Table 2. As can be noted, the LC50-96 lies between 70 and 80 ppm.

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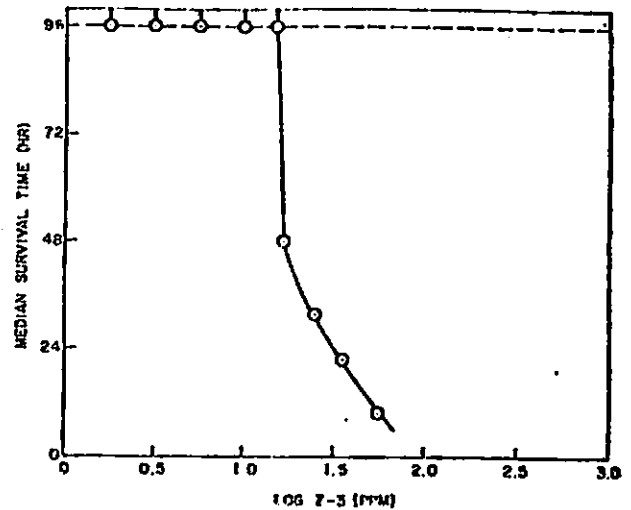


Fig. 2—Median survival time as a function of the logarithm of Z-3 (potassium ethyl xanthate) concentration at 12°C .

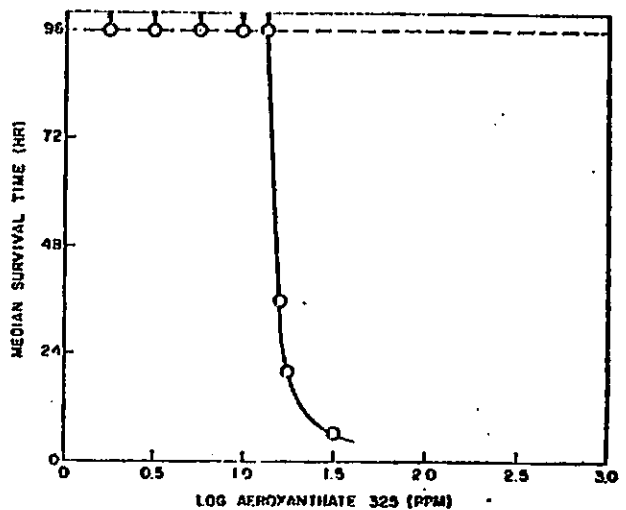


Fig. 3—Median survival time as a function of the logarithm of Aeroxanthate 325 (sodium ethyl xanthate) concentration at 12°C .

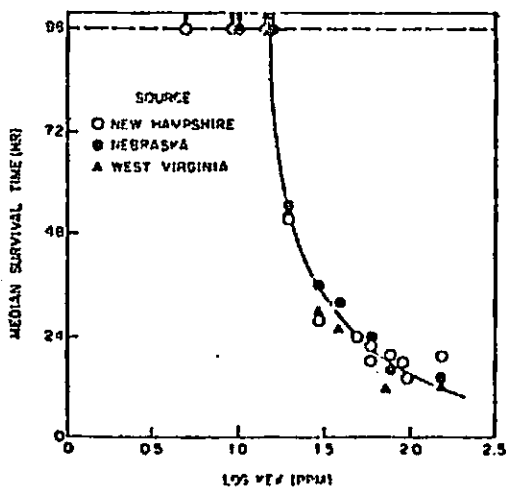


Fig. 1—Median survival time as a function of the logarithm of purified potassium ethyl xanthate concentration and egg source of fingerlings.

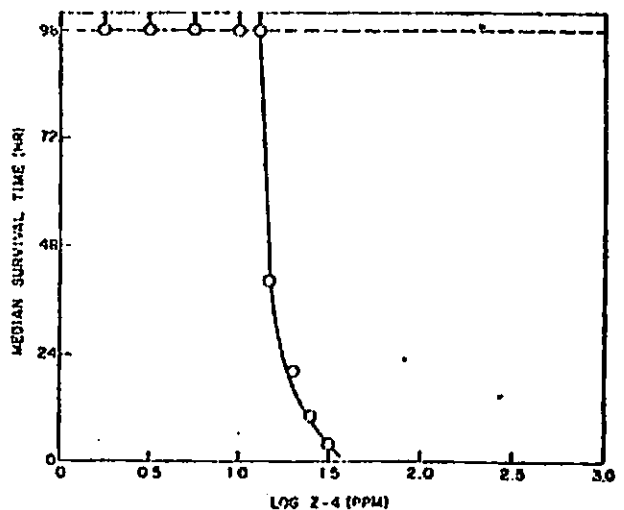
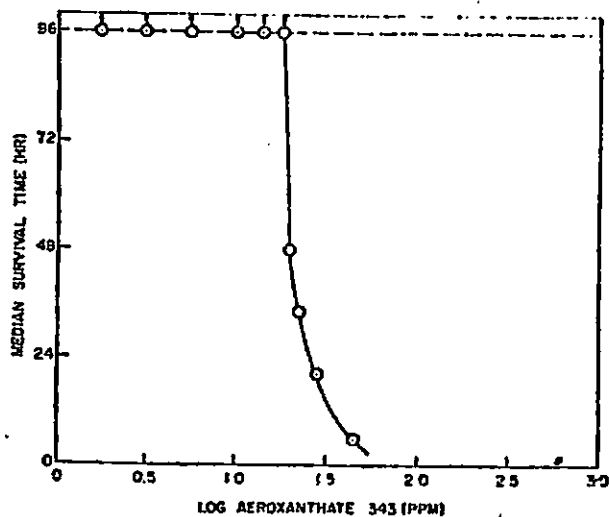


Fig. 4—Median survival time as a function of the logarithm of Z-4 (sodium ethyl xanthate) concentration at 12°C .

The role that temperature assumes in these systems was also investigated (Fig. 8). Median lethal concentration of ethyl xanthate was found to be the same at 8° and 12°C. Increasing the temperature to 16° and 20°



Median lethal concentrations of the commercially available forms of dithiophosphate are shown in Figs. 10 and 11. Median lethal concentrations of commercially available sodium diethyl dithiophosphate (Sodium Aerofloat) are 400 to 410 ppm and 310 to 330 ppm at 12°C and 16°C, respectively. The median lethal concentration of commercially available sodium disecundary-butyl dithiophosphate (Aerofloat 238) is 600 to 625 ppm at 12°C.

The median lethal concentrations of purified and commercially available isopropyl ethylthionocarbamate (Z-200) were also established. As shown in Figs. 12 and 13, at 12°C these values are 40 to 45 ppm and 45 to 48 ppm, respectively.

Discussion of Results

The conditions under which the bioassays were conducted were characteristic of the natural habitat of trout. As a result, the effects of acclimatization may be excluded as the primary cause of death of the fish. Further, the accumulation of waste products, especially lactic acid, during transportation may also be excluded from consideration as a holding period of 24 hr was used to allow ample time for reliefment of stress incurred during transportation. As a result, the presence of collector can be assumed to be the primary cause of respiratory stress and death.

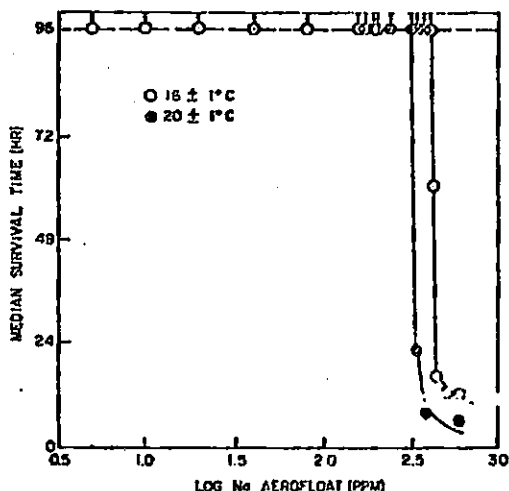


Fig. 10—Median survival time as a function of the logarithm of Sodium Aerofloat concentration at two temperatures.

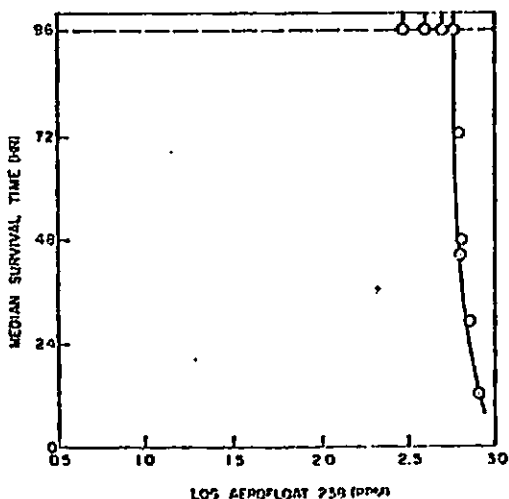


Fig. 11—Median survival time as a function of the logarithm of Aerofloat 238 concentration at 12°C.

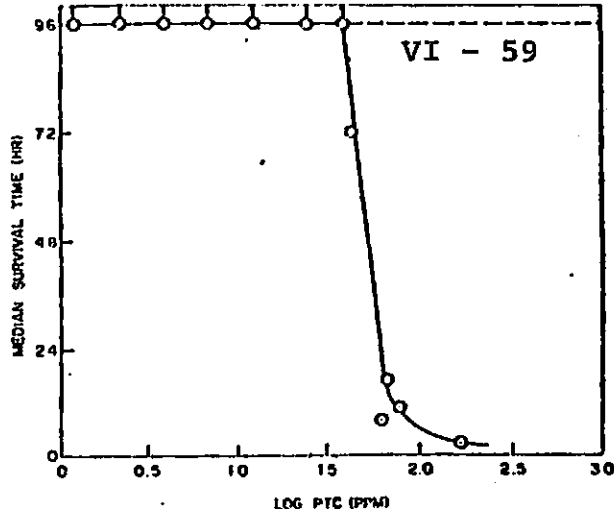


Fig. 12—Median survival time as a function of the logarithm of purified isopropyl ethylthionocarbamate concentration at 12°C.

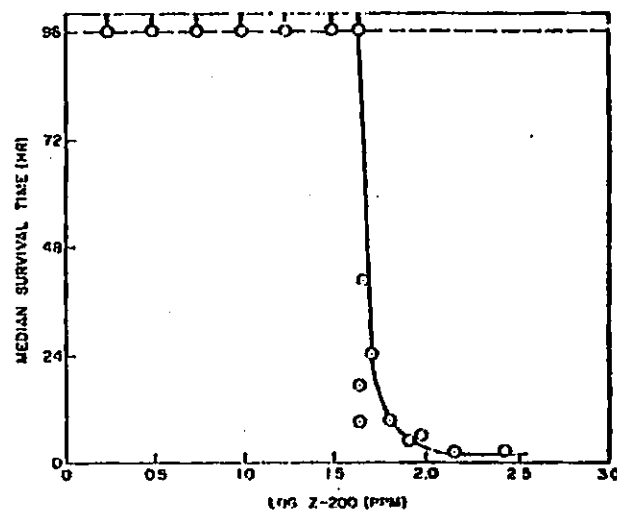


Fig. 13—Median survival time as a function of the logarithm of commercially available isopropyl ethylthionocarbamate (Z-200) concentration at 12°C.

During the course of the experiments, it was noted that the presence of these reagents caused a change in the respiratory pattern of the fish. Upon addition of the reagents, an increase in breathing movements was noted which, in turn, caused an increase in volume of collector-bearing solution to pass over the gill surfaces. Since the gill surfaces are probably the site of absorption of most poisons⁹ and also since no mucus precipitates were observed, these collectors were assumed to affect the trout by way of the gills.

Although the determination of the toxic mechanism for these reagents was beyond the scope of this investigation, two possible mechanisms are (1) absorption through the lamellae and reaction with hemoglobin, and (2) adsorption on the gill surfaces resulting in interference with waste removal and oxygen utilization.^{9,10} Either of these mechanisms could account for the period of equilibrium loss and swimming difficulty if these symptoms were caused by the accumulation of urea and lactic acid. The accumulation of these waste products, caused by interference from the collector with normal excretory processes, could reach lethal levels and kill the fish. The loss of equilibrium, decreased skeletal muscle control, and the disruption of the central nervous system have been shown to be the results of lethal

lactic acid concentrations.¹⁰ However, the fact that more amyl xanthate than ethyl xanthate and that more dibutyl dithiophosphate than diethyl dithiophosphate are required to attain LC50 indicates that absorption through the lamellae is involved. If adsorption on the gill surfaces were involved, less amyl xanthate than ethyl xanthate would be required, since greater adsorption of the longer-chain homologue would occur at equal concentration.

With regard to data established with purified and commercially available collectors, the fact that essentially the same values of LC50-96 were obtained for purified and commercially available ethyl and amyl xanthate and isopropyl ethylthionocarbamate indicates that the commercially available samples were essentially 100% active and also that the samples did not contain any toxic impurities.

In the case of the dithiophosphates, however, considerable differences in values of LC50-96 were obtained between the pure and impure forms of the reagent. The fact that values of LC50-96 were considerably lower for the commercially available forms can probably be attributed to the hydrogen sulfide content of these samples. That is, hydrogen sulfide is a reaction product in the manufacture of dithiophosphates, accounting for its presence in the sample. Secondly, the distinctive odor of H₂S was apparent when dithiophosphate was added to the test water. Thirdly, the lethal concentration of H₂S is 1 to 3 ppm.¹¹

Median lethal concentration of sodium ethyl xanthate is lower than that for sodium isopropyl xanthate or potassium ethyl xanthate. This is expected in view of the differences in molecular weight between these collectors. That is, the molecular weight of sodium ethyl xanthate is 144 g per mole, that of sodium isopropyl xanthate is 156 g per mole, and that of potassium ethyl xanthate is 160 g per mole. If the median lethal concentration of potassium ethyl xanthate is 15 to 17 ppm, the median lethal concentration of sodium ethyl xanthate should be 15 to 17 ppm × 144/160 or 13 to 15 ppm, which is the value determined experimentally.

Conclusions

The following conclusions can be drawn from this investigation with rainbow trout (*Salmo gairdnerii*):

- 1) The median lethal concentration of Aeroxanthate 325 (sodium ethyl xanthate) is 14 to 16 ppm at 12°C
- 2) The median lethal concentration of Z-4 (sodium ethyl xanthate) is 13 to 15 ppm at 12°C.
- 3) The median lethal concentration of Z-3 (potassium ethyl xanthate) is 15 to 17 ppm at 12°C.
- 4) The median lethal concentration of pure potassium ethyl xanthate is 15 to 20 ppm at 8°C and 12°C, 1.8 to 2.0

ppm at 16°C, and 1.5 to 1.8 ppm at 20°C. VI - 60

5) The median lethal concentration of Aeroxanthate 343 (sodium isopropyl xanthate) is 18 to 20 ppm at 12°C.

6) The median lethal concentration of Z-11 (sodium isopropyl xanthate) is 18 to 20 ppm at 12°C.

7) The median lethal concentration of Aeroxanthate 350 (potassium amyl xanthate) is 70 to 80 ppm at 12°C.

8) The median lethal concentration of pure potassium amyl xanthate is 70 to 75 ppm at 12°C.

9) The median lethal concentration of Sodium Aerofloat is 400 to 410 ppm at 12°C and 310 to 330 ppm at 16°C.

10) The median lethal concentration of pure potassium diethyl dithiophosphate is 800 to 825 ppm at 12°C.

11) The median lethal concentration of Aerofloat 238 is 600 to 625 ppm at 12°C.

12) The median lethal concentration of pure potassium disecundarybutyl dithiophosphate is greater than 1000 ppm at 12°C.

13) The most toxic constituent of commercially available dithiophosphates is hydrogen sulfide.

14) The median lethal concentration of Z-200 (isopropyl ethylthionocarbamate) is 45 to 48 ppm at 12°C.

15) The median lethal concentration of pure isopropyl ethylthionocarbamate is 40 to 45 ppm at 12°C.

Acknowledgments

The financial support provided by U. S. Bureau of Mines, the Dept. of the Interior, is gratefully acknowledged.

The authors also wish to thank Larry Ferber of the Cleghorn Fish Hatchery, Rapid City, S. D., and Dale Lambertson and Charles Sowards of the McNenny National Fish Hatchery, Spearfish, S. D., for their helpful suggestions and also for providing the fingerlings used in this investigation.

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- ¹⁰ Westfall, H.A., "Coagulation Anoxia in Fishes," *Ecology*, Vol. 28, 1945.
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VII

SURFACE PLANTS AND FACILITIES

- (a) Silvermines Mine Area, Big Glen District
- (b) Enon, Loch Lomond District
- (c) Road

SURFACE PLANT AND FACILITIES:(a) Silvermines Mine Area, Big Glen District
See Plan(i) Portal

The approach to the portal and decline will be on the regular proposed grade of minus 7 degrees or a minus gradient of 12.25%. This open cut through overburden will accommodate a 33 foot ballasted roadway with a 5 foot ditch allowance on both sides and a sump, outside the portal, to accommodate run-off water with a capacity of at least 100,000 Imperial gallons.

