Division of
ORE DRESSING AND
METALLURGY

010

DEPARTMENT OF MINES

CANADA

MINES BRANCH

O T T A W A June 29th, 1933

R E P O R T

of the

ORE DRESSING AND METALLURGICAL LABORATORIES

Experimental Tests on a Sample of Gold Ore
from Alcona Gold Mines Ltd., at Alcona, Ont.

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ORE DRESSING AND
METALLURGY

ALL OFFICIAL CORRESPONDENCE
SHOULD BE ADDRESSED TO THE DIRECTOR

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O T T A W A June 29th, 1933

R E P O R T

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ORE DRESSING AND METALURGICAL LABORATORIES.

Report No.....

Experimental Tests on a Sample of Gold Ore
from Alcona Gold Mines Ltd., at Alcona, Ont.

Shipment-

A sample of ore contained in two sacks, net weight approximately 200 pounds, was received March 23, 1933. The sample was submitted by Chas. A. Richardson, Manager, Alcona Gold Mines Ltd., Alcona, Ontario.

Characteristics of the Ore -

The gangue is chiefly white quartz. The metallic minerals present are pyrite, chalcopyrite, sphalerite, galena and native gold.

O T T A W A June 29th, 1933.

R E P O R T

of the

ORE DRESSING AND METALLURGICAL LABORATORIES.

Report No. ....

Experimental Tests on a Sample of Gold Ore from Aloona Gold Mines Ltd., at Aloona, Ont.

Shipment -

A sample of ore contained in two sacks, net weight approximately 200 pounds, was received March 23, 1933. The sample was submitted by Chas. A. Richardson, Manager, Aloona Gold Mines Ltd., Aloona, Ontario.

Characteristics of the Ore -

The gangue is chiefly white quartz. The metallic minerals present are pyrite, chalcopyrite, sphalerite, galena and native gold.

The pyrite is commonly disseminated as comparatively coarse grains in the quartz. The remaining sulphides form relatively fine stringers in the quartz and are mutually associated. The chalcopyrite, which is not abundant, is associated with rare sphalerite which contains numerous tiny dots of chalcopyrite. Calena is likewise comparatively rare, and is associated mostly with the chalcopyrite and sphalerite, but may occur alone in the quartz.

The only native gold observed was present as small irregular graines enclosed in galena. Its behaviour with etching reagents was misleading, because it was not attacked by KCN/ the effect on the polished surface was only to form an extremely thin yellowish film, which was perhaps responsible for protecting the metal against attack. Subsequent microchemical and spectrographic analyses, however, prove those grains to be impure gold.

An average analysis of the sample was as follows:-

Au	-	2.94 oz./ton
Ag	-	10.01 " "
Cu	-	0.43 %
Pb	-	3.10 %
Zn	-	0.50 %
Fe	-	5.10 %
S	-	5.00%

Experimental Tests-

A series of small scale tests was made on the ore to determine the best method of recovering the gold and silver from it. The work included tests by cyanidation, amalgamation, and concentration. About 20 percent of the gold is recoverable by amalgamation and from 10 to 15 percent by cyanidation and that with a rather high consumption of lime and cyanide. About 86 percent of the gold and silver can be recovered in a high grade flotation concentrate with a ratio of concentration of 6 or 7 to one. The flotation tailing will assay about 0.5 oz./ton in gold and this cannot be appreciably reduced either by amalgamation or

The pyrite is commonly disseminated as comparatively coarse grains in the quartz. The remaining sulphides form relatively fine stringers in the quartz and are mutually associated. The chalcopyrite, which is not abundant, is associated with rare sphalerite which contains numerous tiny dots of chalcopyrite. Galena is likewise comparatively rare, and is associated mostly with the chalcopyrite and sphalerite, but may occur alone in the quartz.

The only native gold observed was present as small irregular grains enclosed in galena. Its behaviour with etching reagents was misleading, because it was not attacked by KCN; the effect on the polished surface was only to form an extremely thin yellowish film, which was perhaps responsible for protecting the metal against attack. Subsequent microchemical and spectrographic analyses, however, prove these grains to be impure gold.

An average analysis of the sample was as follows:-

Au	-	2.94 oz./ton
Ag	-	10.01
Cu	-	0.45 %
Pb	-	3.10 %
Zn	-	0.50 %
Fe	-	8.10 %
S	-	6.00 %

Experimental Tests -

A series of small scale tests was made on the ore to determine the best method of recovering the gold and silver from it. The work included tests by cyanidation, amalgamation, and concentration. About 20 percent of the gold is recoverable by amalgamation and from 10 to 15 percent by cyanidation and that with a rather high consumption of lime and cyanide. About 88 percent of the gold and silver can be recovered in a high grade flotation concentrate with a ratio of concentration of 6 or 7 to one. The flotation tailing will assay about 0.5 oz./ton in gold and this cannot be appreciably reduced either by amalgamation or

tabling although a small amount of high grade table concentrate may be produced. Details of the tests follow.

Tests 1-4 Amalgamation & Flotation

In this series of tests four lots of the ore at - 14 mesh were ground in jar mills for 15, 20, 25, and 30 minutes respectively. The pulp was then amalgamated with mercury for thirty minutes and the amalgamation tailings were floated with the following reagents.-

Na_2CO_3	-	10 #/ton
Potassium Amyl Zanthate	-	0.2 "
Pine Oil	-	0.1 "

The flotation concentrates and tailings were assayed for gold and the amalgamation tailing assay was calculated back from these.

Summary Tests 1-4 -

Head Sample Au 2.94 oz./ton

Test No.	Product	Weight %	Assay Au oz	Recovery by Amalg.%	Recovery in Flot. Conc.
1	Flot. Conc.	13.7	12.71	20.1	59.1
	Flot. Tailing	86.5	0.71		
	Amal. Tail (Cal)	100.0	2.35		
2	Flot. Conc.	12.7	12.87	19.4	55.3
	Flot. Tailing	87.3	0.845		
	Amal. Tail (Cal)	100.0	2.37		
3	Flot. Conc.	10.9	13.98	19.7	51.8
	Flot. Tailing	89.1	0.94		
	Amal. Tail (Cal)	2.36			
4	Flot. Conc.	14.8	11.42	18.7	57.5
	Flot. Tailing	85.2	0.81		
	Amal. Tail (Cal)	100.0	2.39		

Tests 5-8 Cyanidation

In this series of tests four lots of the ore were ground in far mills for the same periods of time as those in tests 1-4. The pulp was then agitated for 48 hours at 2.5: 1 dilution in cyanide solution. The strength of the solution was kept at

tabling although a small amount of high grade table concentrate may be produced. Details of the tests follow.

Tests 1 - 4 Amalgamation & Flotation

In this series of tests four lots of the ore at - 14 mesh were ground in jar mills for 15, 20, 25, and 30 minutes respectively. The pulp was then amalgamated with mercury for thirty minutes and the amalgamation tailings were floated with the following reagents. -

Na_2CO_3	-	10 #/ton
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The flotation concentrates and tailings were assayed for gold and the amalgamation tailing assay was calculated back from these.

Summary Tests 1 - 4 -

Head Sample Au 2.94 oz./ton.

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	Flot. Tailing	87.3	0.845		
	Amal. Tail (Cal)	100.0	2.37		
3	Flot. Conc.	10.9	13.98	19.7	51.8
	Flot. Tailing	89.1	0.94		
	Amal. Tail(Cal)	100.0	2.36		
4	Flot. Conc.	14.8	11.42	18.7	57.5
	Flot. Tailing	85.2	0.81		
	Amal. Tail (Cal)	100.0	2.39		

Tests 5 - 8 Cyanidation

In this series of tests four lots of the ore were ground in jar mills for the same periods of time as those in tests 1 - 4. The pulp was then agitated for 48 hours at 2.5: 1 dilution in cyanide solution. The strength of the solution was kept at

2 lb./ton in KCN by additions of the salt from time to time. Protective alkalinity was maintained by the addition of lime. A screen test showed the grinding to be 52.9, 67.0, 71.0, and 80.4 percent minus 200 mesh in tests 5 - 8 respectively.

Summary Tests 5 - 8-

Head Sample Au 2.94 oz./ton, Ag 10.01 oz./ton

Test No.	Tailing Assay Au oz/ton	Tailing Assay Ag oz/ton	Extraction % Au	Extraction % Ag	Reagents Consumed #/ton KCN	Reagents Consumed #/ton CaO
5	2.57	7.10	12.6	29.1	8.6	16.9
6	2.50	6.95	15.0	30.6	9.6	17.0
7	2.60	7.02	11.6	29.9	9.6	17.2
8	2.70	7.08	8.2	29.3	9.1	17.2

The high consumption of cyanide is no doubt due to the copper contained in the ore and the high lime consumption due to its oxidized condition.

Failing to obtain any worth while recovery by amalgamation or cyanication, an attempt was made to concentrate the ore by flotation. A series of small scale tests was made for this purpose using different reagent combinations and different grinding. In one care the flotation tailing was tabled and in another care part of the flotation tailing was amalgamated and cyanided and of it cyanided only. Selective flotation of the sulphides, tried in test no. 10, proved a failure so the concentrates were mixed together and assayed as a bulk concentrate.

Test No. 9 - Flotation.

The ore at - 14 mesh was ground for 40 minutes in a jar mill, the charge being as follows. -

Ore	-	2000 grams
Water	-	1500 "
Boda Ash	-	20 lb./ton
Arofloat #25	-	0.07 "

Reagents to cell -

Potassium Amyl Zanthate	0.1 lb./ton
Pine Oil	0.05 " "

8 lb./ton in KCN by additions of the salt from time to time. Protective alkalinity was maintained by the addition of lime. A screen test showed the grinding to be 52.0, 67.0, 71.0, and 80.4 percent minus 200 mesh in tests 5 - 8 respectively.

Summary Tests 5 - 8 -

Head Sample Au 0.04 oz./ton, Ag 10.01 oz./ton.

Test No.	Tailing Assay		Extraction %		Reagents Consumed #/ton	
	Au oz/ton	Ag oz/ton	Au	Ag	KCN	CaO
5	0.07	7.10	12.0	29.1	0.0	16.9
6	0.00	6.95	15.0	30.6	0.6	17.0
7	0.00	7.02	11.0	29.9	0.6	17.2
8	0.70	7.00	8.2	29.3	0.1	17.8

The high consumption of cyanide is no doubt due to the copper contained in the ore and the high lime consumption due to its oxidized condition.

Failing to obtain any worth while recovery by amalgamation or cyanidation, an attempt was made to concentrate the ore by flotation. A series of small scale tests was made for this purpose using different reagent combinations and different grinding. In one case the flotation tailing was tabled and in another case part of the flotation tailing was amalgamated and cyanided and part of it cyanided only. Selective flotation of the sulphides, tried in test no. 10, proved a failure so the concentrates were mixed together and assayed as a bulk concentrate.

Test No. 9 - Flotation.

The ore at - 14 mesh was ground for 40 minutes in a jar mill, the charge being as follows. -

Ore	-	2000 grams
Water	-	1500 "
Soda Ash	-	30 lbs./ton
Aerofloat #25	-	0.07 "

Reagents to cell -

Potassium Amyl Xanthate	0.1 lbs./ton
Pine Oil	0.05 "

The products were assayed for gold, silver, copper, lead and zinc.

Summary Test No. 9 -

Product	Wt. %	Assays Au oz/ton	Assays Ag oz/ton	Assays Cu %	Assays Pb %	Assays Zn %	Recovery % Au	Recovery % Ag	Recovery % Cu	Recovery % Pb	Recovery % Zn
Conc.	16.2	15.82	57.08	2.80	11.7	2.80	84.8	85.9	87.1	63.5	91.5
Tail.	83.8	0.55	1.81	0.08	1.30	0.05	15.2	14.1	12.9	36.5	8.5
Head (Cal)	100.0	3.02	10.76	0.52	2.98	0.50					

A screen test of the flotation tailing showed the grinding to be as follows: -

Mesh	%
+100	0.2
-100 +150	3.3
-150 +200	9.9
-200	86.6
TOTAL	100.0

Test No. 10 Flotation

Selective flotation was attempted in this test but did not prove successful so the concentrates were all united and treated as a bulk concentrate.

The ore at - 14 mesh was ground for 40 minutes in a jar mill, the charge being as follows:-

Ore	2000 grams
Water	1500 "
Na_2CO_3	30 lb/ton
NaCN	0.2 "
Aerofloat #25	0.07 "

Reagents to Cell,-

Cu, Pb float - Cresylic acid 0.07 lb./ton

Zn float - CuSO 1.0 " "

Sodium Aerofloat 0.1 " "

Cresylic acid 0.07 " "

Re float - Potassium Amyl Kenthate 0.2 lb./ton

Pine Oil 0.05 "

The products were assayed for Gold, silver, copper, lead and zinc.

Summary Test No. 9 -

Product	Wt. %	Assays					Recovery %				
		Au oz/ton	Ag oz/ton	Cu %	Pb %	Zn %	Au	Ag	Cu	Pb	Zn
Conc.	16.2	15.82	57.03	2.00	11.7	2.00	84.8	85.9	87.1	63.5	91.5
Tail.	83.8	0.55	1.81	0.08	1.30	0.05	15.2	14.1	12.9	36.5	8.5
Head (Cal)	100.0	3.02	10.76	0.52	2.08	0.50					

A screen test of the flotation tailing showed the grinding to be as follows:-

Mesh	%
+100	0.2
-100 +150	3.3
-150 +200	9.9
-200	86.6
Total	100.0

Test No. 10 Flotation

Selective flotation was attempted in this test but did not prove successful so the concentrates were all united and treated as a bulk concentrate.