As the decline proceeds into the sub-outcrop the roadway will be narrowed to 16 feet and the rock walls trimmed neat to accommodate concrete walls and eventually a concrete roof. This concrete roof will continue until the rock roof thickness is sufficient to be self supporting. Ventilating air, high pressure compressed air, fresh water, mine discharge water pipes and electric power feed cables will be introduced to the decline through a short vertical culvert through the roof.

entire roof support bolts

(ii) Settling Ponds

We have no accurate forecast as to the quantity of mine water discharge but provision will be made for pumping 1,000 U.S. gallons per minute. This discharge water, together with run-off water from the open cut will be discharged to three settling ponds in a progressive flow. It is proposed the first two ponds will be each 200 feet diameter with an average depth of 8 feet, and a capacity of 1,800,000 U.S. gallons. The third pond will be 300 feet in diameter with an average depth of 6 feet and a capacity of 3,100,000 U.S. gallons. Thus provision will be made for a minimum of 4.75 days retention time before discharging to free flow and eventually to the Salmon River.

*ponds will
be limited?*

(iii) Stockpile

An area 200 feet wide by 300 feet long has been reserved west of the open cut draw to the portal, for an ore stockpile. At a height of 20 feet this will provide stockpile capacity for 60,000 tons of ore. Drainage from this stockpile area will be to the first settling pond.

(iv) Mine Plant Buildings

In the early stages of development and production a minimum of plant buildings are planned, these being a 10' x 52' office trailer, a 10' x 52' wash and dry trailer, a 40' x 60' Atco Fold-Away prefabricated building for shop and garage and necessary sewage septic tanks and tile drainage bed. A small frame pumphouse will

(iv) Mine Plant Buildings (Cont'd)

accomodate a water supply pump with a capacity of 100 U.S. gallons per minute.

Eventually the main plant building may have a floor area of 16,000 square feet. (80' x 200').

SURFACE PLANT AND FACILITIES

(b) Enon, Loch Lomond District

The description and information on surface plant and facilities at Enon listed on the following pages has been secured from: -

"Equipment Lists - Cost Centre #17 - Pages 1-2
Cost Centre #13 - Page 1
Cost Centre #20 - Page 1 "

as prepared by Kaiser Celestite Limited.

STATION NAME (1) Non (2) Utilities

Quantity	Equipment	Year	Manufacturer	Model	Size	Capacity	Weight	Serial No.
1	Electrical Power Substation	1970	Nova Scotia Power	69/12KV	36' x 40'			
3	Transformers				66/12KV			
	The above owned by Nova Scotia Power Corporation.							
1	Sewage Treatment Plant	1970	Smith Shoreless	Addigest	with 2HP Size 4R			
1	Concrete Pad	1970		SCA2-SC23	comminutor & sludge tank			
1	Building	1970						
1	Reaction Blower		Fuller-Sutorbilt	7L120-2L		25 CFM @ 3.5 psi		
1	Motor		Brook		Frame KS145T	1 1/2 HP 1720 RPM		59018T
1	Reaction Blower		Fuller-Sutorbilt	7L128-2L		25 CFM @ 3.5 psi		
1	Motor		Brook		Frame KS145T	1 1/2 HP 1720 RPM		59019T
1	Pump House				20' x 22'	Concrete block insulated, heated.		
1	Fire Pump	1971	V.M. White Fairbanks-Morse	433178	5314N	600 G.P.M. 90 psi		
1	Gasoline Motor	Used						
1	Process Water Pump	1974	Ingersoll-Rand	6 cyl. gasoline HC	4x3x8 HC			0673-4725
1	Motor	1974	Crompton Parkinson	A206TS	Frame AG31T	40 HP 3,500 RPM		
1	Process Water Pump		Ingersoll-Rand	3CRVH	3"	500 GPM		
1	Motor		C.G.E.	AB YF1405M	Frame 286T	40 HP 3520 RPM		
1	Process Water Booster Pump	1970	Ingersoll-Rand	2 CORVI		190 GPM		22764-K
1	Motor		U.S. Electric	AB		10 HP 1745 RPM		P4387758
1	Process Water Reclaim Pump	1970	Northington	8H-4E-3	4"			VTP 30917
1	Motor		U.S. Holic Shaft	Enclosed	Frame 256 UPH	10 HP 1,800 RPM		
1	Submersible Reclaim Pump & Motor	1971	Flygt Canada Ltd.	B-2151	4" H.H.	30 HP	330	P439126
1	Submersible Pump & Motor		Flygt	B-2200	6" H.H.	90 HP	1210	
1	Submersible Pump & Motor		Flygt		6" H.H.	90 HP		
1	Submersible Pump & Motor	1971	Flygt		6" H.H.	75 HP		
1	Pneumatic Diaphragm Pump		Tahco-Le Roi	DA-6	2 1/2" outlet	75 GPM		102x582
1	Domestic Water Pump Building (near concentrator)							
1	Domestic Water Pump							
1	Motor							
1	Process Water Storage Tank	1970				100,000 Gal.		
1	Domestic Water Storage Tank	1970	Drummond Welding		12'x12'	10,000 Gal.		Insulated & Aluminum Jacket 1971

SURFACE PLANT (b) Enon (11) Mining

<u>Quantity</u>	<u>Equipment</u>	<u>Year</u>	<u>Manufacturer</u>	<u>Model</u>	<u>Size</u>	<u>Capacity</u>	<u>Weight</u>	<u>Serial No.</u>
1	Calcestone Stockpile	1974 -75	Mined by Martell Construction			56,200 Tons		
1	Mine Sump		Kaiser Employees		10'x10'			
1	Mine Sump Pump with motor	1971	Stenburg-Flygt	B150L	6"	75 HP	1,210	
1	Mine Sump Pump with motor	1973	Flygt	B200L	6"	90 HP	1,210	
1	Mine Sump Pump with motor	1973	Flygt	B200L	6"	90 HP	1,210	
1	Settling Pond #5	1972			200'x200'	832,000 Gals.		
1	Settling Pond	1975	Martell Construction		80'x25'	2,500,000 Gals.		

SURFACE PLANT AND FACILITIES:(c) Road

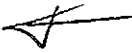
The gravel road which follows the south bank of the Salmon River between Marion Bridge and Enon is presently designated a Class H road, suitable for light traffic only. During the period of spring break-up in April and May of each year, sections of this road become impassable.

The Department of Highways have been carrying out a road improvement programme recently whereby sections of two or three miles of this road are given a top coating of coarse gravel which is graded in to widen the road somewhat and supply a gravel top which will push in to the soft surface. This programme has been progressing from the east, or Marion Bridge end of the road, toward the Silvermines area, and this year should see this improvement come within one mile of the proposed mine site. Such road improvement will be of considerable assistance in the movement of freight and equipment from Sydney via Highway 327 to the mine site.

However, we are faced with the problem of moving 4,000 tons of ore per week (675 tons per day on a 6-day week) which means 23 truck loads at 30 tons each in a 24-hour day from the mine site at the west end of the Silvermines area, a distance of 8 miles to the concentrator at Enon. Keeping in mind the need for an all-weather gravel road capable of handling this traffic, it is thought the road should be straightened wherever possible at minimum expense, and should be widened to a 33-foot top, with good ditches, a higher grade and improved drainage with rock ballast dumped into the road base in low, troublesome areas.

(c) Road (Cont'd)

Since there will be little time left for road improvement this year and since road improvement and efficient ore haulage is vital to the viability of the project, the following programme is suggested:-

1. Commence straightening and widening the road, with good ditches and improved culverts where necessary, commencing at the mine site and progressing to the west.
2. Establish stock pile areas on either side of the road at or near sections which are known to be soft and impassable during spring break-up.
3.  Yava Mines Limited now owns approximately 56,000 short tons of celestite ore which is located in the concentrator yard at Enon. Yava is willing to donate this ore, at no charge, to the road improvement programme. As per correspondence from the Department of Highways, samples of this celestite ore, as submitted by Mr. M. A. K. Grice, were tested and found to pass specifications for soundness and abrasion for gravel, Class C. It is suggested this material would probably be more suitable for ballast to build and establish a good base in the road.
4. We suggest this celestite ore be prepared for road ballast by coarse crushing same by a single

*What about Subtotal of all
Bill Grice's has done
cost to have
off pile sampled
to determine its
suitability.
Oct 27*

(c) Road (Cont'd)

pass through a portable jaw crusher to 4" to 6" size. A portable jaw crusher on a trailer with suitable feed hopper and feeder and a discharge conveyer could be moved into the concentrator yard at Enon as soon as possible and crushing and hauling of this coarse ballast to the various stock pile areas could continue with the probability this 56,000 tons could be crushed and moved before any heavy snowfall.

5. The tentative Yava programme and timetable calls for the commencement of development ore production by early January, 1978 and the haulage of ore to the concentrator yard at Enon where it will be stockpiled at the rate of 4,000 short tons per week. The overhaul of the concentrator and changes to the circuit should be completed from mid-December to the end of February, with concentrator operation commencing March 1, 1978. If ore haulage can continue on a regular basis through March, then sufficient stockpiled ore should be on hand to provide concentrator feed through the break-up period of April and May. During this time, road ballast should be introduced and where possible road improvement resumed so that ore haulage could be resumed on June 1st. Thus sufficient feed would be provided to the concentrator and the treatment rate of 500 tons per day on a continuous basis would not be interrupted.

(c) Road (Cont'd)

6. The suggested 33-foot road width would provide sufficient room for road improvement while truck haulage of ore continued.



NOVA SCOTIA

File
SEP 27 1977

Materials Laboratory
611 Portland Street
Dartmouth, N.S.
B2W 2M6
File No. II-22-77
1 SEPT 77.

M.A.K. Grice
Nova Scotia Technical College
P. O. Box 1000
Halifax, N.S.

Dear Mr. Grice:

RE: MATERIALS TESTING - LAB NO. 1957.

RECOMMENDATION:

Test results are shown in Table I.

BACKGROUND:

As per your request, the gravel sample from Keizer Mill, Salmon River submitted on the 8 Jun 77 had the following tests performed; Soundness of aggregate by use of sodium sulfate ASTM C88-73, Resistance to abrasion of small size coarse aggregate by use of the Los Angeles machine ASTM C131-69 and Procedure for the Petrographic Analysis of coarse aggregate CSA A23.2.30 1973.

DISCUSSION AND CONCLUSION:

This material passes our specifications for soundness and abrasion for gravel Class C with the exception of the grading requirement. The grading can be met by crushing the material to size and by bleeding off some of the fine material.

Yours very truly,

Frank A. Gervais
Frank A. Gervais, P.Eng.
Manager of Laboratory Services
EGD/jsp

cc: L. L. Centa
lab

Enclosure

what about sulphides?

SEP 14 1977

Source-Gravel from Keizer Mill

Abrasion 40%

Soundness 6.1%

Petrographic Number 603

SIEVE ANALYSIS

SIEVE SIZE	PERCENT PASSING	CLASS C SPECIFICATION
64 mm (2½")	100	
51 mm (2 ")	97	100
38 mm (1½")	78	
25 mm (1 ")	65	60 to 80
19 mm (¾ ")	57	
13 mm (½")	46	
9.5 mm (⅜ ")	40	
4.8 mm (No. 4)	30	25 to 45
2.4 mm (No. 8)	25	
1.2 mm (No. 16)	21	
0.60 mm (No. 30)	18	
0.30 mm (No. 50)	15	
0.15 mm (No.100)	13	0 to 10
0.074 mm (No.200)	10.9	

SEP 14 1977

VIII

ENVIRONMENTAL STUDY DATA

- (a) Baseline Statement by Dames and Moore
- (b) Updated Baseline Statement

ENVIRONMENTAL

- (a) Baseline Statement on Water Quality
by Dames and Moore

The report on the water quality survey as carried out by Dames and Moore in September 1974 is reproduced on the following pages.

WATER QUALITY ENVIRONMENTAL BASELINE STATEMENT

SALMON RIVER, NOVA SCOTIA

Prepared for: Yava Mines Ltd.

17 September 1974

by

Dames & Moore

6584-001

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- FIGURE 2 MAJOR SOIL TYPES
- FIGURE 3 LOCATION OF DAMES & MOORE SAMPLING STATIONS
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INTRODUCTION

The ore body comprising up to seven million tons of reserves contains 3.5 to 4% lead, and is located close to the Salmon River near Silver Mine (45° 51.6'N, 60° 24.3'W) in Cape Breton Co., Nova Scotia

Lead is present as galena, with zinc usually associated with lead, present only as a minor constituent.

Dames & Moore have carried out a water quality survey of the vicinity to establish the baseline conditions for Yava Mines. During the sampling procedure, additional environmental data was gathered and is presented here as background information to the water quality study.

Current concern for the protection of the environment, recognizes the increased impact of man's technological advances on seemingly delicate interactions between organisms. These first became apparent in the late nineteenth century and were further developed in the thirties with the popularity of conservation practises. These came to the fore, as organizations concerned about water conservation and erosion control; practises now form part of basic engineering philosophy.

With the development of more sophisticated analytical and experimental techniques it became apparent during the 60's that many minor constituents such as heavy metals, organic materials and stack gases affect the environment by causing disease, interfering with reproduction, affecting food chains and growth and disintegrating tires,

clothing, etc.

Development of natural resources involves therefore an element of social cost which may be in terms of aesthetics or economics. Environmental studies in their broadest sense are an attempt to summarize existing or ambient costs with those expected to accrue from the benefits and hazards of project developments.

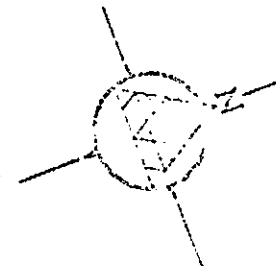
Man, wishes increased control of the environment. Control in this context implies gathering and evaluating more information about his surroundings. The most relevant aspect of this is the formulation by federal and provincial government agencies of new regulations to control mining activities in relation to the environment.

GEOLOGY

The rocks underlying the Salmon River area are of Palaeozoic age. (Figure 3) An extensive mantle of drift limits the amount of outcropping to a few widely scattered points mostly associated with creek beds. A band of Windsor limestone runs parallel to the Salmon River which passes through the area.

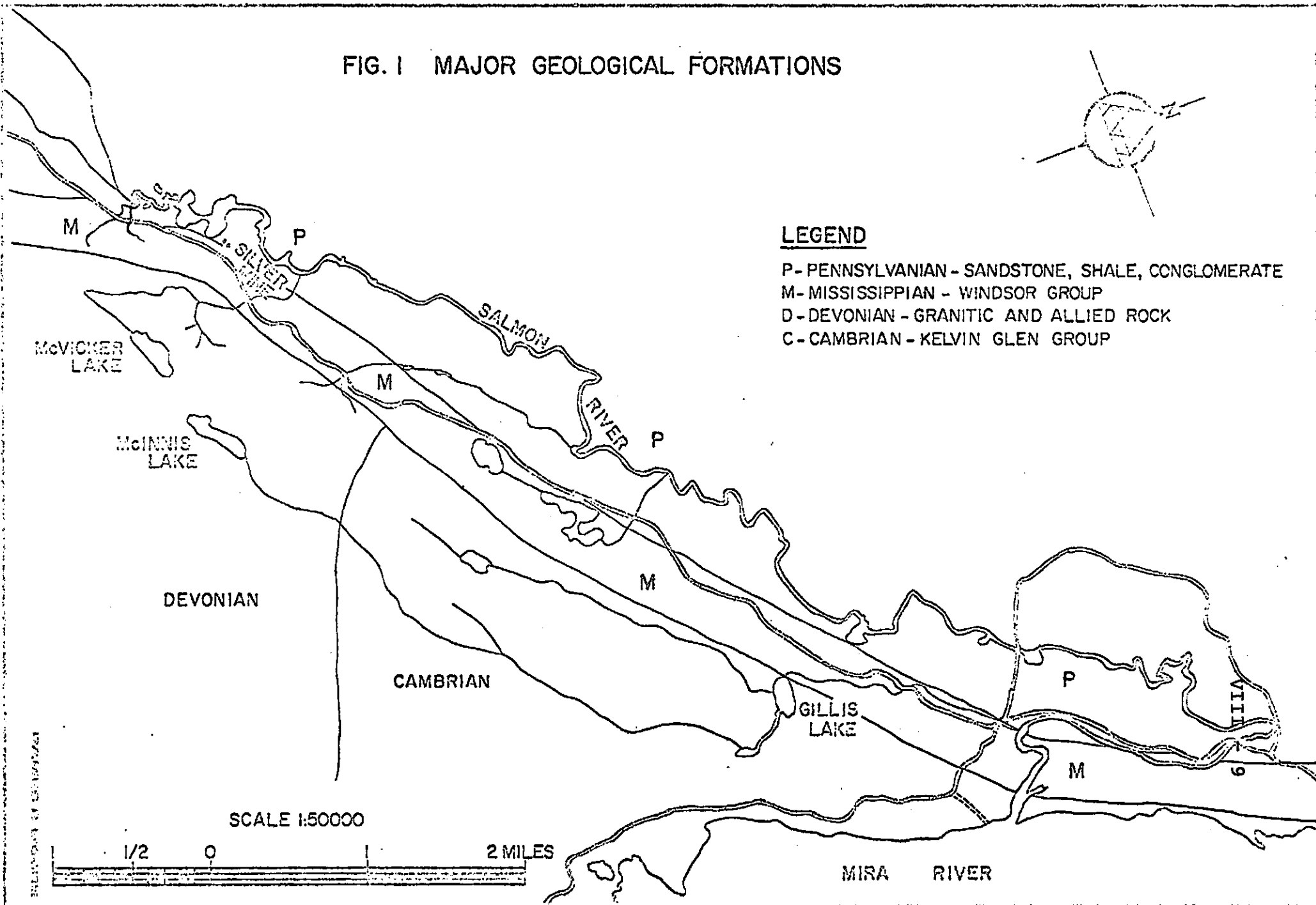
Glacial drifts comprise unsorted glacial material and sorted materials in kames, eskers and outwash plains. Generally the glacial and glaciofluvial deposits are derived from fine grained reddish-brown Carboniferous rocks. Some silt size particles are mixed with the material reducing permeability at depth. Some moderately coarse textured overburden is present but more restricted in distribution.

FIG. 1 MAJOR GEOLOGICAL FORMATIONS



LEGEND

- P- PENNSYLVANIAN - SANDSTONE, SHALE, CONGLOMERATE
- M- MISSISSIPPIAN - WINDSOR GROUP
- D- DEVONIAN - GRANITIC AND ALLIED ROCK
- C- CAMBRIAN - KELVIN GLEN GROUP



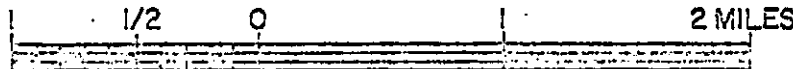
DEVONIAN

CAMBRIAN

GILLIS LAKE

MIRA RIVER

SCALE 1:50000



SOILS

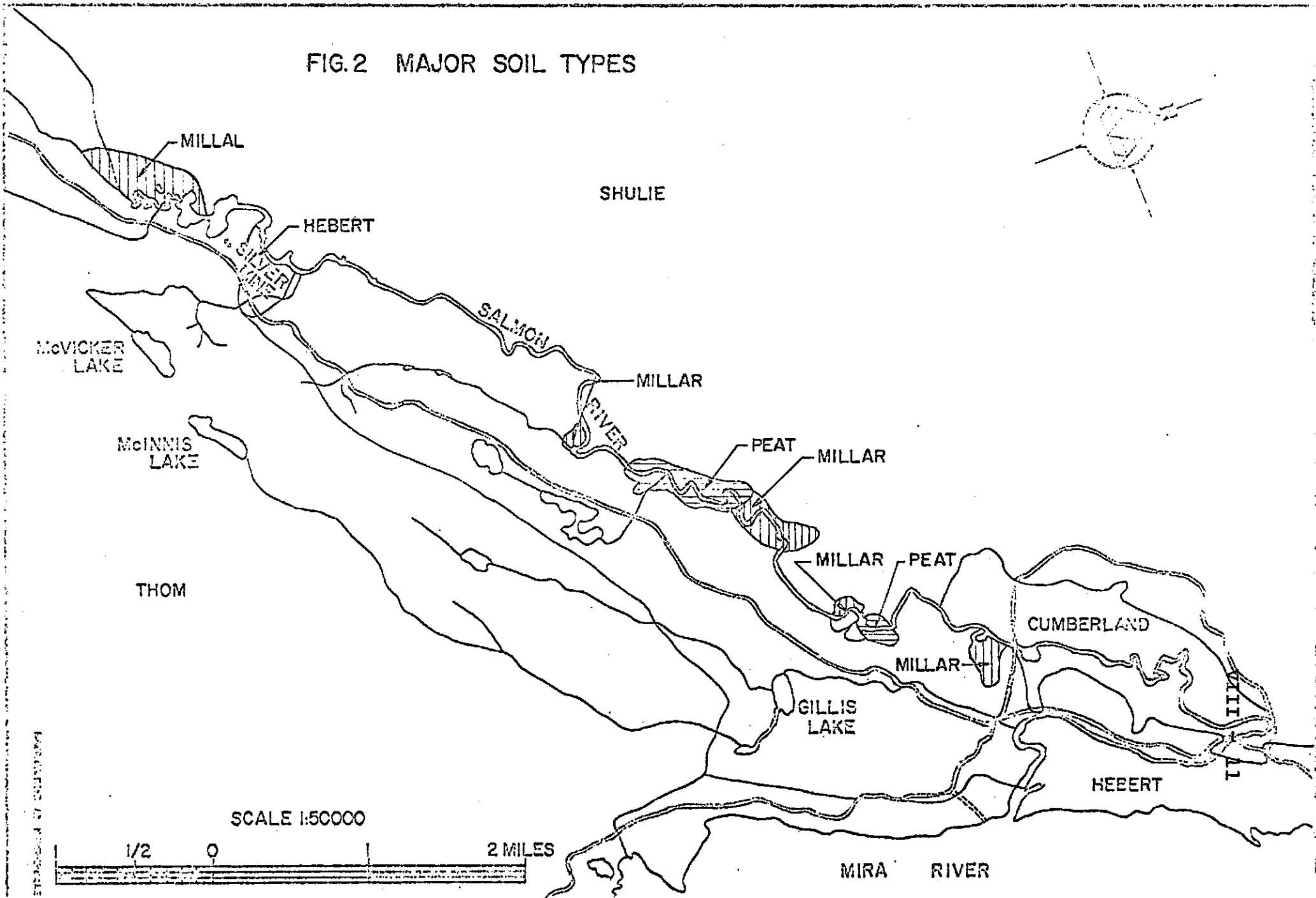
The soils in the vicinity of Silver Mine are all podsoils and consist mainly of three series (Figure 2):

- 1) The Hebert series, derived from greyish brown gravelly sandy loam stratified materials deposited by glacial streams;
- 2) The Thom soils are stony and rarely cultivated. Some limited pasturage is used, however, because of their natural acidity, liming and fertilization are required for successful crop production. The Thom series are prone to erosion and best suited for growth of timber.
- 3) Shulie soils are derived from brown sandy loam till and are very acid (pH 3.5-4.5) and porous. As a result these soils have good surface drainage and moderately rapid internal drainage.

Drainage from these three soils is likely to be acidic and to contain considerable amounts of iron leached out of the surface soil by the acidic surface water. Levels of nitrate, phosphate and calcium removed from the soil are likely to be low, so that most calcium would be derived from calcite in the overburden.

Ammonia would be expected as the more abundant nitrogen compound removed from the soil.

FIG.2 MAJOR SOIL TYPES



VEGETATION

Vegetation is important to soil development. It supplies organic material and is vital in recycling of nutrients in ecosystems. Vegetation retards processes of erosion and consequently affects water quality parameters such as dissolved and suspended solids.

The majority of the mine vicinity is covered by conifers (softwoods). White and red spruce with balsam fir are the main tree species.

The humus layers in the forest is of the acidic duff type. Rates of nutrient cycling are slow and acidity develops in the upper horizons rapidly. Low nitrogen and phosphorous mineralization occur in such soils.

HYDROLOGY

The Salmon River drains an area of approximately 76.8 square miles. The topography in the area ranges from gently undulating to hilly. During the Dames & Moore field study four active creeks were identified draining the region of the proposed mines. These creeks were named starting from Silver Mine; first, second, etc. for this report.

Flow in the Salmon River itself is low and variable and does not peak consistently at any particular time of year. (Table 1 - 3) Flow estimates were made at stations 1 and 3. (Table 4) Discussion with local residents indicated that the water level of the river was low during our study period.

Soils and overburden are moderately to freely draining with moderate erosion hazards with Thom soils on steeper slopes. Little erosion is visible in the mine site area.

Some peat soils are present in the area and have high water tables and low permeabilities (hydraulic conductivity). Peats retain heavy metals and are potentially useful materials in water quality control.

TABLE 1

MONTHLY MEAN DISCHARGE OF SALMON RIVER
AT SALMON RIVER BRIDGE, ENVIRONMENT CANADA
GAUGING STATION

YEAR	JAN	FEB	MAR	APR	MAY	JUNE	JULY	AUG	SEPT	OCT	NOV	DEC
1973	337	552	405	430	540	357	291	358	83.0	129	353	435
1972	468	338	602	309	619	265	97.4	106	119	527	662	432
1971	109	655	323	899	154	95.3	30.0	102	231	101	512	371
1970	71.5	176	130	410	134	214	118	352	287	362	519	263
1969	356	411	249	373	248	133	108	59.4	80.5	177	1020	638
1968	217	146	543	343	154	81.0	50.9	115	192	93.6	624	497
1967	209	82.8	422	596	1160	63.4	17.6	60.2	82.7	319	563	406
1966	101	267	450	311	207	130	159	85.0	117	448	208	439

All readings in CFS

TABLE 2

MONTHLY MAXIMUM DISCHARGE OF SALMON RIVER
AT SALMON RIVER BRIDGE, ENVIRONMENT CANADA
GAUGING STATION

YEAR	JAN	FEB	MAR	APR	MAY	JUNE	JULY	AUG	SEPT	OCT	NOV	DEC
1973	1170	3370	796	921	1670	3530	1820	1360	142	605	1430	1300
1972	1040	1870	2330	753	2440	2160	423	400	400	1320	2590	1830
1971	273	5690	1660	2560	529	237	110	400	2050	645	1760	1320
1970	323	610	599	1570	242	1180	469	2570	1220	1620	4110	723
1969	356	411	249	373	248	133	108	59.4	80.5	177	1020	638
1968	217	146	543	343	154	81.0	50.9	115	192	93.6	624	497
1967	533	172	730	1350	3640	276	32.4	396	630	1710	1560	1980
1966	527	918	983	753	473	396	694	346	424	1160	458	2260

All readings in CFS

TABLE 3

MONTHLY MINIMUM DISCHARGE OF SALMON RIVER
AT SALMON RIVER BRIDGE, ENVIRONMENT CANADA
GAUGING STATION

YEAR	JAN	FEB	MAR	APR	MAY	JUNE	JULY	AUG	SEPT	OCT	NOV	DEC
1973	80.2	54.1	60.0	249	188	48.7	43.0	72.5	45.5	39.6	82.2	113
1972	178	95.8	90.0	110	65.3	44.1	18.0	35.0	48.0	150	176	144
1971	56.0	62.9	109	250	69.8	36.5	15.0	11.0	27.0	26.3	37.2	97.4
1970	30.8	54.8	45.8	189	47.5	32.8	28.8	42.5	82.3	67.9	98.4	59.2
1969	81.3	80.7	53.8	133	62.0	24.5	7.8	5.5	3.2	52.6	61.9	222
1968	83.1	75.8	81.6	104	18.3	20.0	8.5	6.4	23.2	15.9	61.8	131
1967	60.5	53.9	189	296	314	20.5	8.0	8.0	6.9	37.7	257	75.8
1966	43.2	49.3	114	95.1	60.0	40.8	14	4.7	9.6	106	111	124

All readings in CFS

METHODS

Field investigations were conducted in the Salmon River and adjacent streams from August 19 to 21 of this year. Ten sampling stations were selected to develop a comprehensive statement of the condition of surface water in the area. (Figure 1)

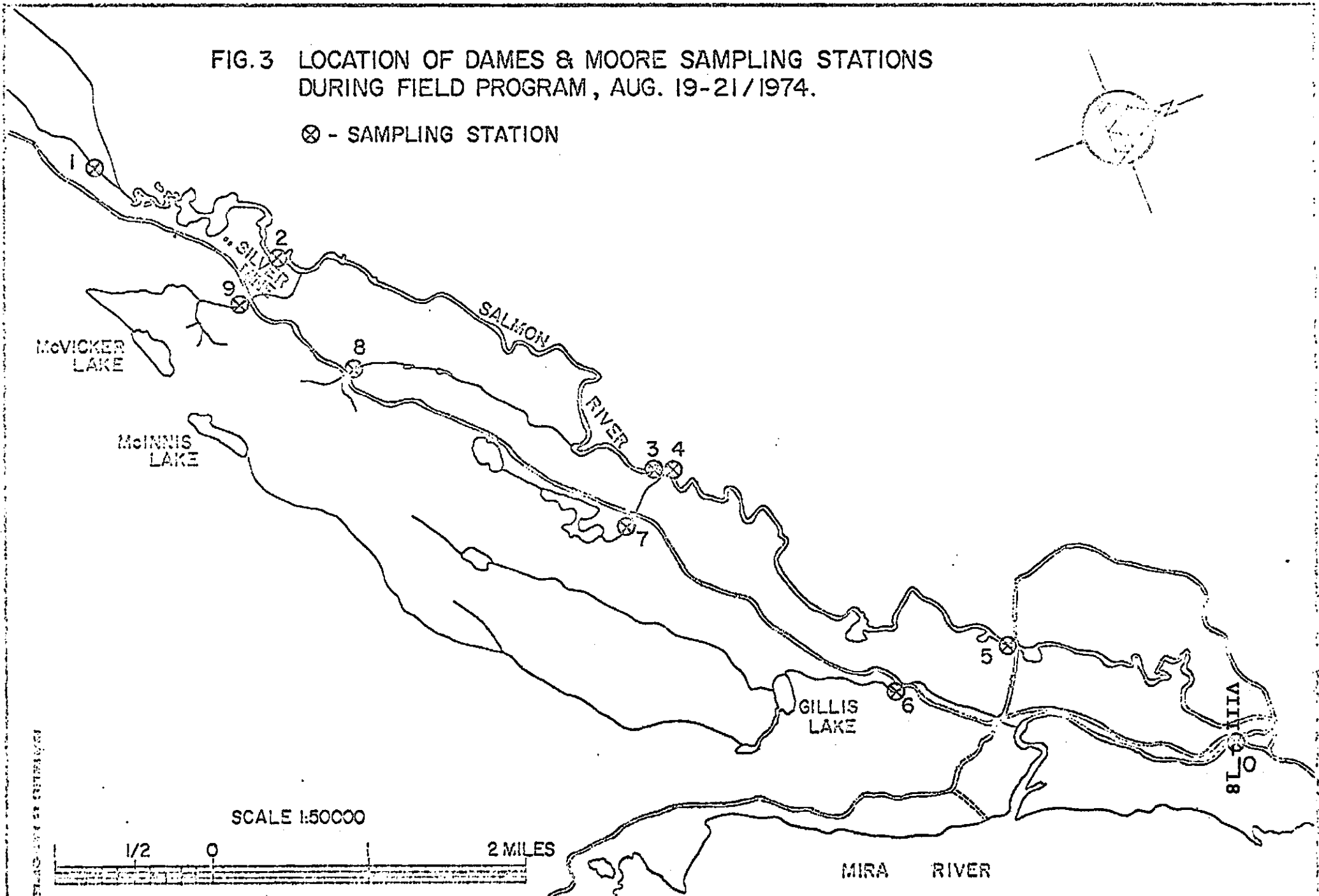
Water samples were collected in polythene bottles. Samples to be analyzed for C.O.D., Nitrate - N and Kjeldahl - N, were preserved with sulfuric acid while those for metal analysis were preserved with nitric acid. Sediments samples were collected in plastic bags. All samples were flown to laboratories in Toronto within 72 hours of collection. Analysis was completed within two weeks and all samples were refrigerated during this time.

Water samples were analyzed in accordance with selected procedures contained in "Standard Methods for the Examination of Water and Wastewater", 13th edition, 1971. Metallic constituents, with the exception of lead and nickel, were determined by atomic absorption spectroscopy on concentrated samples. Lead and nickel were determined by colorimetric techniques. The limit for detection of chromium was 0.01 mg/l, copper was 0.01 mg/l, lead was 0.02 mg/l and nickel was 0.05 mg/l.

Components subject to deterioration in storage, specifically pH, suspended solids, ammonia and nitrate-nitrogen, and C.O.D. were determined within two days of arrival of the samples in Toronto. The limit of detection for nitrate-nitrogen was 0.05 mg/l and ammonia-nitrogen was 0.05 mg/l.

FIG. 3 LOCATION OF DAMES & MOORE SAMPLING STATIONS DURING FIELD PROGRAM, AUG. 19-21/1974.

⊗ - SAMPLING STATION



Sediment samples were analyzed for lead, mercury and zinc by atomic absorption spectroscopy on a solution obtained by digestion in nitric acid (Methods for Chemical Analysis of Waters and Wastes (1971), Environmental Protection Agency).

In addition to the collection of samples, measurements of pH and conductivity were made during the field and water sampling study.

The following account treats some of the environmental factors which are important in a consideration of the water quality study.