The ore at - 14 mesh was ground for 40 minutes in a jar mill, the charge being as follows:-

Ore	2000 Grams
Water	1600 "
Na_2CO_3	50 lb/ton
NaCN	0.2 "
Aerofloat #25	0.07 "

Reagents to Cell.-

Cu, Pb float - Cresylic acid	0.07	lb./ton
Zn float - $CuSO_4$	1.0	lb./ton
Sodium Aerofloat	0.1	"
Cresylic acid	0.07	"
Fe float - Potassium Amyl Xanthate	0.2	lb./ton
Pine Oil	0.05	"

The products were assayed for gold, silver, copper, lead and zinc.

Summary Test No. 10 -

Product	Wt. %	Assay Au oz/ton	Assay Ag oz/ton	Assay Cu %	Assay Pb %	Assay Zn	Recovery % Au	Recovery % Ag	Recovery % Cu	Recovery % Pb	Recovery % Zn
Conc.	13.5	17.32	65.18	3.26	13.81	3.30	86.5	87.3	92.7	64.2	91.2
Tail.	86.5	0.42	1.48	0.04	1.32	0.05	13.5	12.7	7.3	35.8	8.8
Head (Cal)	100.0	2.70	10.08	0.47	2.91	0.49					

Test No. 11 - Flotation with Cyanidation and Amalgamation Tests on the Plotation Tailing.

In this test the reagent combination was the same as that for test No. 9 except that 1.0 lb/ton $CuSo_4$ was added to the cell near the end of the operation to see if it would reduce the tailing assay.

The ore at - 14 mesh was ground in a jar mill for 40 minutes as in test No. 9, the charge being as follows:-

Ore	2000 grams
Water	1500 "
Na_2Co_3	20 lb/ton
Aerofloat #25	0.07 " "

Reagents to Cell :-

Potassium Amyl Xanthate	0.1 lb/ton
Pine Oil	0.05 " "
$CuSo_4$	1.0 " "

The concentrate and tailing were assayed for gold and silver. A sample of the tailing was cyanided and another sample was amalgamated and then cyanided, the tailings in these three cases being assayed for gold only.

The products were assayed for gold, silver, copper, lead, and zinc.

Summary Test No. 10 -

Product	Wt. %	Assay Au oz/ton	Assay Ag oz/ton	Assay Cu %	Assay Pb %	Assay Zn	Recovery % Au	Recovery % Ag	Recovery % Cu	Recovery % Pb	Recovery % Zn
Conc.	13.5	17.52	65.18	3.20	13.81	3.30	86.5	87.3	92.7	64.2	91.2
Tail.	86.5	0.42	1.48	0.04	1.32	0.05	13.5	12.7	7.3	35.8	8.8
Head (Cal)	100.0	2.70	10.08	0.47	2.01	0.49					

Test No. 11 - Flotation with Cyanidation and Amalgamation Tests on the Flotation Tailing.

In this test the reagent combination was the same as that for test No. 9 except that 1.0 lb./ton $CuSO_4$ was added to the cell near the end of the operation to see if it would reduce the tailing assay.

The ore at - 14 mesh was ground in a jar mill for 40 minutes as in test No. 9, the charge being as follows:-

Ore	2000 grams
Water	1000 "
Na_2CO_3	20 lb./ton
Aerofloat #25	0.07 "

Reagents to Cell :-

Potassium Amyl Xanthate	0.1 Lbs./ton
Pine Oil	0.05 "
$CuSO_4$	1.0 "

The concentrate and tailing were assayed for gold and silver. A sample of the tailing was cyanided and another sample was amalgamated and then cyanided, the tailings in these three cases being assayed for gold only.

Summary Test No. 11 -

Product	Weight %	Assay Au oz/ton	Assay Ag oz/ton	Recovery % Au	Recovery % Ag	Reagents oz./ton KCN	Reagents oz./ton CaO
Concentrate	20.3	11.82	46.86	85.8	87.0		
Tailing	79.7	0.50	1.78	14.8	13.0		
Head (Cal)	100.0	2.80	10.93	100.0	100.0		
Amal. Tailing	79.7	0.38		10.8			
Cyanide Tailing	79.7	0.205		5.8		3.2	9.8
Amal Tailing Cyanided	79.7	0.125		3.5		3.8	10.2

Net Recovery of gold by Amalgamating flotation tailing 3.4%

Net Recovery of gold by Cyaniding flotation tailing 8.4%

Net Recovery of Cyanidation of amalgamation tailing 7.3%

Test No. 12 - Flotation and Tabling

In this test the flotation tailing was tabled to see if its gold content could be reduced in this way.

The ore at - 14 mesh was ground for 40 minutes in a jar mill, the charge going as follows:-

Ore	2000 grams
Water	1500 "
Na_2CO_3	20 lb/ton
Aerofloat#25	0.07 "

Reagents to Cell-

Potassium Amyl Xanthate	0.1 lb/ton
Pine Oil	0.05 "

All products were assayed for gold and silver.

Summary Test No. 12 -

Product	Weight %	Assay Au oz/ton	Assay Ag oz/ton	Recovery % Au	Recovery % Au
Flot. Concentrate	15.9	15.93	55.52	84.6	84.1
Table Concentrate	1.8	7.36	11.18	4.4	1.9
Table Sand Tailing	65.7	0.375	1.36	8.2	8.5
Table Slime Tailing	16.6	0.50	3.47	2.8	5.5
Head (Cal)	100.0	2.99	10.50	100.0	100.0

Summary Test No. 11 -

Product	Weight %	Assay Au oz/ton	Assay Ag oz/ton	Recovery % Au	Recovery % Ag	Reagents oz./ton KCN	Reagents oz./ton CaO
Concentrate	20.3	11.82	46.86	85.8	87.0		
Tailing	79.7	0.50	1.78	14.2	13.0		
Head (Cal)	100.0	2.80	10.93	100.0	100.0		
Amal. Tailing	79.7	0.38		10.8			
Cyanide Tailing	79.7	0.205		5.8		3.2	0.8
Amal Tailing Cyanided	79.7	0.185		5.5		5.8	10.8

Net Recovery of gold by Amalgamating flotation tailing 8.4 %

Net Recovery of gold by Cyaniding flotation tailing 8.4 %

Net Recovery by Cyanidation of amalgamation tailing 7.3 %

Test No. 12 - Flotation and Tabling

In this test the flotation tailing was tabled to see if its gold content could be reduced in this way.

The ore at - 14 mesh was ground for 60 minutes in a jar mill, the charge being as follows:-

Ore	2000 grams
Water	1500 "
$NaCO_3$	30 lb./ton
Aerofloat #25	0.07 "

Reagents to Cell -

Potassium Amyl Xanthate	0.1 lb./ton
Pine Oil	0.05 "

All products were assayed for gold and silver.

Summary Test No. 12 -

Product	Weight %	Assay Au ozs./ton	Assay Ag oz./ton	Recovery % Au	Recovery % Ag
Flot. Concentrate	15.9	15.93	55.58	84.6	84.1
Table Concentrate	1.8	7.56	11.18	4.4	1.9
Table Sand Tailing	65.7	0.375	1.35	8.2	8.5
Table Slime Tailing	16.6	0.50	3.47	2.8	5.5
Head (Cal)	100.0	2.99	10.50	100.0	100.0

Test No. 13 - Flotation

Extremely fine grinding was tried in this test as a means of obtaining a clean flotation tailing.