USES OF SALMON RIVER

The section of the Salmon River above The Interval Road bridge is used infrequently for recreational fishing and boating. Two operating farms were observed facing on the river between stations 1 and 5. Both had small cattle herds which could use the river for drinking. Apart from wildlife in the area which may consume the waters, no other uses of the Salmon River were observed.

WATER QUALITY

In general, the waters of the Salmon River and four adjacent creeks tend to be slightly acidic, low in alkalinity and hardness.

(Table 5) Ideally alkalinity should not normally drop below 20 mg/l to adequately protect aquatic biota from sudden changes in pH. The waters are strongly affected by the acidic Shulie soils in the area. Because of low bicarbonate levels the river has a limited capacity to accept additional acids without lowering the pH excessively below acceptable levels of about 6.5. Hardness is defined as a water quality parameter which represents the total concentration of calcium and magnesium ions in solution expressed in terms of calcium carbonate. The present of Windsor limestone in the region appears to have little influence on the river due to the covering of drift material.

Suspended solids concentration is a measure of the amount of material in the water that can be removed by physical means. Dissolved solids includes all constituents in solution, except organic colloids or dissolved gases. The conductance of a sample is an approximate measure of the total concentration of ionised salts dissolved in the stream water. Station 1 was selected to be above the influence of the proposed mine; however, levels of dissolved solids and conductance are well above readings taken at stations further downstream. Higher chlorides and sulfates concentrations at Station 1 contribute to this situation.

Sedimentary rocks and the action of precipitation flushing sulfur compounds from the atmosphere are major sources of sulfates in surface water. In the absence of dissolved oxygen bacteria may reduce

sulfates to sulfides which may react to form insoluble iron sulfide or hydrogen sulfide gas. This process occurring in the two lakes at Silver Mine could account for the lower sulfate levels at Station 4.

The most important source of chlorides in coastal regions in surface waters is precipitation where minute salt crystals from sea spray constitute the nuclei for rain drops rather than dust particles as further inland. Chlorides in concentration above 250 mg/l may impart a salty taste to water in the presence of sodium ions.

Phosphate levels along the entire river and in the creeks were moderate (greater than .15 mg/l as PO_4). The presence of excess phosphorus encourages the growth of some nuisance algae however only typical stream algae (as diatoms and desmids) were observed as a dark brown scum on rock in the stream bed at most stations. The moderate phosphorus levels may be due in part to abundant plant growth and decay in and about the river and the extreme low flows during the survey.

Analysis for heavy metals revealed high levels of iron at Stations 6 and 8 and the second and fourth creek respectively. (Table 6) Concentrations in excess of 0.3 mg/l of iron impart a bitter taste to water. Iron sulfide discolouration in bottom material was noticeable at Station 6. The iron at Station 8 was likely present in the suspended solids which were higher than average in this small creek.

High levels of lead were detected at Station 4. Lead in drinking water above 0.05 mg/l is considered hazardous. It should be noted at the same time that concentrations of zinc were well above ambient

levels for the region at this station. These metals were almost certainly contained in the suspended solids which were at their highest levels here of any of the stations sampled. Station 4 at the mouth of the third creek, station 7, in the creek at the Salmon River Road, and station 3 in the Salmon River above the influence of the third creek do not reflect these same levels of zinc and lead.

Concentrations in excess of .3 mg/gm of mercury, 60 mg/gm of lead and 200 mg/gm of zinc in sediments can be considered high. The levels of lead and zinc in sediments at station 4 are high in spite of the peat soils around the mouth of the creek which would be expected to filter out soluble metals. (Table 7) Lead was present in sediments in extremely high amounts at station 8 in the second creek at the Salmon River Road. The control station 1 beyond the influence of the drainage area of the proposed mine site exhibited lower levels of both zinc and lead in sediment samples. Mercury at all stations was well below levels for concern.

To obtain a picture of the seasonal characteristics of water quality in the Salmon River, the National Water Quality Data Bank of the Inland Waters Directorate was consulted. The station maintained by Environment Canada coincides with Dames & Moore station 10. No specific pattern or seasonal peaks were observed. During 1973, alkalinity, dissolved solids, hardness and turbidity remained low throughout the year. (Table 8) The river water was acidic at all times. Analysis for heavy metals revealed that iron and manganese concentrations frequently reach levels that could impart a bitter taste to the water. (Table 9) Zinc, copper and lead were present in very low concentrations.

TABLE 4

FIELD MEASUREMENTS

Station No.	Date	Time	Temperature (°F)	Conductivity (micromhos at 25° C)	pH	Turbidity/ Colour	Flow rate (cFs)	Comments
1	8/20/74	18:50	-*	208	6.75	Clear	15.0	River bed 36 ft. wide, river width 21 ft., 8 inches avg. depth
2	8/21/74	12:00	74	96	6.25	Clear	-	200 feet wide, avg. depth approx. 4 ft.
3	8/20/74	17:10	-	72	6.75	Clear	32.8	White foam on surface, bottom gravel, 6" boulders, 40 feet wide, 1 ft. deep
4	8/20/74	16:30	-	78	6.82	Clear	-	Creek 15 feet wide at entrance to river
5	8/20/74	11:00	70	76	6.65	Clear	-	Many reeds, brown scum on bottom, avg. depth 6 feet
6	8/21/74	10:05	-	70	6.55	Clear	-	Black scum on bottom 20 feet wide, avg. depth one foot
7	8/20/74	15:20	68	82	6.75	Clear	-	7 feet wide, bottom brown mud and leaves
8	8/20/74	19:30	-	79	6.30	Clear	-	5 feet wide, 8 inches deep
9	8/21/74	11:00	-	44	6.30	Clear	-	Black scum on bottom, 5 feet wide, 6 inches deep
10	8/21/74	15:20	72	59	6.65	Clear	-	Dark brown sediment, location of Environment Canada gauging station

* not recorded

TABLE 5

WATER ANALYSIS, GENERAL PARAMETERS

Station No.	pH Value	Alkalinity (as CaCO ₃)	Hardness (as CaCO ₃)	Solids		Chloride (Cl)	Sulphate (SO ₄)	Phosphate (PO ₄)	Nitrate (N)	Ammonia Nitrogen (N)	C.O.D.
				Suspended	Dissolved						
1	7.1	12.0	57.0	4.0	148	37.0	29.0	0.25	N.D.	N.D.	8.2
2	6.7	6.4	22.6	2.8	54	12.2	7.6	0.61	N.D.	N.D.	15.2
3.	6.8	6.0	19.0	1.2	52	10.6	6.4	0.38	N.D.	N.D.	11.6
4	7.0	11.5	25.4	9.2	57	9.0	5.8	0.31	N.D.	N.D.	9.0
5	6.8	8.8	21.0	6.4	56	10.0	5.5	0.44	N.D.	N.D.	10.8
6	6.8	17.3	25.0	1.6	59	5.0	5.0	0.48	N.D.	N.D.	15.2
7	7.0	12.3	26.3	2.8	54	8.3	6.8	0.25	N.D.	N.D.	8.4
8	6.5	15.9	16.2	7.6	52	5.7	0.1	0.58	N.D.	N.D.	7.0
9	6.6	6.6	11.3	0.8	31	5.5	0.3	0.20	0.1	N.D.	4.0
10	6.7	7.2	19.0	nil	54	8.0	4.4	0.28	N.D.	N.D.	9.6

N.D. = Not Detected

(All values, except pH, are calculated as milligrams/litre)

TABLE 6

WATER ANALYSIS, METALS

STATION	Chromium (Cr)	Copper (Cu)	Iron (Fe)	Lead (Pb)	Nickel (Ni)	Zinc (Zn)
1	N.D.	N.D.	0.18	N.D.	N.D.	0.02
2	N.D.	N.D.	0.28	N.D.	N.D.	0.01
3	N.D.	N.D.	0.18	N.D.	N.D.	0.02
4	N.D.	N.D.	0.22	0.14	N.D.	0.42
5	N.D.	N.D.	0.30	N.D.	N.D.	0.01
6	N.D.	N.D.	0.37	N.D.	N.D.	< 0.01
7	N.D.	N.D.	0.05	N.D.	N.D.	0.02
8	N.D.	N.D.	0.96	N.D.	N.D.	0.01
9	N.D.	N.D.	0.06	N.D.	N.D.	< 0.01
10	N.D.	N.D.	0.28	N.D.	N.D.	< 0.01

N.D. -- not detected
All values are in milligrams/liter.

TABLE 7

TOTAL LEAD, MERCURY AND ZINC LEVELS IN THE
RIVER SEDIMENT IN THE YAVA MINES VICINITY

Station No.	Lead (Pb) ug/g Dry Basis	Mercury (Hg) ug/g Dry Basis	Zinc (Zn) ug/g Dry Basis
1	37.6	0.035	98
2	58.4	0.034	218
4	262.0	0.098	608
5	106.0	0.068	173
6	24.5	0.009	144
8	822.0	0.029	175

1 ug/g = 1 ppm = 0.001 mg/g

DISCUSSION

The existing heavy metal levels occurring normally in sediment and water samples from the Salmon River vary from well below to above recommended guidelines. At this time it is unclear whether such values are a result of sporadic outcrops or are related to seasonal flow variation. Data collected by Environment Canada indicate seasonal variation in total dissolved solids of nearly 50% from the mean (25.8 mg/l). Such variation would be unlikely to result in the observed variations. The high levels of lead and zinc observed from site 4 but not from adjacent stations 3 and 7 would appear to indicate outcropping or water seepage from an ore body zone to the station. Such levels are to be regarded as normal to the stream.

The mean levels of lead (0.014), chromium and copper are well within recommended levels for receiving water objectives. Levels for both zinc and iron should be examined more carefully. The Salmon River is a soft water system containing less than 30 mg/l total hardness (as Ca CO_3); Ontario recommended receiving water levels should not exceed 0.03 mg/l zinc and 0.3 mg/l iron. In both instances these are exceeded as averaged data for portions of the river.

In note #3 of the mine-mill effluent guidelines (Ontario) it is recommended that "where the natural background concentration in the receiving watercourse exceeds the value given for any parameter in Table 2, the natural background concentration will become the receiving water quality objective." In the table below we give a comparison between these objectives and data collected by Environment Canada and Dames & Moore.

TABLE 8A
(continued)

ENVIRONMENT CANADA WATER QUALITY DATA ON
SALMON RIVER AT SALMON RIVER BRIDGE (STATION 10)

Date	Dissolved Calcium (mg/l)	Dissolved Magnesium (mg/l)	Dissolved Potassium (mg/l)	Total Alkalinity as CaCO ₃ (mg/l) ³	Dissolved Chloride (mg/l)	Dissolved Sulphate (mg/l)	Dissolved Nitrate and Nitrate (mg/l)
2/3/73	1.40	0.7	0.3	1	5.4	4.4	0.040
4/3/73	1.80	0.5	0.3	- *	4.4	7.2	0.020
5/25/73	1.40	0.4	0.3	2	2.9	5.0	.001
6/11/73	3.00	0.8	0.4	5	5.6	7.5	.001
7/15/73	3.30	0.9	0.5	7	6.5	5.0	.001
8/12/73	2.50	0.7	0.5	1	4.7	9.0	.001
9/9/73	3.60	0.9	0.4	7	6.8	6.0	0.010
10/9/73	5.00	1.1	0.6	7	9.9	9.0	0.020
11/29/73	3.20	1.0	0.4	3	8.5	9.0	0.030

* - not recorded

TABLE 8B

ENVIRONMENT CANADA WATER QUALITY DATA ON
SALMON RIVER AT SALMON RIVER BRIDGE (STATION 10)

DATE Date	Temperature (°C)	pH	Apparent Colour	Turbidity (JTV)	Specific Conductance (umkos/cm)	Total Dissolved Solids (mg/l)	Total Hardness as CaCO ₃ (mg/l) ³
2/3/73	2.2	4.9	30	1.0	32	18	6.4
4/3/73	2.2	4.4	30	1.0	46	-*	6.6
5/25/73	5.6	5.6	50	1.9	25	15	5.1
6/11/73	18.3	6.0	30	1.2	43	25	10.8
7/15/73	20.0	6.3	50	1.1	47	26	11.9
8/12/73	20.0	4.8	100	0.8	38	24	9.1
9/9/73	16.7	6.6	40	0.7	50	29	12.7
10/9/73	9.9	6.3	30	1.4	65	37	17.0
11/29/73	7.8	6.2	30	12.0	50	32	12.1

* - not recorded

TABLE 9

ENVIRONMENT CANADA SEDIMENT ANALYSIS DATA ON
SALMON RIVER AT SALMON RIVER BRIDGE (STATION 10) - METALS

Date	Extractable Zinc (mg/l)	Extractable Manganese (mg/l)	Extractable Iron (mg/l)	Extractable Copper (mg/l)	Extractable Lead (mg/l)
1/5/70	0.02	0.14	0.330	0.05	- *
4/6/70	.01	0.08	0.130	0.02	-
7/2/70	.01	0.08	0.390	0.04	-
8/1/70	.01	0.08	0.550	0.05	-
11/8/70	-	0.03	0.450	0.03	-
2/13/72	-	0.08	0.200	.001	0.005
5/2/72	-	0.08	0.180	.001	0.002

* - not recorded

TABLE 10

	Receiving Water Quality Objective	Dames & Moore (mean)	Dames & Moore Station 10	Environment Canada (mean) 1973-74
Suspended solids	15 mg/l	3.60	nil	
pH	5.5-10.6	6.80	6.70	-
Ammonia	1.50	N.D.	N.D.	-
Copper	0.03	N.D.	N.D.	-
Chromium	0.50	N.D.	N.D.	-
Iron	0.30	0.29	0.28	0.26
→ Lead	0.05	0.01	N.D.	-
Nickel	0.40	N.D.	N.D.	-
Zinc	0.02	0.05	0.01	-
Phosphate	-	0.38	0.28	0.01 (dissolved inorg)

TABLE 11

	Receiving Water Quality Objectives (Ontario Guidelines)	Average of D. & M. Samples	Highest Concentration Analyzed (Station)
Ammonia	1.50	N.D.	N.D.
Copper	0.03	N.D.	N.D.
Chromium	0.50	N.D.	N.D.
Iron	0.30	0.29	0.96 (8)
Lead	0.05	0.01	0.14 (4)
Zinc	0.02	0.05	0.42 (4)

APPENDIX I

ECOLOGICAL DESCRIPTION OF DAMES & MOORE
SAMPLING STATIONS

Site 1

At this station the Salmon River is 20 - 30 feet wide and up to 6 feet deep. At the time of sampling there was considerable turbulent water movement. The substratum was composed of stones and cobbles with no gravel or sand present. Abundant green algae including diatom growth occurs on these boulders.

No floating leaved plants were found. Submerged species were represented by pondweeds (Potamogeton zosteriformis and P. crispus).

Water striders were present in the quieter areas but aquatic invertebrates generally were low in numbers (predominately mayfly larvae (Heptagenia spp.)).

The bank vegetation was forested with balsam fir (Abies balsamea), white spruce (Picea glauca) and red maple (Acer rubrum). There were patches of alder (Alnus rugosa) with several ferns and white topped aster (Aster umbellatus).

Site 2

This station is by the widening of the Salmon River to over 200 feet. The river is shallower with a maximum depth of 34 inches. The substratum is composed of coarse sand and gravel except in some small bays which have a peat layer. Large amounts of green algae and diatoms were evident throughout the river with many submerged aquatics, such as

Canadian Pondweed (Elodea canadensis) and Pondweed (Potamogeton filiformis) forming dense beds. Floating leaved species such as white water lilies (Nymphaea alba) were abundant. The diversity of emergent species is particularly noteworthy. A partial list includes Pickerelweed (Pontederia cordata), Eriocaulon septangulare, Dulichium arundinaceum, Juncus articulatus, Typha latifolia, Carex intumescens and Spiraea latifolia.

Abundant minnows (including red-belly dace) were seen. Abundant clams, gammarids and other invertebrates were found. Few caddisfly were evident and no gerrids were seen.

The marginal vegetation was predominantly alder thickets (carrs) with some willows (Salix spp). The communities were being invaded by young balsam fir. The understory consisted of grass and sedge dominated communities with Carex stricta and Calamagrostis canadensis being most abundant. Several asters including the New England aster were common. At this station considerable bird populations with boreal chickadee, yellowthroats, and red-breasted nuthatch were noted.

Site 3

This site is very similar to Site 4. The river here is about 40 feet wide and averages about 10 - 12 inches deep. The site location is between the bank and a shingle island covered with alder and yellow birch (Betula lutea). The river bed was rocky with small sand and gravel sections. The presence of Canadian Pondweed and some Potamogeton spp indicated high nutrient loading. Polygonum amphibium was abundant.

The invertebrate fauna was abundant with numerous Caddisfly larvae attesting to good water quality. Many clams (Margaritacea sp) were found.

The marginal forest comprises red maple, balsam fir with red (Picea rubens) and white spruce.