The ore at - 14 mesh was ground for one hour in a jar mill, the charge being as follows:

Ore	2000	grams
Water	1500	"
Na_2CO_3	20	lb/ton
Aerofloat #25	0.07	"

Reagents to Cell:

Potassium Amyl Xanthate	0.1	lb/ton
Pine Oil	0.05	lb/ton

The flotation tailing was screened on a 200 mesh screen, the plus and minus along with the concentrate being assayed for gold and silver.

Summary Test No. 13 -

Product	Weight %	Assay Au oz/ton	Assay Ag oz/ton	Recovery % Au	Recovery % Ag
Concentrate	15.0	15.6	63.62	78.1	83.9
+ 200 mesh Tailing	2.9	7.1	2.75	6.9	0.7
- 200 mesh Tailing	82.1	0.55	2.14	15.0	15.4
Head (Cal)	100.0	3.00	11.40	100.0	100.0

Screen Test Flotation Tailing

Mesh	Weight %
+ 200	3.4
- 200	96.6

Conclusions -

The ore, as represented by the sample submitted, does not respond readily to any ordinary method of treatment. The sample is rather highly oxidized and therefore consumes a relatively large quantity of lime during cyanidation and due to its copper content consumes an abnormal amount of cyanide. Extraction by cyanidation is very poor as may be seen from the

Test No. 13 - Flotation

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Ore	2000 grams
Water	1000 "
Na_2CO_3	20 lb./ton
Aerofloat #25	0.07 "

Reagents to Cell ;

Potassium Amyl Xanthate	0.1 lb./ton
Pine Oil	0.05 "

The flotation tailing was screened on a 200 mesh screen, the plus and minus along with the concentrate being assayed for gold and silver.

Summary Test No. 13 -

Product	Weight %	Assay Au oz/ton	Assay Ag oz/ton	Recovery % Au	Recovery % Ag
Concentrate	15.0	18.6	63.63	78.1	83.9
+ 200 mesh Tailing	2.9	7.1	2.75	6.9	0.7
- 200 mesh Tailing	82.1	0.55	2.14	15.0	15.4
Head (Cal.)	100.0	3.00	11.40	100.0	100.0

Screen Test Flotation Tailing

Mesh	Weight %
+ 200	3.4
- 200	96.6

Conclusions -

The ore, as represented by the sample submitted, does not respond readily to any ordinary method of treatment. The sample is rather highly oxidised and therefore consumes a relatively large quantity of lime during cyanidation and due to its copper content consumes an abnormal amount of cyanide. Extraction by cyanidation is very poor as may be seen from the

results of tests 5 - 8. Some grains of free gold were panned from a sample of the ore and while they appeared to be clean and of a nice bright yellow colour they showed no reaction whatever other than the formation of a thin yellowish film when treated with cyanite solution. Details of further microchemical and spectrographic analyses are given in a report of the Mineragraphic Laboratory, June 24, 1935, a copy of which is attached.

Maximum recovery by amalgamation was 20.1% of the gold and this dripped slightly with finer grinding to 18.7% at 80.4% through 200 mesh.

High grade flotation concentrates can be produced with a fair ratio of concentration but it has not been possible to produce a flotation tailing lower than 0.5 oz/ton in gold. This is probably due to the presence of limonite resulting from alteration of the pyrite and containing the gold originally in the pyrite from which it was formed.

A good recovery was obtained in test no. 11 by floating the ore and then amalgamating and cyaniding the flotation tailing. The final tailing assayed 0.125 oz/ton in gold which is high and there still resins(?) the problem of treating the flotation concentrate to recover the gold from it. While the overall recovery in this test was 96.5%, this figure might be greatly reduced if ore of the same characteristics but lower in grade were being treated.

It is therefore suggested that another sample of ore, taken from depth to avoid the oxidized condition and more representative of the grade to be milled, be submitted and further test work will be carried out.

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Maximum recovery by amalgamation was 80.1 % of the gold and this dropped slightly with finer grinding to 16.7 % at 80.4% through 200 mesh.

High grade flotation concentrates can be produced with a fair ratio of concentration but it has not been possible to produce a flotation tailing lower than 0.6 oz./ton in gold. This is probably due to the presence of limonite resulting from alteration of the pyrite and containing the gold originally in the pyrite from which it was formed.

A good recovery was obtained in test no. 11 by floating the ore and then amalgamating and cyaniding the flotation tailing. The final tailing assayed 0.125 oz./ton in gold which is high and there still remains the problem of treating the flotation concentrate to recover the gold from it. While the overall recovery in this test was 96.6 %, this figure might be greatly reduced if ore of the same characteristics but lower in grade were being treated.

It is therefore suggested that another sample of ore, taken from depth to avoid the oxidized condition and more representative of the grade to be milled, be submitted and further test work will be carried out.

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O T T A W A June 24th, 1933

R E P O R T

of the

ORE DRESSING AND METALLURGICAL LABORATORIES

- Mineralgraphic Laboratory -

Report No.....

Microscopic Examination of Gold Ore
from Alcona Gold Mines Limited,
Alcona , Ontario

by

Maurice Enycock

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The samples examined were taken from a shipment of gold ore from Alcona Gold Mines Limited, Alcona, Ontario, received by the Ore Dressing and Metallurgical Laboratories on March 23, 1933. Seven polished sections were prepared and examined microscopically, and grains of the native gold

O T T A W A June 24th, 1933.

R E P O R T

of the

ORE DRESSING AND METALLURGICAL LABORATORIES.

- Mineralographic Laboratory -

Report No. ....

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separated by J. D. Johnston during test treatment were investigated both chemically and spectrographically.

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The purpose of the examination was to determine the characteristics of the ore.

Mineralogy -

The gangue of the polished sections is chiefly white quartz. The metallic minerals present are pyrite, chalcopyrite, sphalerite, galena, and native gold.

The pyrite is commonly disseminated as comparatively coarse grains in the quartz. The remaining sulphides form relatively fine stringers in the quartz and are mutually associated. The chalcopyrite, which is not abundant, is associated with rare sphalerite which contains numerous tiny dots of chalcopyrite. Galena is likewise comparatively rare, and is associated mostly with the chalcopyrite and sphalerite, but may occur alone in the quartz.

The only native gold observed was present as small irregular grains enclosed in galena. Its behaviour with etching reagents was misleading, because it was not attached by KCN: the effect on the polished surface was only to form an extremely thin yellowish film, which was perhaps responsible for protecting the metal against attack. Subsequent microchemical and spectrographic analyses, however, proved these grains to be impure gold.

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The occurrence of native gold in the galena has been noted. Spectrographic analyses of the metallic

minerals gave the following:

Mineral	Au	Ag
Galena	faint trace	strong trace
Pyrite	nil	very faint trace
Chalcopyrite	nil	very faint trace
Sphalerite	nil	very faint trace

The very faint traces of silver reported in the pyrite, chalcopyrite and sphalerite are negligible.