Site 4

The site is located at the confluence of the Salmon River and a small creek (Site 7). The substratum is gravelly with some cobble sections. The river reaches a width of 45 feet before narrowing. Maximum depths were about 18 inches. The larger stones had some algae growth (diatomaceous). Submerged species were few, covering less than 1% of the river substratum. These were water cress (Nasturtium officinalis) and pondweed (Potamogeton filiformis). No floating leaved species were recorded and emergent plants were limited to cattails and pickerelweed.

The invertebrate fauna was rich in species, but poor in numbers. Water striders, water spiders, caddisfly, mayfly and stonefly larvae were found. Some tubificids were collected.

The marginal vegetation comprises sugar maple (Acer saccharum) mountain (A. spicatum) and red maples, white spruce and alder were abundant.

Site 5

The bottom sediment here is loose organic silt. Algae growth covers sediment and plants. A great amount of submerged species (pondweeds) are present, many are covered with freshwater sponges. No invert-

brates were collected.

Several minnows and small fish were observed. About 30% of the river is covered with emergents and water lilies. Common species include pickerelweed, cattail and water horsetail (Equisetum fluviatile). The banks are grassed with occasional white spruce and tamarack (Larix laricina).

Site 6

This site is located just downstream from a beaver dam. The maximum depth was 27 inches. The river was about 20 feet across with a gravel substrate. Many logs and branches slow down flow rates so that in shallower areas there is a build up of organic silt. Sponges and green algae growth were found on vegetation. Diatoms predominate on the few rocks.

Floating leaved pondweed (Potamogeton natans). Some water lilies and some cattail were found but generally vegetation was in poor condition. Some small fish, chub and dace were observed.

Other plants found marginally include Royal Fern (Osmunda regalis) Potamogeton pectinatus, P. interior, Sensitive fern (Onoclea sensibilis), Calamagrostis. Tree species on the terrestrial site include balsam fir, yellow birch, red maple and white spruce.

Site 7

This is located by a creek 4 - 8 feet wide and 18 inches deep, flowing with the Salmon River. The substratum is sand or fine gravel

with no evident growth of algae.

Cattails, water cress, curly leaved pondweed Potamogeton crispus. Invertebrates are represented mainly by water striders, caddisfly. Some red-belly dace were observed.

The marginal plants include white spruce, red maple and yellow birch. Sweet fern (Comptonia peregrina) white topped aster, blue cohosh (Caulophyllum thalictoides) and sedges are abundant.

Site 8

Represents a small stream with no visible flow draining a bog area. The stream was 42 inches deep and two feet across. No invertebrates were observed. Large numbers of frogs in the area (leopard frog Rana pipiens).

Vegetation includes bog mosses (Sphagnum) pitcher plants (Sarracenia purpurea) with young birch, tamarack, sweet gale (Myrica gale) and many other species.

Site 9

Stream about 5 feet wide and 6 inches deep. Some rocky sections with diatom algae growth. No submerged, floating or emergent species. The invertebrate fauna is sparse consisting of mayfly larvae and several species of water strider. No caddisfly larvae.

The marginal vegetation consists of red maple, balsam fir, red spruce and yellow birch. The understory is composed of sedges (Carex rostrata) and grasses. There are some mature tamarack and white pine

in the area.

Site 10

Bottom consists of stones and cobbles with diatoms and other algae covering about 40% of the area. Some invertebrates were collected including caddisfly and mayfly larvae.

Red spruce with yellow and paper birch formed the forest cover on the banks.

The Salmon River is a diverse river system with a great variety of plants and animals. Deer use the river area extensively, and numerous browse signs and droppings were observed at nearly every location.

As a fishing river, it is not highly esteemed locally, however it appears to contain some salmon. However, the abundant fish appear to be shad, perch, black bass and eel. The availability of better fishing areas locally limits its recreational use.

APPENDIX 2

YAVA MINES - CLIMATE BASELINE

The nearest climatic station to the mine site is approximately 20 miles to the northeast at Sydney. The meteorological data acquired for the Sydney station should be directly applicable because of only minor differences in elevation and exposure with respect to predominant southwest winds. Seasonal differences from year to year at the site depend upon the intensity and frequency of passing synoptic storms.

The climatic parameters of temperature, precipitation and wind are outlined below.

TEMPERATURE

The warmest month is July with an average temperature of 64.4°F and the coldest is February - average temperature 22°F. The monthly average temperatures are shown in Figure 1. Figure 2, the number of days with frost per month, indicates that the onset of significant snow melt will usually occur in April. The months of June through September inclusively are usually frost-free.

PRECIPITATION

There is a marked seasonality of precipitation with a maximum of 6.45 inches in November (29.7 inches snow, .75 inches rain) and a minimum in July of 3.1 inches of rain. The annual total precipitation is 52.8 inches. The maximum recorded one day rainfall is 3.5 inches and

the 15 minute rainfall is 0.4 inches (occurrence once in ten years).
Precipitation data is illustrated in Figure 3.

WIND

Figure 4 displays the frequency of occurrence of wind and their magnitude in graphic form. The prevalent wind is from the southwesterly direction. The maximum observed hourly speed is 72 m.p.h. and the maximum observed gust speed 100 m.p.h.

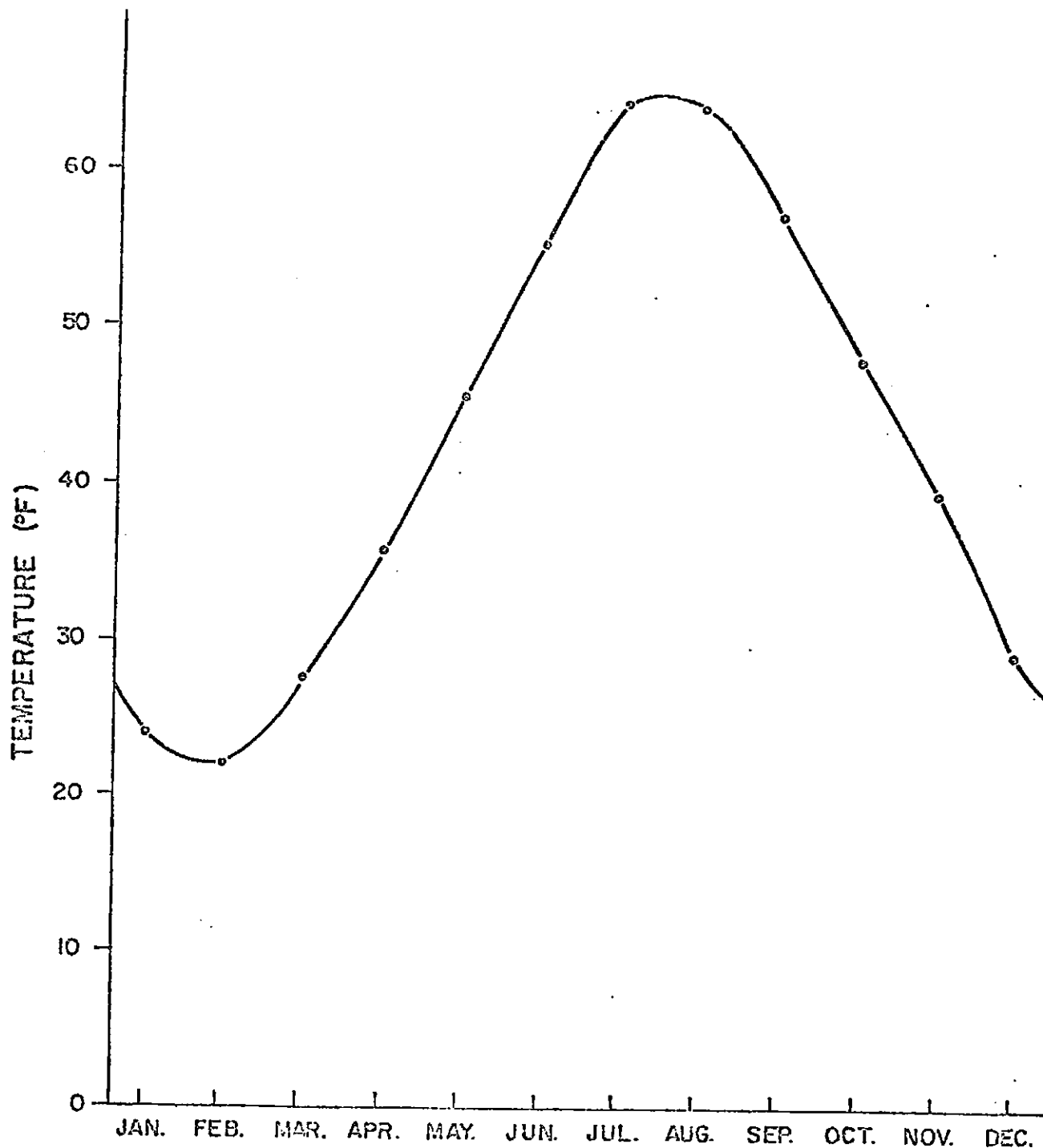


FIG. 1 MONTHLY AVERAGE TEMPERATURE VARIATION

REVISIONS
BY _____ DATE _____

FILE 6584-001

BY L.S. _____ DATE 9-26-74
CHECKED BY _____

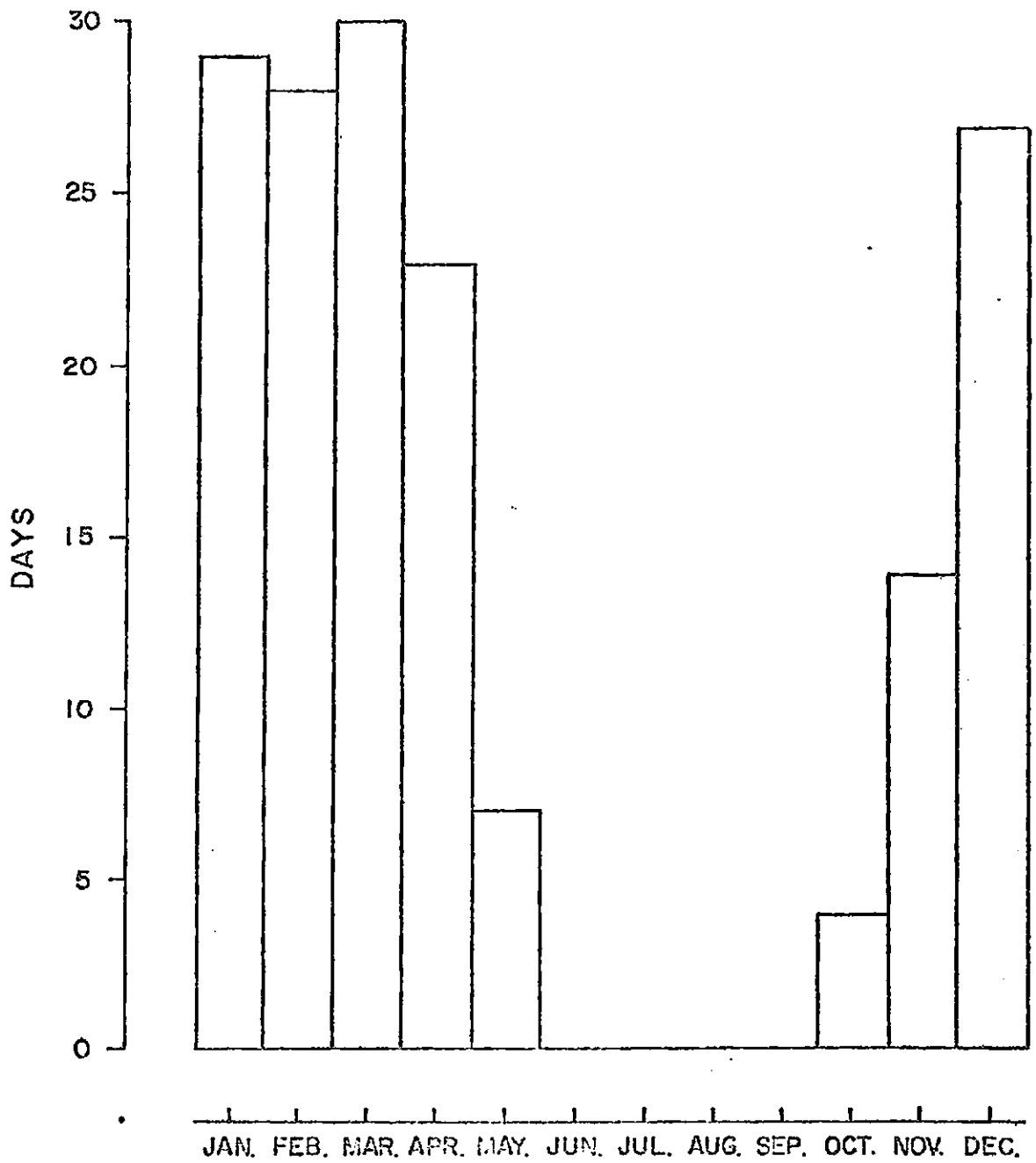


FIG.2 No. OF DAYS WITH FROST PER MONTH
 (DAY ON WHICH DAILY MINIMUM TEMPERATURE
 IS 32° F OR LOWER)

BY L.S. DATE 9-26-74
 CHECKED BY DATE
 FILE 6584-CO
 REVISIONS BY DATE

KEY:

- o RAIN
- * SNOWFALL
- + TOTAL PRECIPITATION

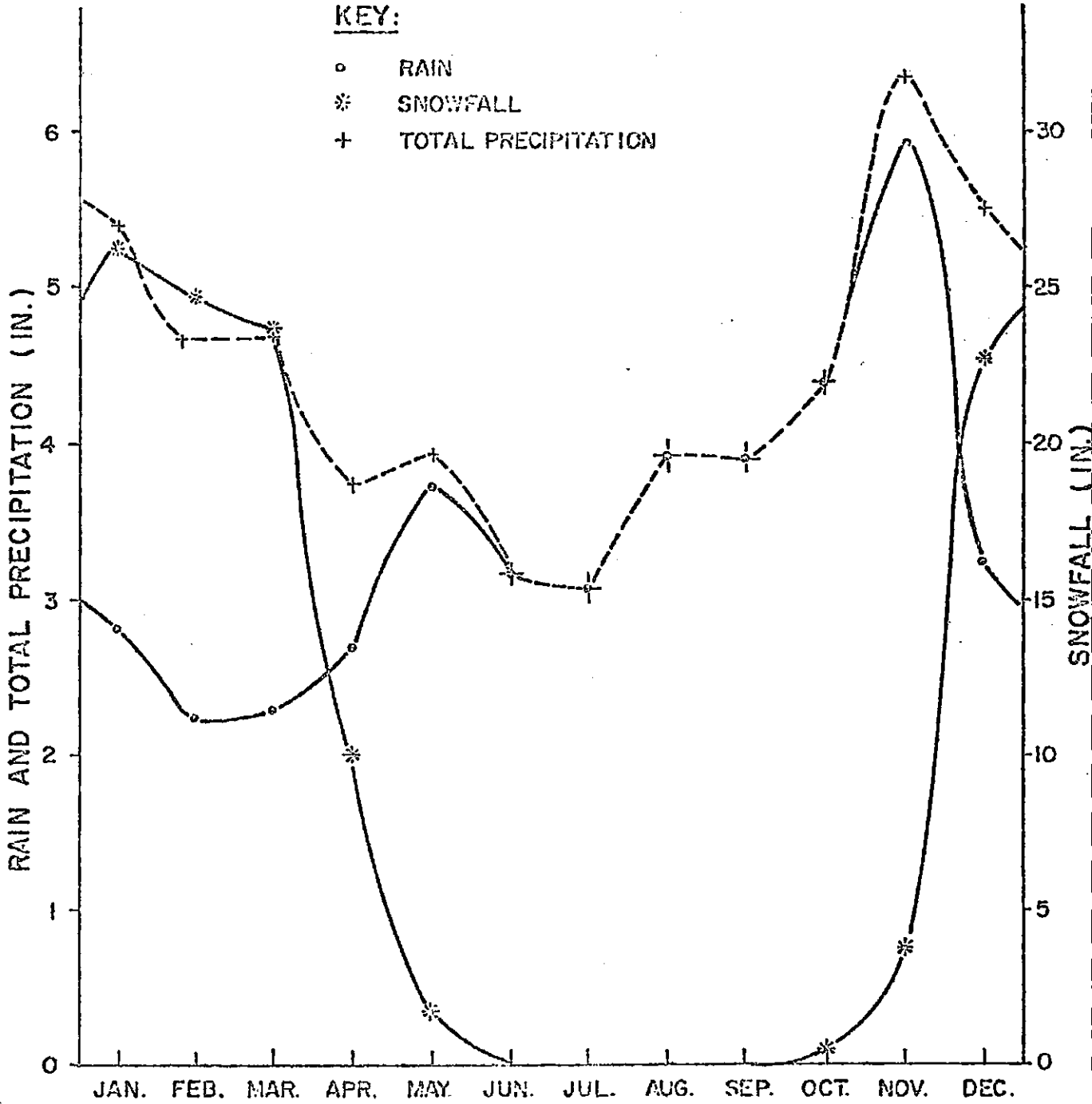


FIG.3 MONTHLY AVERAGE PRECIPITATION VARIATION

BY L.S. DATE 9-26-74
 CHECKED BY [Signature] DATE [Blank]
 FILE 6584-001
 REVISIONS BY [Blank] DATE [Blank]