Character of the Native Gold -

The resistance of the native gold to the attack of KCN has already been noted. A summary of the spectrographic analyses carried out on gold grains separated by J. D. Johnston is given below.

(1) Treated with hot HNO for 5 minutes.
The solution showed Ag - strong trace,
Au - nil.

(2) The residue from (1) was washed with 1:1 HNO_3 and with water, and then treated for 2 minutes with cold aqua regia.
The solution showed Au - essential
Ag - strong trace,
Ca, Mg, Ti, V - strong traces.

(3) The residue from (2) was treated for 15 minutes with hot aqua regia.
The solution showed Au - essential,
Ag - strong trace,
Ti, V - strong traces,
Ca, Mg - traces.

(4) The residue from (3) showed - Si - essential,
Ag - essential,
Au - essential,
Ti, V - traces.

It was noted that the residue after the above treatment was composed of individual grains of the same size and shape as the original gold grains. These grains, however, consisted of a grayish-white curmbly or ash-like material.

The above analyses show that the gold grains are exceedingly impure. They contain abundant Ag and Si,

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The solution showed Au - essential,
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The above analyses show that the gold grains are exceedingly impure. They contain abundant Ag and Si,

small amounts of Ca, Mg. Ti and V, and (not reported above) traces of Cu, Mn and Fe. Furthermore, the above impurities are so finely divided that they are imperceptible under the highest powers of magnification obtainable in this laboratory (approximately 10000).

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E. K. FOCKLER
MINING GEOLOGIST
TORONTO

Suite 1401,
80 Richmond St. W.,
Toronto, Ontario.

Mr. E. Farlinger,
Box 98,
Sioux Lookout, Ontario.

Dear Mr. Farlinger:

Re: Alcona Mines Limited

As you requested during our discussion in my office on February 26, I have reviewed the various mine records in my files in order to decide whether there is any merit in retaining the Alcona property.

All things considered I have reached the same conclusion as set out in my report dated November 15th, 1939, a copy of which I handed to you during our interview. Inasmuch as there appears to be renewed interest in gold at the present time I would definitely advise that the Alcona property be kept in good standing as a valuable asset to the Company. The property can be described as a gold prospect of merit requiring further exploration to determine its ore potential.

Under the above mentioned report I recommended ten or twelve shallow drill holes at surface locations to test the three main veins in the immediate area of the shaft. There is considerable doubt in my mind whether the No. 1 vein lying to the north of the shaft was actually reached by the underground workings. Very definitely the Central vein lying to the south of the shaft has not been opened up by underground work.

The proposed surface drilling might amount to about 2,500', costing in the neighbourhood of $10,000. If results of the recommended drilling are positive it would seem to be in order to unwater the mine and proceed with underground exploration. This phase of the work would probably require about $50,000. to carry out, having in mind about 1,000' of lateral work. This stage could be conducted by using rented mining plant and equipment. If underground results were definitely encouraging then it would be in order to consider purchase of a permanent mining plant. I am assuming that the old plant has been dismanteled and sold. Any existing equipment on the property would probably be largely obsolete, although possibly the buildings could be rehabilitated.

A point for consideration is the present value of exploration performed during the earlier operations in 1937 and 1939. I would estimate that the mine openings and diamond drilling for which we have records of value could be presently appraised at about $150,000. The recommended programme of surface drilling assumed to cost $10,000., seems a small bet relative to the evaluation of valuable pre-development expenditure already paid for. The time factor in carrying out the existing underground work is also a valuable consideration.

The various mine records were duly submitted to the Head Office of the Company. I have not inquired whether they are still intact but I believe my office files are sufficiently complete to permit undertaking any future work at the mine. I shall be glad to provide what further information might be required if and when the Company decides to proceed with further work.

Yours truly,

E.K. Fockler

EKF:JL

E.K. Fockler.

Report on Alcona Mines, Limited Superior Jct. Area Kenora District

Summary

The principal veins on the Alcona property are situated on Claim K.3254, approximately 3500 feet southwest of the Split Lake granite stack. They occur in fractures which have been developed close to cb. andesite - porphyritic basalt contact.

No. 1 vein, which is the most important, strikes N 77° W and dips 65° South. It is made up of a massive to sugary quartz, containing variable amounts of pyrite, galena, chalcopyrite and sphalerite. It averages 19" in width and has been traced on surface for 360 feet. Channel sampling by the operators returned two shoots grading 9.65 dwts./1.4' for 97 feet and 12.70 dwts./1.7' for 155 feet. Sylvanite check sampling taken in the best location returned 27.30 dwts./2.2' for 33 feet.

Veins No. 2, 3, and Central occur south of #1 but have not experienced much development work.

A 325' shaft has been sunk on the property to test the possibilities of the veins. Two levels were driven from the 180' and 305' horizons to the south and north respectively. Very indifferent results were obtained from this work.

Conclusion

If the Alcona Mines Limited can prove the vertical continuation of #1 vein with their present program of diamond drilling, then the property will deserve a more active interest.

Location

The Alcona Mines Limited own nine claims; K.3253-54-55-56-57-58-62-63-64; in the Superior Junction Area of the Kenora District. The property is reached by a 6-mile wagon road leading from Alcona, a station on the branch line of the C.N.R. from Superior Jct. to Port Arthur.

A 2600 foot pipe line to Split Lake provided the necessary water supply.

Geology

The property lies in an area of Keewatin greenstones, andesites and porphyritic basalts, one-half mile southwest of the Split Lake granite stock. One mile to the south, a large area of granite occurs. This granite is mapped as post Temiskaming but of an earlier age than the Split Lake stock. Numerous past vein dykes of quartz diorite were noted throughout the workings.

The principal veins occur as fracture fillings which have been developed close to the contact between porphyritic basalt and cb. andesite. This contact strikes N 57° W and dips 56° South. The fractures which strike parallel and at low angles to the contact develop shears which, in places, attain a width of four feet. The quartz is sugary in character and appears to depend upon the presence of chalcopyrite, galena, pyrite and sphalerite for the carrying of gold values. Where the mineralization is weak, the values are correspondingly low.

No. 1 Vein, lying 240 feet North of the shaft, strikes N 77° W and dips 65° South. It crosses the por. basalt - cb. andesite contact at an angle of 20° and continues on its way with no apparent change in character. This vein has been traced on surface for a distance of 360 feet. On its east end it appears to die out; to the west it maintains a 22" width before plunging under the swamp. The vein is cut at eight places along its length by quartz diorite dykes which eliminate in the process a total of some 49 feet of vein material.

Three channel samples taken at intervals along the strongest mineralized portion of the vein returned an average of 27.30 dwts./2.2' for 33 feet. Two other channels taken at points on the vein farther west and with lesser amounts of mineralization, gave 1.60 dwts./1.2' and 3.20 dwts./1.9'.

Channel sampling by the operators indicates two ore sections - one West grading 9.65 dwts/1.4' for 97 feet and one on the east grading 12.70 dwts./1.7' for 155 feet. The area in between these shoots is almost entirely taken up by five crosscutting quartz diorite dykes. The east shoot apparently dies out on its east end but the west shoot is still open as it plunges into the swamp. Using their known areas, the two ore sections combine to a total of 33 tons per vertical foot.

It has been found from tests that the gold is directly connected with the galena and sphalerite and is thus not amenable to either flotation or cyanidation. This would mean concentration by flotation with resulting shipping and smelting charges.

No. 2 Vein strikes N 57° W and dips 60° South. It lies 20' south of the porphyritic basalt - cb. andesite contact and runs parallel to it. An uncut average of 5.60 dwts./1.5' for 250' is reported for this vein. Due to pits being filled in, no idea as to the correctness of this information could be obtained.

The Central Vein, lying some 200' south of the shaft, strikes N 50° W and dips 50° south. It appears to occur in fractured andesite and in one place reaches a width of 6 feet - generally, however, it occurs as a series of poorly mineralized stringers.

Development

Veins No. 1 and 2 were explored by four diamond drill holes during 1933. Results from this were very disappointing.

A vertical, three compartment shaft has been sunk on the property to a depth of 350'. Off this, levels have been established at 180' and 305'. The 180' level was driven south to explore the Central vein system. A narrow quartz vein intersected and then drifted upon averaged 10 inches in width with a grade of 3.00 dwts. At the completion of work, they appeared to have intersected the Central vein at the west end of the drift. No work was done upon it.

A crosscut north on the 305 level failed to intersect either No. 2 or No. 1 veins. At a point 120 feet from the shaft, a small vein was drifted upon but it failed to return any values. It is possible the vein has been faulted to a farther north position, in which case the crosscut was not driven far enough. This, however, is only a supposition and would require more detailed information for substantiation.

Proposed Additional Exploration

Under the direction of E. K. Fockler, the Alcona Mines Limited propose to enter a short diamond drill campaign to test No. 1 Vein at a shallower depth than has been done by previous work. In this way, it will be possible to tell if the drifting done on the 305 Level was carried out at the correct position.

It is also proposed to further test the Central vein by diamond drilling.

MINING PLANT EQUIPMENT

Buildings

Dry
Boiler House
Hoist House
Cap House
Machine and Blacksmith Shop
Assay Office - completely equipped
Bunkhouse) for 40 men.
Cookery)

Combined Office and Storehouse.

Equipment

2 40 H.P. Boilers
1 8 x 10 Steam Hoist
1 150 C.F. Compressor
1 Steam Generator
1 Air Receiver
3 Drifters
1 Steel Sharpener
1 Forge

Supplies

7000 tons Coal
300 cords Wood
900 ft. Used Drill Steel
6 Stilson Wrenches
100 ft. Air Hose
1500 ft. of 2" Pipe
1500 ft. of 1" Pipe

Numerous other small supplies.

D. K. Burke.

D.K.B.

Kirkland Lake, Ont.
Aug. 9, 1939.

COPY

80 Richmond St. West,
Toronto, Ontario,
November 15, 1939.

The Directors,
Alcona Mines Limited,
372 Bay Street,
Toronto, Ontario.

Dear Sirs:

In my report dated August 31st, 1939 recommendations were made for the further investigation of the principal veins in the mine area with a view to determining the advisability of resuming underground work on certain of these deposits in the near future. A limited amount of surface mapping and diamond drilling was outlined as necessary to ascertain immediate possibilities on the No. 1 and Central veins. It was estimated this work would cost in the neighborhood of $5,000.00, and require about two months to complete.

At a meeting of the directors of the Company held on September 11th, 1939, arrangements were made to carry out recommendations appertaining to examination of the Central Vein. An immediate program of 750 feet of diamond drilling together with surface mapping and sampling was approved.

On September 22nd C. R. Chataway arrived at the property to act as resident engineer for the duration of the above program. Geological work on the Central Vein in the mine area was commenced immediately. The position of the underground workings was tied in relative to the surface exposures in order to accurately locate bore holes planned to test the Central Vein structure.

A diamond drilling contract for 750 feet was awarded to Boyles Bros. of Port Arthur on September 19th. The drilling equipment reached the property October 9th and operations commenced immediately. The drilling program was completed October 21st for a total of 777 feet, following which I visited the property for inspection of the work to November 1st. During this recent program 41 surface samples and 20 core samples, all from the Central Vein System, were submitted to J. W. N. Bell, Kenora, for assay.

SUMMARY OF SURFACE RESULTS

Present work has been confined almost entirely to investigation of the Central Vein System in the mine area. Several surface trenches were cleaned out and where possible resampled. Since no record of surface workings on the Central Vein structure was available it was necessary to examine the showings in some detail. As a result of this work three veined sections of some importance have been outlined on the Central Vein system. For reference the sections are designated hereunder as C-1, C-2 and C-3. The showings noted lie in the immediate vicinity of the mine workings and occur at some 250 ft. intervals along the strike of the Central zone shearing. Results of our recent surface sampling on these sections are summarized as follows:

Occurrence	Approx. Dist. from 180' L. Crosscut	Actual Length	Probable Length	Ave. Width	Gold Oz.	Gold $35/oz.
C-1	0 - 80' West	45'	80'	8.0'	.09	3.15
C-2	275' East	50'	100'	10'	Approx.	4.00
C-3	500' East	60'	100'	3.4'	.27	9.45
C-4	100' East	30'	50'	1.5'	.65	22.75

Detailed mapping of the Central vein reveals a rather irregularly developed veined, silicified shearing carrying a moderate dissemination of sulphides and along which lie a number of lens of mineralized vein quartz. Subsidiary to the Central shearing are narrow quartz veins occupying tension fractures developed at angles of 30° to 40° to the main shearing. Vein matter in the main shearing strikes about N. 50° W. and dips 30 to 70° southwesterly whereas the subsidiary veins strike nearly due west, dip at somewhat higher angles, and occur on either side of the main fracturing.

Section C-1 located about 200 ft. southwest of the Shaft is exposed by two trenches marked 2/W and 3-W on the accompanying map. Vein matter is sugary white to bluish quartz carrying scant pyrite and a trace of galena. The highest individual assay secured on this occurrence is $5.60 over 4½ ft. The quartz vein reported encountered at the west heading of the west drift on the 180 ft. level is undoubtedly a downward extension from the surface exposure described. Gold values reported on the underground exposure are low and similar to those obtained at surface. On a surface assay sketch plan prepared in 1936 assays shown in a position corresponding to the C-1 occurrence are very much higher than obtained during the recent work. It is thought that the values shown may be a compilation of assays taken at

various points throughout the Central Vein structure and consequently are useless for purposes of evaluation.

Section C-2 exposed in trenches 3-E and 4-E approximately 270 ft. east of section C-1, consists of a mineralized, silicified veined zone in altered andesite. The more quartzose phases of the vein matter are well mineralized with fine pyrite, and subordinate amounts of chalcopyrite, galena and sphalerite. Trench 3-E gave an average of $4.00 gold at $35/oz. across a width of 19.3 feet with values ranging up to $10.50 over 3½ ft. The full width of the lens could not be ascertained in trench 4-E owing to heavily caved ground. Trenches 1-E and 2-E located about midway between sections C-1 and C-2 were too badly caved to reach bedrock readily so that no surface information is available for the intervening 270' interval. The widespread, strongly developed character of the veining in section C-2 is particularly encouraging in respect to tonnage possibilities.