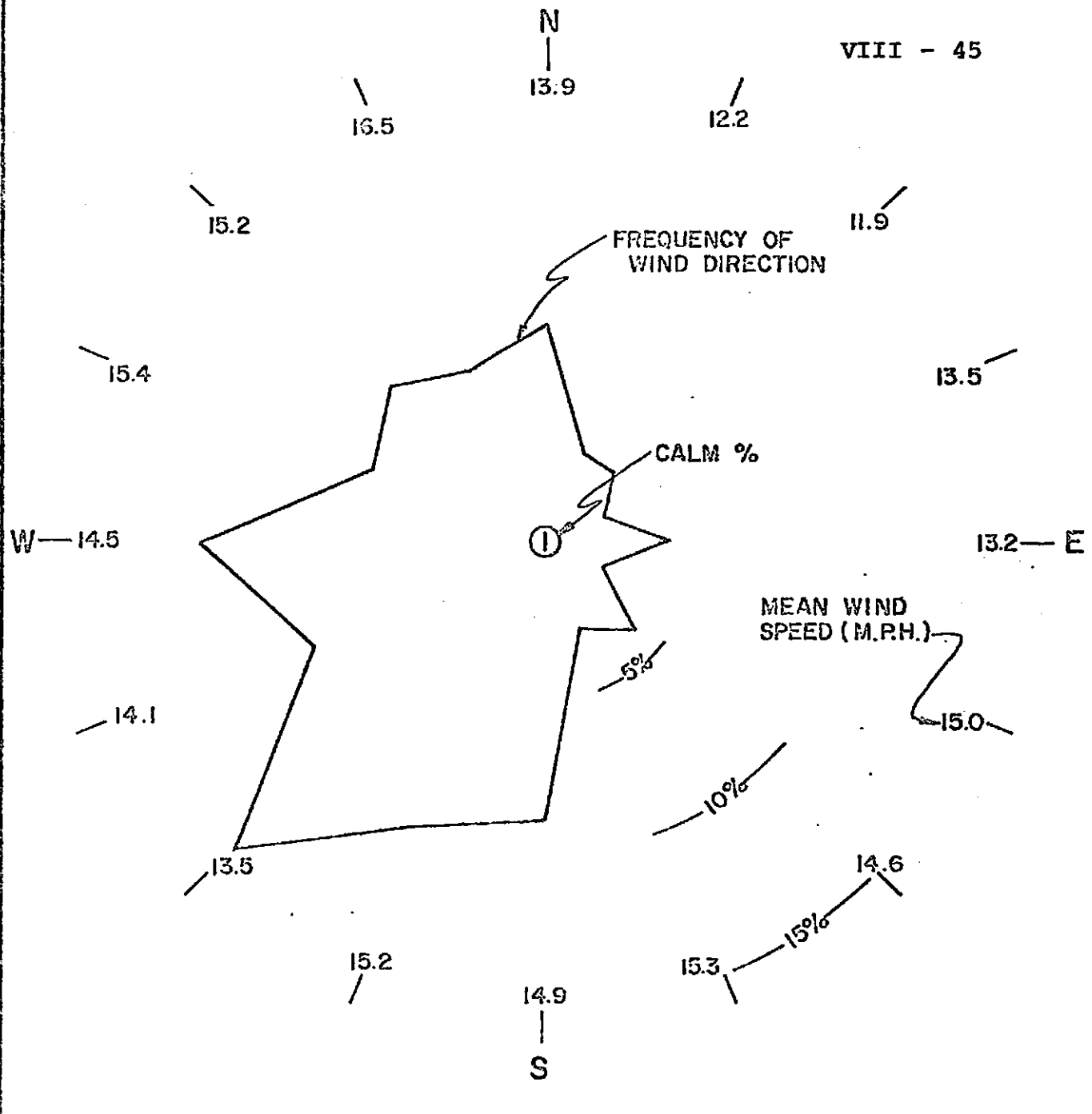


FIG.4 ANNUAL WIND ROSE FOR SYDNEY, NOVA SCOTIA

REVISIONS BY DATE

FILE 6584-001

CHECKED BY DATE

ENVIRONMENTAL:(b) Updated Baseline Statement(i) Water Quality Sampling, Salmon River, 1977

Water and stream sediment samples were collected during August-September, 1977 to provide recent data in addition to that of the Dames & Moore Study (1974). Water samples were collected at six locations, all in proximity to the mine site. Stream sediments were collected at 4 of the 6 locations. The sample sites are identified on the map included.

Site 1. This site corresponds with Dames & Moore (1974) Site 1 on the Salmon River.

Site 2. Site 2 is located on the Salmon River where two brooks which pass through the mine site empty into the river.

Site 3. Drill hole(s) 294 (295). An artesian flow comes from hole(s) 294(5). As the ore zone and upper fifteen feet of bedrock were cemented in these holes it is believed that the flow originates at the base of the over burden.

Site 4. Old dug water well on the John Pronk farm. This sample is essentially a sample from the water table.

(b) Updated Baseline Statement (Cont'd)

Site 5. This site corresponds with Dames & Moore (1974) Site 2 on the Salmon River.

Site 6. This site corresponds with Dames & Moore (1974) Site 9.

Methods of CollectionWater Samples

1. Samples were collected in polythene bottles (0.5 litres).
2. Two samples were taken from each site on each date of collection. One set of samples to be analysed for metals was preserved with 50 ml. of nitric acid. Stream sediment samples were taken with a shovel and placed in a polythene bag.

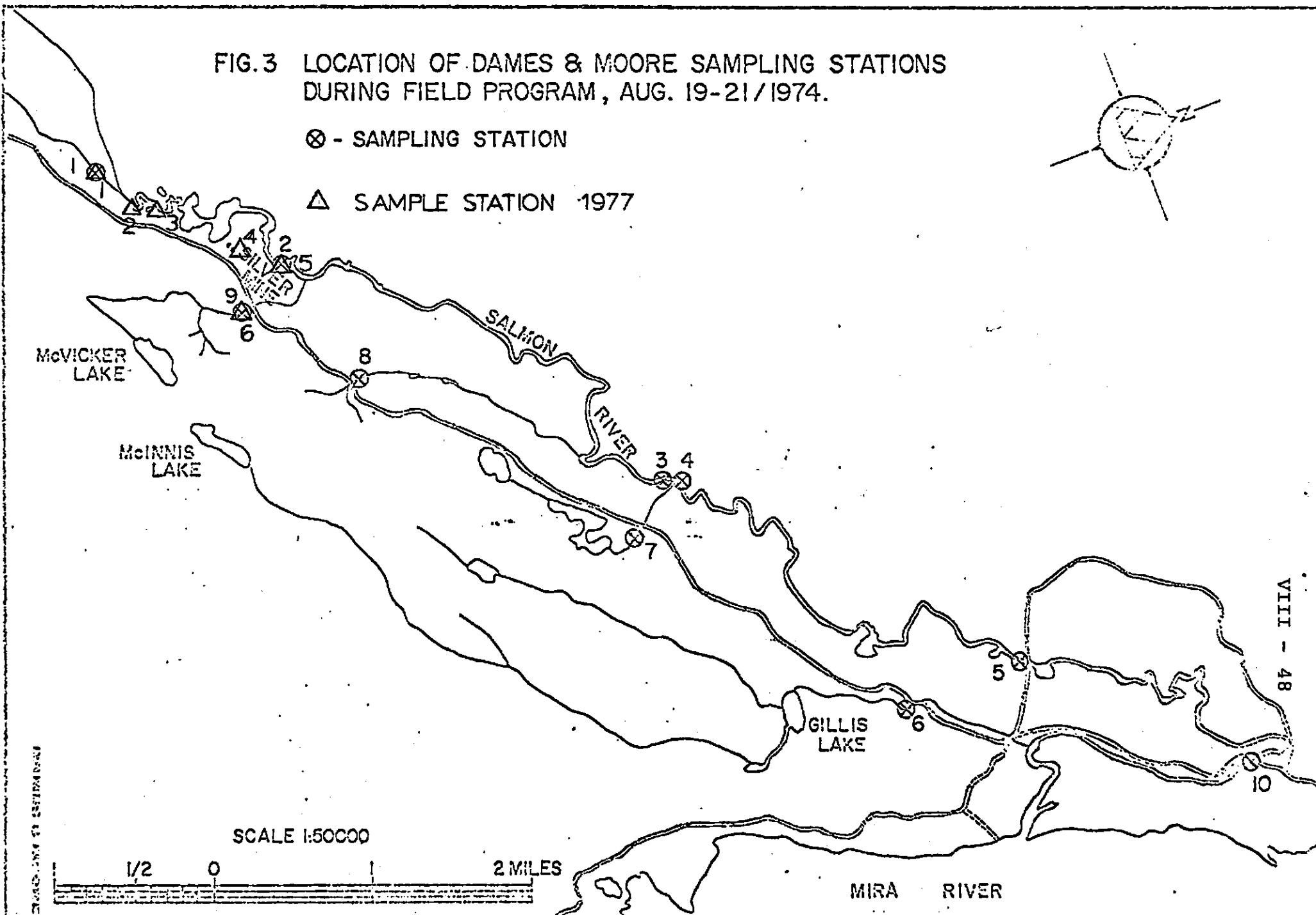
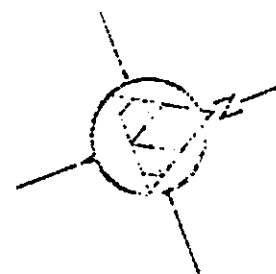
All samples were sent by Acadian Bus Lines to the Atlantic Industrial Research Institute, Nova Scotia Technical College, Halifax, for analysis.

The results are included.

FIG. 3 LOCATION OF DAMES & MOORE SAMPLING STATIONS DURING FIELD PROGRAM, AUG. 19-21/1974.

⊗ - SAMPLING STATION

△ SAMPLE STATION 1977



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PLATE 13

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 Atlantic Industrial Research Institute
 Nova Scotia Technical College
 Halifax, Nova Scotia. B3J 2X4 VIII - 49

Telephone: 429-8300 Ext. 243

Laboratory No. 77-7
 Sample No: Site #1
 Area: Salmon River
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled: August 2, 1977
 Date of Analysis: August 18, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	<u>ppm</u>	<u>epm</u>		
Calcium	2.4		Alkalinities:	
Magnesium	2.88		- Phenolphthalein as CaCO ₃	NIL
Sodium	3.3		- Methyl Orange as CaCO ₃	6.45
Potassium	.12		Hardness (Total as CaCO ₃)	18.63
Iron Total	.26		Loss of Ignition (1 hr. @ 500°C)	
Manganese Total	.01		Total Dissolved Solids (ppm)	46.5
Sulphate	10.41		Suspended Matter (ppm)	<1
Chloride	4.		Specific Conductance (micromhos/cm @ 25°C)	48
Fluoride			pH value	7.5
Nitrate	3.1		Colour	65
Silica			Turbidity	6.5
Boron				
	<u>ppb</u>			
Copper	12			
Cadmium	<1			
Lead	14			
Nickel	<2			
Zinc	8			
Arsenic	4			
Other (specify)				
Cobalt	<2			

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by: *E.P. Cole*

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Laboratory for the Investigation of Pollution
 Atlantic Industrial Research Institute
 Nova Scotia Technical College
 Halifax, Nova Scotia, B3J 2X4
 VIII - 50
 Telephone: 429-8300 Ext 243

Laboratory No. 77-8
 Sample No: Site #2
 Area: Salmon River
 Grid Location: _____
 Sources: _____
 Remarks: _____

Date Sampled: August 2, 1977
 Date of Analysis: August 18, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	<u>ppm</u>	<u>epm</u>		
Calcium	2.4		Alkalinities:	
Magnesium	2.88		- Phenolphthalein as CaCO ₃	NIL
Sodium	3.0		- Methyl Orange as CaCO ₃	8.6
Potassium	.1		Hardness (Total as CaCO ₃)	18.63
Iron Total	.1		Loss of Ignition (1 hr. @ 500°C)	
Manganese Total	.05		Total Dissolved Solids (ppm)	33.7
Sulphate	11.03		Suspended Matter (ppm)	<1
Chloride	2.2		Specific Conductance (micromhos/cm @ 25°C)	43.
Fluoride			pH value	7.
Nitrate	4.43		Calcium	20.
Silica			Turbidity	3.2
Boron				
	<u>ppb</u>			
Copper	6			
Cadmium	<1			
Lead	<10			
Nickel	<2			
Zinc	18			
Arsenic	2			
Other (specify)				
Cobalt	<2			

NOTE: — denotes trace amount (less than 0.01 ppm)

Analysed by: *P. D. Cole*

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Laboratory for the Investigation of Minerals
 Atlantic Industrial Research Institute
 Nova Scotia Technical College
 Halifax, Nova Scotia, B3J 2X4 VIII - 51

Telephone: 429-8300 Ext. 243

Laboratory No. 77-9
 Sample No: Site #3
 Area: Salmon River
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled: August 2, 1977
 Date of Analysis: August 18, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	ppm	ppm		
Calcium	40.8		Alkalinities:	
Magnesium	8.64		- Phenolphthalein as CaCO ₃	NIL
Sodium	9.8		- Methyl Orange as CaCO ₃	73.1
Potassium	.34		Hardness (Total as CaCO ₃)	176
Iron Total	.19		Loss of Ignition (1 hr @ 500°C)	
Manganese Total	.5		Total Dissolved Solids (ppm)	211
Sulphate	45.02		Suspended Matter (ppm)	3.2
Chloride	27.		Specific Conductance (micromhos/cm @ 25°C)	290
Fluoride			pH value	6.4
Nitrate	4.21		Colour	5
Silica			Turbidity	2.4
Boron				
	ppb			
Copper	4			
Cadmium	<1			
Lead	17			
Nickel	2			
Zinc	16			
Arsenic	8			
Other (specify)				
Cobalt	<2			

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by: *L. D. Cook*

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Laboratory for the Investigation of Contaminants
 Atlantic Industrial Research Institute
 Nova Scotia Technical College
 Halifax, Nova Scotia B3J 2X4
 VIII - 52
 Telephone: 429-8300 Ext. 245

Laboratory No. 77-10
 Sample No: Site # 4
 Area: Salmon River
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled: August 2, 1977
 Date of Analysis: August 18, 1977
 Sampled by: _____
 Analyte: _____
 pH: _____ Fe: _____

	ppm	cpm		
Calcium	2.4		Alkalinities:	
Magnesium	1.44		- Phenolphthalein as CaCO ₃	NIL
Sodium	3.1		- Methyl Orange as CaCO ₃	6.45
Potassium	.28		Hardness (Total as CaCO ₃)	12.42
Iron Total	.07		Loss of Ignition: -	
Manganese Total	.067		(1 hr @ 500°C)	
Sulphate	9.75		Total Dissolved Solids (ppm)	23.7
Chloride	5		Suspended Matter (ppm)	1.4
Fluoride			Specific Conductance	37
Nitrate	2.66		(micromhos/cm @ 25°C)	
Silica			pH value	6.6
Boron			...	1.0
			...	1.6
	ppb			
Copper	12			
Cadmium	<1			
Lead	<10			
Nickel	<2			
Zinc	→ 150			
Arsenic	1			
Other (specify)				
Cobalt	<2			

NOTE: T denotes trace amount (less than 0.1) ppm

Analysed by: *L. D. Gault*

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Atlantic Industrial Research Institute
 Nova Scotia Technical College
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Telephone: 429-8300 Ext. 243

Laboratory No. 77-11
 Sample No: Site #5
 Area: Salmon River
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled: August 2, 1977
 Date of Analysis: August 18, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	<u>ppm</u>	<u>epm</u>		
Calcium	7.2		Alkalinities:	
Magnesium	1.44		- Phenolphthalein as CaCO ₃	NIL
Sodium	3.7		- Methyl Orange as CaCO ₃	6.45
Potassium	.12		Hardness (Total as CaCO ₃)	22.77
Iron Total	.38		Loss of Ignition (Thru 500°C)	
Manganese Total	.24		Total Dissolved Solids (ppm)	56.3
Sulphate	15.27		Suspended Matter (ppm)	<1
Chloride	4.4		Specific Conductance (micromhos/cm @ 25°C)	53
Fluoride			pH value	6.4
Nitrate	2.88		Colour	80
Silica			Turbidity	1.6
Boron				
	<u>ppb</u>			
Copper	32			
Cadmium	<1			
Lead	<10			
Nickel	<2			
Zinc	20			
Arsenic	6			
Other (specify)				
Cobalt	<2			

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by: *P. V. Leach*

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Telephone: 429-8300 Ext. 243