Section C-3 lying 250 ft. east of C-2 is a relatively narrow, well-defined shearing containing well mineralized vein quartz and schist striking at an acute angle to the main Central shearing. Surface sampling indicates that commercial gold values are persistent over an actual length of 60 ft. No surface work of any account has been done here. The occurrence is open at each end, disappearing beneath overburden. Mineralization consists of fine pyrite and fairly plentiful galena and sphalerite.

Section C-4 marks a narrow subsidiary vein exposed just west of the mine road at a point near the main shearing. It is a well-fractured quartz vein averaging 1½ ft. in width carrying plentiful galena, pyrite and chalcopyrite. Two surface samples gave commercial gold values.

Sampling elsewhere on the Central Vein structure failed to indicate gold values of importance. Trenches 4, 5 and 6 west show values ranging from 35 cents to $1.75. Most of the so-called "angular" veins are narrow and poorly mineralized. It is not known whether enrichments occur where these occurrences come off from the main showing. It is noted, as in the case of the #1 vein north of the shaft, that wherever galena occurs in abundant amount a correspondingly good gold content is found.

SUMMARY OF DIAMOND DRILLING RESULTS

Under the recent program three parallel holes totalling 777 ft. were put down to probe the Central Vein at the 180' level at intervals of 125 ft. easterly from the mine workings. The holes were drilled normal to the strike and dip of the main Central shearing.

Hole No.	Location of Collar	Dip at Collar	Bearing	Length of Hole	Date Completed
6	100' S. 180' Xcut heading	50°	N 42° E	264'	Oct. 15, 1939
7	125' E.Hole 6	60°	N 42° E	260'	Oct. 19, 1939
8	250' E.Hole 6	50°	N 42° E	253'	Oct. 22, 1939

Hole No. 6 is briefly summarized as follows:

0	-	16'	Casing
16'	-	37'	Basalt
37'	-	141'	Quartz diorite porphyry
141'	-	187.5'	Basalt
187.5'	-	190'	Vein Zone
190'	-	215.5'	Basalt
215.5'	-	221'	Vein Zone (Central Vein)
221'	-	264'	Andesite

Throughout this bore section are numerous narrow sections of schist, silicified material and barren white quartz veinlets, none of which are commercially important. The most promising sections were split for assay, returning only low values in gold. The Central Vein was encountered at its calculated position at about the 100 ft. horizon. It consists of mineralized carbonate schist carrying irregular quartz veinlets and scant sulphides. A 5' core section on the vein assayed $1.40. The character of the intersection does not resemble the occurrence described at surface or that reported exposed in the underground workings. It would appear that the Central Vein at this point is below commercial grade and of insufficient importance to open up in the immediate future. The intersection lies about 30 feet beyond the heading of the south crosscut, and about 100 ft. east of the reported intersection in the west drift heading.

The occurrence of a 100 ft. section of quartz diorite porphyry in the upper part of this hole was unexpected since no surface exposure of the intrusive is uncovered in the vicinity. This intrusive may very conceivably have some structural relation to fracturing in the Central Vein system.

Hole No. 7 was logged as follows:

0	-	7.5'	Casing
7.5	-	9.5'	Quartz porphyry
9.5	-	82'	Basalt
82	-	88.5'	Vein material
88.5	-	223.2	Basalt with narrow porphyry dykes
223.2	-	226.2	Vein Zone (Central Vein?)
226.2	-	243	Andesite
243	-	255	Basalt
255	-	260	Diorite porphyry.

Two vein sections were encountered in this hole, the lower of which from 223' to 226' is thought to represent the Central Vein. The vein matter consisting of white quartz carrying scant pyrite and a trace of galena, assays 'Nil'. The upper vein section occurring at 82 ft. in the hole is believed to represent the downward continuation of an "angular" vein exposed at surface near the bore section. The section in question assays 70 cents over 3.7 feet. The surface occurrence gave $3.85 over 2.0 feet.

Since the intersection on the Central Vein indicates a weakly developed shearing containing a small amount of vein matter it may be inferred that no important lens occurs in close proximity to the bore section. The surface exposure of the "angular" vein designated C-4 is believed to dip south too steeply to have been encountered in Hole 7.

Hole No. 8 is summarized as follows:

0	-	5'	Casing
5	-	142'	Basalt
142	-	183'	Andesite
183	-	231'	Basalt
231	-	240.5'	Porphyry dyke
240.5	-	253'	Andesite

While a number of narrow mineralized quartz veinlets and schistose sections were encountered in this hole there is no definite intersection on the Central Vein. It would appear that the plane of Hole 8 lies slightly to the west of the surface section C-2. Several narrow schistose sections in the lower portion of the hole may represent the fingering out of the C-2 occurrence at this horizon. None of the mineralized sections carried more than a trace of gold. At 100 ft. in the hole a one-foot section of vuggy quartz and schist carrying abundant pyrite and galena assayed 2.44 oz. ($85.40). This intersection is believed to represent a very narrow, "angular" vein which cannot be correlated with any known surface occurrence.

None of the above bore sections indicate commercially important vein structures in the area probed along the Central Vein. Although the holes are spaced at horizontal intervals of 125 ft. it is unlikely that important lenses occur intermediate to the bore sections. It is felt, therefore, that the ground in question has been satisfactorily probed and that resumption of underground work in the immediate vicinity is unwarranted at this time.

Although the No. 1 and 2 Veins lying to the north of the shaft have recently been located accurately with respect

to the position of the underground workings no check sampling was done on the occurrences under this program. As pointed out in my previous report five samples were cut on the No. 1 Vein by visiting Sylvanite field men last August. According to their results the main surface exposure on the No. 1 vein gives a weighted uncut average of 1.44 oz. over a width of 2.2' for an actual length of 30 feet. Gold values elsewhere along the No. 1 vein, judging from appearance of the exposures, will be very much lower than the above noted average. Horwood reports in the Bureau of Mines' report, Volume 46, Part 6, 1937, an average of .29 ozs. over a width of 17 inches for a length of 100 feet. west of the above mentioned section on No. 1 Vein. He also gives an average of .28 ozs. over 1.6' for a length of 250 ft. on the No. 2 vein which lies 50 to 60 ft. south of No. 1 Vein. These figures were apparently supplied by the former mine operators. Exposures in trenches on the No. 2 Vein substantiate the widths mentioned. The character of the vein stuff is such that important gold values could be anticipated. The north cross-cut on the 305 ft. level would appear to have cut the downward extension of the No. 2 Vein at a point where little or no true vein matter is developed. A 15 ft. section of schist shown at the contact of the porphyritic lava and greenstone corresponds with a sheared condition found at surface along the contact. From this it is inferred that the downward extension on the No. 2 vein proper would lie west of the 305' Level crosscut and some 20 or 30 feet south of the west 305' Level drift. No bore hole has been put down in this section to establish the position of the vein below surface. Reasonas were given in my previous report for the apparent failure to locate the downward extension of the No. 1 Vein in the underground workings. A limited program of drilling was recommended to probe the downward extension of No. 1 and 2 veins but owing to the decision to confine work to the Central Zone the proposed drilling has not been executed. Possibilities on the No. 1 and No. 2 Veins thus remain to be fully investigated.