Laboratory No. 77-12
 Sample No: Site # 6
 Area: Salmon River
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled: August 2, 1977
 Date of Analysis: August 18, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	<u>ppm</u>	<u>ppm</u>		
Calcium	1.2		Alkalinities:	
Magnesium	2.16		- Phenolphthalein as CaCO ₃	NIL
Sodium	2.5		- Methyl Orange as CaCO ₃	6.45
Potassium	.1		Hardness (Total as CaCO ₃)	12.42
Iron Total	.2		Loss of Ignition (1 hr @ 500°C)	
Manganese Total	.009		Total Dissolved Solids (ppm)	32.6
Sulphate	8.4		Suspended Matter (ppm)	<1
Chloride	1.2		Specific Conductance (micromhos/cm @ 25°C)	31
Fluoride			pH value	6.4
Nitrate	2.22		Colour	30
Silica			Turbidity	2.4
Boron				
	<u>ppt</u>			
Copper	10			
Cadmium	<1			
Lead	<10			
Nickel	2			
Zinc	16			
Arsenic	1			
Other (specify)				
Cobalt	2			

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by: h. P. bal

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Laboratory for the investigation of minerals
 Atlantic Industrial Research Institute
 Nova Scotia Technical College
 Halifax, Nova Scotia. B3J 2X4 VIII - 55

Telephone: 427-8300 Ext. 243

Laboratory No. 77-14
 Sample No: Site # 1 S 2
 Area: Salmon River
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled: _____
 Date of Analysis: August 22, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	<u>ppm</u>	<u>ppm</u>		
Calcium			Alkalinities:	
Magnesium			- Phenolphthalein as CaCO ₃	
Sodium			- Methyl Orange as CaCO ₃	
Potassium			Hardness (Total as CaCO ₃)	
Iron Total	.29		Loss of Ignition (1 h. 500°C)	
Manganese Total	.14		Total Dissolved Solids (ppm)	
Sulphate			Suspended Matter (ppm)	3.1
Chloride			Specific Conductance (micromhos/cm @ 25°C)	115
Fluoride			pH value	6.4
Nitrate			Colour	
Silica			Turbidity	
Boron				
	<u>ppb</u>			
Copper	7			
Cadmium	<1			
Lead	10			
Nickel	4			
Zinc	13			
Arsenic	4			
Other (specify)				
Cobalt	<2			

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by *P. D. Cole*

2 St. Clair Avenue West, Suite 1500
 Toronto, Ontario
 M4V 1L5

Atlantic Industrial Research Institute
 Nova Scotia Technical College
 Halifax, Nova Scotia. B3J 2X4

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Telephone: 429-8300 Ext. 243

Laboratory No. 77-15
 Sample No: Site #2 S 2
 Area: Salmon River
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled: _____
 Date of Analysis: August 22, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	<u>ppm</u>	<u>epm</u>		
Calcium			Alkalinities:	
Magnesium			- Phenolphthalein as CaCO ₃	
Sodium			- Methyl Orange as CaCO ₃	
Potassium			Hardness (Total as CaCO ₃)	
Iron Total	.17		Loss of Ignition (1 hr. @ 500°C)	
Manganese Total	.3		Total Dissolved Solids	
Sulphate			Suspended Matter (ppm)	<1
Chloride			Specific Conductance (micromhos/cm @ 25°C)	71
Fluoride			pH value	6.0
Nitrate			Colour	
Silica			Turbidity	
Boron				
	<u>ppb</u>			
Copper	5			
Cadmium	<1			
Lead	10			
Nickel	2			
Zinc	20			
Arsenic	2			
Other (specify)				
Cobalt	<2			

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by: *h. D. Pool*

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Telephone: 429-8300 Ext. 243

Laboratory No. 77-16
 Sample No: Site #3 S 2
 Area: Salmon River
 Grid Location:
 Source:
 Remarks:

Date Sampled:
 Date of Analysis: August 22, 1977
 Sampled by:
 Aquifer:
 pH: Fe:

	ppm	ppm	
Calcium			Alkalinities:
Magnesium			- Phenolphthalein as CaCO ₃
Sodium			- Methyl Orange as CaCO ₃
Potassium			Hardness (Total as CaCO ₃)
Iron Total	1.63		Loss of Ignition (1 hr. 500°C)
Manganese Total	.55		Total Dissolved Solid
Sulphate			Suspended Matter (ppm) 18.8
Chloride			Specific Conductance (micromhos/cm @ 25°C) 265
Fluoride			pH value 6.3
Nitrate			Colour
Silica			Turbidity
Barium			
	ppm		
Copper	10		
Cadmium	<1		
Lead	15		
Nickel	8		
Zinc	30		
Arsenic	6		
Other (specify)			
Cobalt	<2		

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by:

B. D. Kelly

Barium Exploration Limited
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 Toronto, Ontario
 M4V 1L5

Laboratory for the Investigation of Minerals
 Atlantic Industrial Research Institute
 Nova Scotia Technical College
 Halifax, Nova Scotia. B3J 2X4

VIII - 58

Telephone: 429-8300 Ext. 243

Laboratory No. 77-17
 Sample No: Site # 4 S 2
 Area: Salmon River
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled: _____
 Date of Analysis: August 22, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	ppm	ppm		
Calcium			Alkalinities:	
Magnesium			- Phenolphthalein as CaCO ₃	
Sodium			- Methyl Orange as CaCO ₃	
Potassium			Hardness (Total as CaCO ₃)	
Iron Total	.09		Loss of Ignition (1 hr. 500 C)	
Manganese Total	.18		Total Dissolved Solids	
Sulphate			Suspended Matter (ppm)	<1
Chloride			Specific Conductance (micromhos/cm @ 25°C)	37
Fluoride			pH value	5.8
Nitrate			Colour	
Silica			Turbidity	
Boron				
	ppm			
Copper	10			
Cadmium	<1			
Lead	→ 25			
Nickel	10			
Zinc	→ 88			
Arsenic	<1			
Other (specify)				
Cobalt	<2			

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by:

H. D. Cook

2 St. Clair Avenue West, Suite 1500
 Toronto, Ontario
 M4V 1L5

Laboratory for the Investigation of Minerals
 Atlantic Industrial Research Institute
 Nova Scotia Technical College
 Halifax, Nova Scotia. B3J 2X4
 VIII - 59
 Telephone: 429-8300 Ext. 243

Laboratory No. 77-18
 Sample No: Site #5 S 2
 Area: Salmon River
 Grid Location:
 Source:
 Remarks:

Date Sampled:
 Date of Analysis: August 22, 1977
 Sampled by:
 Aquifer:
 pH: Fe:

	ppm	epm		
Calcium			Alkalinities:	
Magnesium			- Phenolphthalein as CaCO ₃	
Sodium			- Methyl Orange as CaCO ₃	
Potassium			Hardness (Total as CaCO ₃)	
Iron Total	.36		Loss of Ignition (1 hr. @ 500°C)	
Manganese Total	.067		Total Dissolved Solids	
Sulphate			Suspended Matter (ppm)	<1
Chloride			Specific Conductance (micromhos/cm 25°C)	56
Fluoride			pH value	5.7
Nitrate			Colour	
Silica			Turbidity	
Boron				
	ppm			
Copper	12			
Cadmium	<1			
Lead	10			
Nickel	11			
Zinc	32			
Arsenic	1			
Other (specify)				
Cobalt	<2			

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by: *h. J. Cole*

Barium Exploration Limited
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 Halifax, Nova Scotia B3J 2X4
 VIII - 60

Telephone: 429-8300 Ext. 243

Laboratory No. 77-19
 Sample No. Site # 6 S 2
 Area: Salmon River
 Grid Location:
 Source:
 Remarks:

Date Sampled:
 Date of Analysis: August 22, 1977
 Analyzed by:
 Aquifer:
 pH: Fe:

	ppm	ppm	
Calcium			Alkalinity:
Magnesium			- Phenolphthalein as CaCO ₃
Sodium			- Methyl Orange as CaCO ₃
Potassium			Hardness (total as CaCO ₃)
Iron Total	.09		Loss at Ignition
Manganese Total	.005		(1 hr. @ 500 °C)
Sulphate			Total Dissolved Solids
Chloride			Suspended Matter (ppm) <1
Fluoride			Specific Conductance
Nitrate			(microhm/cm @ 25 °C) 90
Silica			pH value (5.8)
Boron			Color
			Turbidity
	ppb		
Copper	3		
Cadmium	<1		
Lead	<10		
Nickel	7		
Zinc	→ 37		
Arsenic	<1		
Other (specify)			
Cobalt	<2		

NOTE: T denotes trace amount (less than 0.01 ppm)

Analyzed by: *L. D. Cole*

2 St. Clair Avenue West, Suite 1500
 TORONTO, Ontario
 M4V 1L5

Laboratory for the investigation of minerals
 Atlantic Industrial Research Institute
 Nova Scotia Technical College
 Halifax, Nova Scotia. B3J 2X4 VIII - 61

Telephone: 429-8300 Ext. 243

Laboratory No. 77-20
 Sample No: Site 1 S3
 Area: Salmon River
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled: _____
 Date of Analysis: Sept. 1, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	<u>ppm</u>	<u>epm</u>		
Calcium			Alkalinities:	
Magnesium			- Phenolphthalein as CaCO ₃	
Sodium			- Methyl Orange as CaCO ₃	
Potassium			Hardness (Total as CaCO ₃)	
Iron Total	.31		Loss of Ignition (1 hr. @ 500 C)	
Manganese Total	.074		Total Dissolved Solids	
Sulphate			Suspended Matter (ppm)	14.1
Chloride			Specific Conductance (micromhos/cm @ 25°C)	85
Fluoride			pH value	6.0
Nitrate			Colour	
Silica			Turbidity	
Boron				
	<u>ppb</u>			
Copper	12			
Cadmium	<1			
Lead	<10			
Nickel	2			
Zinc	5			
Arsenic	4			
Other (specify)				
Cobalt	<1			

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by: *P. D. Kelly*

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 Atlantic Industrial Research Institute
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Telephone: 429-8300 Ext. 243 VIII - 62

Laboratory No. 77-21
 Sample No: Site 2 S3
 Area: Salmon River
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled: _____
 Date of Analysis: Sept. 1, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	ppm	ppm		
Calcium			Alkalinities:	
Magnesium			- Phenolphthalein as CaCO ₃	
Sodium			- Methyl Orange as CaCO ₃	
Potassium			Hardness (Total as CaCO ₃)	
Iron Total	.16		Loss of Ignition	
Manganese Total	.35		(1 hr. @ 100 C)	
Sulphate			Total Dissolved Solids	
Chloride			Suspended Matter (ppm)	10.8
Fluoride			Specific Conductance	
Nitrate			(micromhos/cm @ 25 C)	68
Silica			pH value	5.7
Boron			Colour	
			Turbidity	
	ppm			
Copper	5			
Cadmium	<1			
Lead	<10			
Nickel	<1			
Zinc	10			
Arsenic	6			
Other (specify)				
Cobalt	<1			

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by:

L. D. Hardy

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TORONTO, Ontario

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Atlantic Industrial Research Institute
Nova Scotia Technical College
Halifax, Nova Scotia. B3J 2X4

VIII - 63

Telephone: 429-8300 Ext 243

Laboratory No. 77-22
Sample No: Site 3 S3
Area: Salmon River
Grid Location: _____
Source: _____
Remarks: _____

Date Sampled: _____
Date of Analysis: Sept. 1, 1977
Sampled by: _____
Aquifer: _____
pH: _____ Fe: _____

	ppm	ppm		
Calcium			Alkalinity:	
Magnesium			- Phenolphthalein as CaCO ₃	
Sodium			- Methyl Orange as CaCO ₃	
Potassium			Hardness (Total as CaCO ₃)	
Iron Total	1.46		Loss of Ignition	
Manganese Total	.47		(1 hr @ 100 C)	
Sulfate			Total Dissolved Solid	
Chloride			Suspended Matter (ppm)	11.0
Fluoride			Specific Conductance	290
Nitrate			(microhm/cm @ 25 C)	
Silica			pH value	5.6
Boron			Viscosity	
			Toxicity	
Copper	10			
Cadmium	<1			
Lead	15			
Zinc	4			
Mercury	→ 43			
Arsenic	8			
Other (specify)				
Cyanide	<1			

Trace elements listed are in trace amount (less than 0.01 ppm)

Analysed by: *B. D. Cook*

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laboratory for the investigation of water
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 Nova Scotia Institute of Technology
 Halifax, Nova Scotia B3J 2X4 VIII - 64

Telephone: 429-8300 Ext. 243

Laboratory No. 77-23
 Sample No. Site 4 S3
 Area: Salmon River
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled _____
 Date of Analysis: Sept. 1, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	ppm	CFM		
Calcium			Alkalinity	
Magnesium			- Total Hardness as CaCO ₃	
Sodium			- Alkalinity range as CaCO ₃	
Potassium			Hardness (Total as CaCO ₃)	
Iron Total	.2		Calcium Hardness	
Manganese Total	.083		Magnesium Hardness	
Sulfate			Total Dissolved Solid	
Chloride			Suspended Matter (ppm)	9.1
Fluoride			Specific Conductance	
Nitrate			(microhm/cm @ 25°C)	41
Silica			pH value	5.4
Boron			Color	
			Turbidity	
Copper	15			
Lead	<1			
Zinc	30			
Cadmium	3			
Mercury	51			
Arsenic	8			
Other metals				
Cyanide	<1			

Note: All values are in mg/l unless otherwise specified.

Analyzed by: *R. D. L.*

2 St. Clair Avenue West, Suite 1500
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Laboratory for the investigation of Minerals
 Atlantic Industrial Research Institute
 Nova Scotia Technical College
 Halifax, Nova Scotia. B3J 2X4
 VIII - 65
 Telephone: 429-8300 Ext 243

Laboratory No. 77-24
 Sample No: Site 5 S3
 Area: Salmon River
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled: _____
 Date of Analysis: Sept. 1, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	<u>ppm</u>	<u>ppm</u>		
Calcium			Alkalinity:	
Magnesium			- Phenolphthalein as CaCO ₃	
Sodium			- Methyl Orange as CaCO ₃	
Potassium			Hardness (Total as CaCO ₃)	
Iron Total	.4		Loss of Ignition	
Manganese Total	.076		(1 hr. 500 C)	
Sulphate			Total Dissolved Solids	
Chloride			Suspended Matter (ppm)	28.3
Fluoride			Specific Conductance	
Nitrate			(micromhos/cm : 25 °C)	62
Silica			pH value	5.25
Boron			Colour	
			Turbidity	
	<u>ppb</u>			
Copper	17			
Cadmium	<1			
Lead	10			
Nickel	<1			
Zinc	10			
Arsenic	1			
Other (specif.)				
Cobalt	<1			

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by: *B. D. Cook*

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 ... TORONTO, Ontario
 ... M4V 1L5

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 Nova Scotia Technical College
 Halifax, Nova Scotia. B3J 2X4

Telephone: 429-8300 Ext. 243

Laboratory No. 77-25
 Sample No: Site 6 S3
 Area: Salmon River
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled: _____
 Date of Analysis: Sept. 1, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	<u>ppm</u>	<u>epm</u>		
Calcium			Alkalinities:	
Magnesium			- Phenolphthalein as CaCO ₃	
Sodium			- Methyl Orange as CaCO ₃	
Potassium			Hardness (Total as CaCO ₃)	
Iron Total	.64		Loss of Ignition (1 hr. @ 500 C)	
Manganese Total	.026		Total Dissolved Solids	
Sulphate			Suspended Matter (ppm)	14.3
Chloride			Specific Conductance (micromhos/cm @ 25 C)	34
Fluoride			pH value	5.3
Nitrate			Color	
Silica			Turbidity	
Boron				
	<u>ppb</u>			
Copper	20			
Cadmium	<1			
Lead	<10			
Nickel	<1			
Zinc	10			
Arsenic	2			
Other (specify)				
Cobalt	<1			

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by: B. D. Cole

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Laboratory for the Investigation of
 Atlantic Industrial Research Institute
 Nova Scotia Technical College
 Halifax, Nova Scotia. B3J 2X4

VIII - 67

Telephone: 429-8300 Ext. 243

Laboratory No. 77-27
 Sample No: Site 1 #54
 Area: _____
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled: _____
 Date of Analysis: Sept. 19, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	<u>ppm</u>	<u>epm</u>		
Calcium			Alkalinities:	
Magnesium			- Phenolphthalein as CaCO ₃	
Sodium			- Methyl Orange as CaCO ₃	
Potassium			Hardness (Total as CaCO ₃)	
Iron Total	.19		Loss of Ignition (1 hr. @ 500°C)	
Manganese Total	.018		Total Dissolved Solids	
Sulphate			Suspended Matter (ppm)	1.2
Chloride			Specific Conductance (micromhos/cm @ 25°C)	58
Fluoride			pH value	6.0
Nitrate			Colour	
Silica			Turbidity	
Boron				
	<u>ppb</u>			
Copper	6			
Cadmium	<1			
Lead	10			
Nickel	3			
Zinc	30			
Arsenic	8			
Other (specify)				
Cobalt	<1			

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by: *P. D. ...*

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TORONTO, Ontario

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Atlantic Industrial Research Institute
 Nova Scotia Technical College
 Halifax, Nova Scotia. B3J 2X4

VIII - 68

Telephone: 429-8300 Ext. 243

Laboratory No. 77-28
 Sample No: Site 2 #S4
 Area: _____
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled: _____
 Date of Analysis: Sept. 19, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	<u>ppm</u>	<u>epm</u>		
Calcium			Alkalinities:	
Magnesium			- Phenolphthalein as CaCO ₃	
Sodium			- Methyl Orange as CaCO ₃	
Potassium			Hardness (Total as CaCO ₃)	
Iron Total	.27		Loss of Ignition (1 hr. @ 500°C)	
Manganese Total	.73		Total Dissolved Solids	
Sulphate			Suspended Matter (ppm)	3.7
Chloride			Specific Conductance (micromhos/cm @ 25°C)	46
Fluoride			pH value	5.4
Nitrate			Colour	
Silica			Turbidity	
Boron				
	<u>ppb</u>			
Copper	8			
Cadmium	<1			
Lead	15			
Nickel	1			
Zinc	≤10			
Arsenic	2			
Other (specify)				
Cobalt	<1			

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by: *E. D. Cole*

Barium Exploration Limited
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 Halifax, Nova Scotia. B3J 2X4

VIII - 69

Telephone: 429-8300 Ext. 243

Laboratory No. 77-29
 Sample No: Site 3 #54
 Area: _____
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled: _____
 Date of Analysis: Sept. 19, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	<u>ppm</u>	<u>epm</u>	
Calcium			Alkalinities:
Magnesium			- Phenolphthalein as CaCO ₃
Sodium			- Methyl Orange as CaCO ₃
Potassium			Hardness (Total as CaCO ₃)
Iron Total	1.5		Loss of Ignition (1 hr. at 500°C)
Manganese Total	.45		Total Dissolved Solids
Sulphate			Suspended Matter (ppm)
Chloride			Specific Conductance (micromhos/cm @ 25°C)
Fluoride			pH value
Nitrate			Colour
Silica			Turbidity
Boron			
	<u>ppb</u>		
Copper	4		
Cadmium	<1		
Lead	<10		
Nickel	1		
Zinc	20		
Arsenic	2		
Other (specify)			
Cobalt	<1		

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by: *B. D. Cook*

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 TORONTO, Ontario
 M4V 1L5

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 Atlantic Industrial Research Institute
 Nova Scotia Technical College
 Halifax, Nova Scotia. B3J 2X4

VIII - 70

Telephone: 429-8300 Ext. 243

Laboratory No. 77-30
 Sample No: Site 4 #S4
 Area: _____
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled: _____
 Date of Analysis: Sept. 19, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	<u>ppm</u>	<u>epm</u>		
Calcium			Alkalinities:	
Magnesium			- Phenolphthalein as CaCO ₃	
Sodium			- Methyl Orange as CaCO ₃	
Potassium			Hardness (Total as CaCO ₃)	
Iron Total	.08		Loss of Ignition (1 hr. @ 500°C)	
Manganese Total	.064		Total Dissolved Solids	
Sulphate			Suspended Matter (ppm)	1.5
Chloride			Specific Conductance (micromhos/cm @ 25°C)	36
Fluoride			pH value	5
Nitrate			Colour	
Silica			Turbidity	
Boron				
	<u>ppb</u>			
Copper	11			
Cadmium	<1			
Lead	<10			
Nickel	1			
Zinc	30			
Arsenic	6			
Other (specify)	<1			
Cobalt				

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by: *E. D. hole*

2 St. Clair W., Suite 1500

TORONTO, Ontario

M4V 1L5

Atlantic Industrial Research Institute
Nova Scotia Technical College
Halifax, Nova Scotia. B3J 2X4

VIII - 71

Telephone: 429-8300 Ext. 243

Laboratory No. 77-31
Sample No: Site 5 #S4
Area: _____
Grid Location: _____
Source: _____
Remarks: _____

Date Sampled: _____
Date of Analysis: Sept. 19, 1977
Sampled by: _____
Aquifer: _____
pH: _____ Fe: _____

	<u>ppm</u>	<u>epm</u>		
Calcium			Alkalinities:	
Magnesium			- Phenolphthalein as CaCO ₃	
Sodium			- Methyl Orange as CaCO ₃	
Potassium			Hardness (Total as CaCO ₃)	
Iron Total	.5		Loss of Ignition (1 hr. @ 500°C)	
Manganese Total	.1		Total Dissolved Solids	
Sulphate			Suspended Matter (ppm)	4.1
Chloride			Specific Conductance (micromhos/cm @ 25°C)	69.
Fluoride			pH value	5.0
Nitrate			Colour	
Silica			Turbidity	
Boron				
	<u>ppb</u>			
Copper	17			
Cadmium	7			
Lead	<10			
Nickel	1			
Zinc	<10			
Arsenic	1			
Other (specify)				
Cobalt	<1			

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by:

b. D. Cole

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 TORONTO, Ontario
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 Atlantic Industrial Research Institute
 Nova Scotia Technical College
 Halifax, Nova Scotia. B3J 2X4

VIII - 72

Telephone: 429-8300 Ext. 243

Laboratory No. 77-32
 Sample No: Site 6 #S4
 Area: _____
 Grid Location: _____
 Source: _____
 Remarks: _____

Date Sampled: _____
 Date of Analysis: Sept. 19, 1977
 Sampled by: _____
 Aquifer: _____
 pH: _____ Fe: _____

	<u>ppm</u>	<u>epm</u>		
Calcium			Alkalinities:	
Magnesium			- Phenolphthalein as CaCO ₃	
Sodium			- Methyl Orange as CaCO ₃	
Potassium			Hardness (Total as CaCO ₃)	
Iron Total	.14		Loss of Ignition (1 hr. @ 500°C)	
Manganese Total	.014		Total Dissolved Solids	
Sulphate			Suspended Matter (ppm)	1.3
Chloride			Specific Conductance (micromhos/cm @ 25°C)	36
Fluoride			pH value	5.2
Nitrate			Colour	
Silica			Turbidity	
Boron				
	<u>ppb</u>			
Copper	8			
Cadmium	<1			
Lead	<10			
Nickel	<1			
Zinc	<10			
Arsenic	1			
Other (specify)				
Cobalt	<1			

NOTE: T denotes trace amount (less than 0.01 ppm)

Analysed by: *b. D. Cook*

September 16, 1977

Barymin Exploration Limited
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 Toronto, Ontario M4V 1L5

Geochemical Analysis

Samples: Silts

Sample	Element P.P.M.							
	Cu	Cd	Co	Ni	Pb	Zn	Hg	As
Site #1	5	.6	82 ✓	13	68 ✓	150 ✓	<.5	3.6
Site #2	5	.5	73 ✓	10	570 ✓	290 ✓	<.5	3.2
Site #5	2	.6	24	5	14	17	<.5	2.4
Site #6	7	.6	65 ✓	9	610 ✓	290 ✓	<.5	3.6

Laboratory for the Investigation of Minerals.

ENVIRONMENTAL:(b) Updated Baseline Statement (Cont'd)(ii) River Flow Rate Measurement, Salmon River, 1977

The rate of flow of the Salmon River was measured at Site 5 on each of the dates of water sample collection. The method of measurement consisted simply of measuring the average width and average depth of the river over a 100-foot distance to determine the volume at this site and then noting the time interval required for an empty sample bottle to float downstream over the 100 foot distance. From this data the flow rate was calculated in cubic feet/minute.

Flow Rates: Site 5

<u>Date</u>	<u>Rate (cu.ft/min.)</u>
2.08.77	4410
16.08.77	367
30.08.77	1082

Flow Rate - Sample CalculationData - from first series of samples - 2.08.77

Length 100'

Time 40 seconds

Width A-A' 45'

Depths A-A'	<u>Depth</u>	0	9"	8"	10"	12"	12"	11"	12"	8"	0
	<u>Position</u>	45	40	35	30	25	20	15	10	5	0

Width B-B' 37'

Depths B-B'	<u>Depth</u>	0	8"	13"	15"	14"	13"	10"	0
	<u>Position</u>	37	30	25	20	15	10	5	0

Calculations

$$\text{Average Depth A-A'} = \frac{(0 + 9" + 8" + 10" + 12" + 12" + 11" + 12" + 8" + 0)}{10} = \underline{8.2"}$$

$$\text{Average Depth B-B'} = \frac{(0 + 8" + 13" + 15" + 14" + 13" + 10" + 0")}{8} = \underline{9.1"}$$

$$\text{Cross Sectional Area A-A'} = \frac{8.2"}{12"} \times 45' = \underline{30.7} \text{ sq.ft.}$$

$$\text{Cross Sectional Area B-B'} = \frac{9.1"}{12"} \times 37' = \underline{28.1} \text{ sq.ft.}$$

$$\text{Average Area of River Section} = \frac{(30.7 + 28.1)}{2} = \underline{29.4} \text{ sq.ft.}$$

$$\text{Volume of River Section} = 29.4 \text{ sq.ft.} \times 100 \text{ ft.} = \underline{2940} \text{ cu.ft.}$$

Flow Rate of River Section

$$2940 \text{ cu.ft. in } 40 \text{ sec.} = \text{cu.ft.} + 60 \text{ sec.}$$

$$\frac{2940}{40} = \frac{\quad}{60} = \underline{4410 \text{ cu.ft./min.}}$$

ENVIRONMENTAL:(b) Updated Baseline Statement (Cont'd)(iii) Water Table Levels

Drill hole 264 (line 34W; 15 + 00N) was cased and left open. Water table level readings were taken at this hole weekly commencing June 1, 1977.

The elevation of the top of the casing hole 264 was established at 4964.88 feet.

Water Table Level Readings

<u>Date</u>	<u>Time</u>	<u>Elev. (ft.) DDH 264 = 4964.88</u>	<u>Remarks</u>
1.06.77	1.30 p.m.	4954.99	Heavy rain, June 3,4 & 5, 1977
7.06.77	10.30 a.m.	4955.59	
14.06.77	4.15 p.m.	4955.53	
21.06.77	3.30 p.m.	4955.38	
28.06.77	9.30 p.m.	4955.43	
4.07.77	1.15 p.m.	4955.19	
11.07.77	5.30 p.m.	4955.19	
18.07.77	1.30 p. m.	4955.15	
25.07.77	12.50 p.m.	4954.98	
2.08.77	1.30 p.m.	4954.96	
9.08.77	12.35 p.m.	4954.80	
16.08.77	2.00 p.m.	4954.58	
26.08.77	1.40 p.m.	4954.80	
4.09.77	1.45 p.m.	4954.66	
12.09.77	3.50 p.m.	4954.54	
20.09.77	6.30 p.m.	4955.19	

IX

ECONOMICS

- (a) Marketing
- (b) Production Statistics

ECONOMICS:(a) Marketing

Metallurgical test conducted on drill core samples of Silvermines lead ore indicate that the Yava Mines concentrator will be able to produce a lead concentrate containing approximately 75% lead with no impurities on which a lead smelter would levy penalties. Annual production at 500 tons ore per day will be approximately 11,000 metric tonnes concentrates, which, in view of the relatively high lead ^{content} and the absence of objectionable impurities, will be a desirable product to any custom lead smelter or refinery - particularly to any which does not make a silver recovery.

As there are no lead refining facilities in Nova Scotia it will be necessary to ship concentrates outside the Province. The closest port of shipment for destination to smelter is Sydney, from which the concentrates can be transported by rail to destinations on the North American Continent, or by sea from deep water loading facilities. Because, with the possible exception of the Brunswick Mining and Smelting lead refinery at Beldune, New Brunswick, rail transportation costs would be excessive compared with shipping out of Sydney, it is anticipated that Yava lead concentrates will be trucked about 30 miles to portside at Sydney and thence shipped by water.

Good covered storage adjacent to deep water is available at the Government wharf off Esplanade Street, Sydney and it is proposed arrangements will be made with Transport Canada and the Nova Scotia Department of Development to

75% Pb?
calculations on
or 70% Pb conc
Mills price of

(a) Marketing (Cont'd)

secure harbour storage and shipping facilities at this site. Depending on destination, it is expected that part cargo bulk shipments of concentrates will be made 3 or 4 times a year and that covered storage will be obtained for up to five thousand tons at dockside.

As soon as definitive operating cost data is available from the initial underground development program, and production targets and dates are set, tenders for Yava's output of lead concentrates will be invited from a number of smelters. In view of their advantage in freight costs, it is expected lead smelters on the European Sea Board will be the most competitive bidders, and considerable interest has already been shown in Yava's future output by several European Smelters. Interest has also been expressed by Brunswick Mining & Smelting, although Yava has been advised they are not in the market for custom concentrates at this time. This situation, however, is open to change by the end of 1977.

Price of Lead Metal

At the time of writing the Producer Price of refined lead in Canada is 32 cents/lb. and 31 cents/lb. in U.S.A. The London Metal Exchange price varies from day to day and is at this time (September 20, 1977) equivalent to 27 - 28 cents/lb. Lead metal, where the principal consumption lies in storage batteries, is energy and thus seasonally related. Summer price weakness in European markets is gradually strengthening with the advent of winter and the price differential with North American

(a) Marketing (Cont'd)

prices which has been as wide as 7 cents/lb. is closing. There is no indication that North American prices will step down as metal inventories are low and primary lead production in 1977 has seen a general reduction due to strikes at mines and smelters and other cut backs. In the circumstances there appears every likelihood that the European prices will rise over the winter months to equate and avoid a net outflow of metal to the U.S.A.

Overall, in common with zinc and copper prices, lead is considered at depressed price levels by the mining industry, and cost inflation is making it uneconomic for current producers to increase production at current prices to meet any expansion in demand. In due course, as general economic conditions improve, an increase in demand for all base metals will take place. In view of the low inventory position in lead metal it is reasonable to anticipate price increases in the future. The timing of production from the Silvermines deposit directed at the spring of 1978 is judged to coincide with a gradual strengthening in market demand.

ECONOMICS: (Cont'd)(b) Production Statistics

It should be understood at this stage of the development of the Silvermines property that until the initial development is done underground, any mining cost estimates are highly qualified. This is true to a lesser extent of milling cost estimates which are subject to test runs through the Company's mill and the finalization of the mill flowsheet.

In the latest feasibility studies made, mining and milling cost estimates have been calculated on a conservative basis to allow for foreseeable contingencies. Underground costs will be substantially affected by variations in conditions such as hanging and footwall conditions, water, and rock breaking, to mention the principals. In general terms drilling at close spacing has indicated that good conditions exist for trackless room and pillar mining and that high underground productivity should be achieved. * *This was the opinion expressed in the Phelps Dodge 1974 feasibility study also.*

Initial daily production rate has been estimated at 500 short tons of ore per day, equivalent at 340 mill working days to 170,000 tons per annum. Underground openings and equipment provided in these estimates have considerable built-in additional capacity. At this time the apparent capacity for tonnage throughout will be governed by the crushing and grinding section in the mill, where otherwise flotation capacity is adequate for over 1000 tons per day. There will be considerable profit advantage to the operation from increasing mine output

(b) Production Statistics (Cont'd)

where this can be handled through the mill without having to increase mine and mill equipment capacity. As a very rough indicator, using current levels of lead prices, a 10% increase in throughput involving a 3% decrease in unit costs results in a 23% increase in operating profit. A 20% increase in throughput involving a 7% decrease in unit costs results in a 55% increase in operating profit.

Variations in the market price of lead metal has an effect on economic grade to be mined. In the case of the Silvermines deposit, where mining will be carried to assay walls, incremental costs of mining lower grade material at different metal prices will be an important cost control factor. Once again, as a very rough indicator, at current lead prices \pm 3 cents in price is equivalent in operating revenue to \pm 10% variation in grade of ore (using average grade of ore of 5%). In other words, an increase in price of 3 cents/lb. lead over current levels would provide the same level of gross operating profit, when mining at a 10% lower grade. In circumstances where incremental costs of mining the lower grade material improve gross operating profit, it is clearly beneficial to mine at an overall lower grade and increase the mine ore reserves accordingly.

*1c + 3 cents/lb
non-mining cost
from profit
ie 4. cents/lb
grade*

On the basis of present estimates and at current prices of lead metal, the following statistics constitute a conservative measure of the economic benefits of the operation:-

Gross Net Smelter Revenues to the Mine
Gross Wages
Mine and Mill Supplies
Truck Haulage Contracted *ore price .96 x 170,000 = 163,200*

Annual \$
3,200,000
1,750,000 } 2,590,000
700,000 } previously
140,000 }
2,288,000

*Operating profit = say \$600,000
Deduct, reclamation costs,
insurance, taxes (min) depreciation, exploration
Royalties (15%) = \$60,000
Divs - \$60,000
To service debt*

70% 2/21/50