RECOMMENDATIONS

As stated above, results of the recent work do not justify immediate resumption of underground operations. Disclosures as to possible ore shoots on the Central Vein, however, have resulted from detailed surface mapping and sampling. These findings together with possibilities indicated on the veins north of the shaft indicate in my opinion that further investigation of the mine area is warranted. The proposed work could be performed most advantageously, I believe, by means of a limited program of shallow diamond drilling laid out to probe the indicated shoots at points above the 100' horizon in order to readily determine length and continuity of the occurrences in question. A tentative plan is to put down 10 or 12 such holes on the three main veins in the mine area. This would entail approximately 1500 ft. of drilling and cost in the neighborhood of $4,000.00. Provided results are encouraging,

a few deeper holes should then be put down to establish objectives for underground work. The complete program would cost approximately $7,500.00.

The number of gold bearing occurrences scattered in the immediate vicinity of the mine, together with the favorable geological features noted makes the property an unusually attractive prospect, deserving of careful investigation.

Accompanying this report is a plan showing surface trenches and underground workings. Bore hole sections and recent assay results are incorporated, and the more important geological features indicated. Diamond drill hole logs, sample records and drill hole sections have been submitted for the Company's files. Cores from the recent drilling are stored in the office building at the property in charge of the caretaker. The old decomposed powder in the magazine has been burned and all danger from this source thus eliminated.

Respectfully submitted,

"E. K. Fockler"

ALCONA MINES LIMITED

SURFACE SAMPLE RECORD

Vein No.	Sample No.	Width	Trench	Remarks	Gold Ozs.	$
Central	351	Grab off dump	2E	White and blue quartz. Fairly good copper and iron mineralization	0.02	0.70
Central	352	2.7'	4E E.side	8.3'-11.0' N. of Centre Line. Iron qtz. Scant pyrite & chalcopyrite on hanging and foot walls. Mainly barren	0.20	7.00
Central	353	3.0'	3E E.side	7.0'-10.0' S. of Centre Line. Rotten schist - silicified andesite and quartz. Poor sample as oxidation bad. Only one 5" band solid mineralized andesite and one 4" band white quartz slightly mineralized.	0.04	1.40
Central	354	2.5'	3E E.side	4.5'-7.0' S. of Central Line. Silicified andesite and blue and white qtz. Hanging wall quartzitic for 1.3'. Remainder highly silicified andesite. Hanging wall mineralization far.	0.02	0.70
Central	355	2.5'	3E E.side	2.0'-45' S. of centre line. Quartz ankerite veining in silicified greenstone. Well mineralized by pyrite and some chalcopyrite.	0.10	3.50
Central	356	5.0'	3E E.side	2.0' S. and 3.0' N. of Centre Line. Greenstone and few quartz stringers Poorly mineralized.	0.01	0.35
Central	357	1.0'	3E E. side	3.0'-4.0' N. of Centre Line. White bull qtz.	0.02	0.70
Central	358	2.0'	3E E.side	4.0'-6.0' N. of Centre Line. Slight qtz. ankerite veining in schist. Highly silicified	0.10	3.50
Central	359	1.5'	3E E.side	6.0'-7.5' N of Centre Line. White quartz. Slight chalcopyrite mineralization. Streaks of mineral confined to hangingwall margin.	0.08	2.80

ALCONA MINES LIMITED

SURFACE SAMPLE RECORD

Vein No.	Sample No.	Width	Trench	Remarks	Gold Ozs.	¢
Central	360	2.5'	3 E E.side	7.5'-10.0' N. of Centre Line. Mineralized schist and quartz veining. Well mineralized, silicified schist with quartz ankerite veining. Schist on footwall	0.14	4.90
Central	361	2.5'	3 E W.side	6.0'-8.5' S. of Centre Line. Well mineralized quartz ankerite.	0.28	9.80
Central	363	3.5'	3 E W.side	0.5'-4.0' N. of Centre Line. White quartz. Narrow zone along hanging wall carrying chalcopyrite. Remainder barren	0.10	3.50
Central	362	3.5'	3 E W.side	1.0'-4.5' S. of Centre Line. Quartz ankerite. Band of min. silicified schist near footwall.	0.06	2.10
Central	364	4.5'	3 E W.side	4.0'-8.5' N. of Centre Line Mainly quartz ankerite. 1.0' band min. greenstone on hanging wall. Rest well min. quartz ankerite, pyrite, chalcopyrite, slight galena	0.14	4.90
Central	365	3.5'	3 E W.side	8.5'-12.0' N. of Centre Line. Quartz ankerite veining in silicified schist. Fair pyrite min.	0.06	2.10
Central	366	3.5'	3 E W.side	12'-15'.5 N. of Centre Line. Quartz ankerite veining in silicified schist. Well min. by Copper, iron and lead sulphides	0.30	10.50
Central	367	3.0'	3 W	2.0' N-1.0' S. of Centre Line. Well min. blue and white quartz. Blue qtz. carries ankerite & finely disseminated min. white qtz. coarse crystals pyrite, slight galena shows in both.	0.04	1.40
Central	368	3.5	3 W	1' - 4'.5 S. of Central line. Blue qtz. some ankerite. Well min. by finely disseminated pyrite. Some galena and chalcopyrite. Very hard.	0.02	0.70

ALCONA MINES LIMITED

SAMPLE RECORD

Vein No.	Sample No.	Width	Trench	Remarks	Gold Ozs.	Gold $
Central	369	2.5'	3 W E.side	4.5'-7' S. of Centre Line Blue qtz. & ankerite well min. Generally well min. with iron, copper and lead sulphides	0.03	1.05
Central	370	5.0'	3 W	40'-45' S. of Centre Line. Rotten schist with quartz veining. Poor sample as oxidation bod. 2 bands quartz to 5", fair mineralization by pyrite.	0.04	1.40
Central	371	3.0'	E.of 6E.	47' E. of north end of Trench 6 E. Quartz ankerite and quartz. Well min. by copper, iron and lead sulphides	0.88	30.80
Central	372	3.0'	E.of 6E.	35' E. of north end of 6 E. Mainly quartz. Well min. by copper, iron and lead sulphides	0.18	6.30
Central Zone	374	1.5'	6 E	26'-29' N. of 637' on Centre Line. Qtz. ankerite veining in rusty quartz porphyry. Poorly mineralized.	0.03	1.05
Central Zone	375	3.5'	6 E	Quartz and siliceous schist. Fair min. in streaks. Quartz confined to 8" band on hanging and footwalls. Centre highly siliceous, quartzitic schist.	0.04	1.40
Central	373	1.5'	6 E	75' N. of Centre Line. White quartz. Only very slight iron & lead sulphides	0.22	7.70
Central	376	3.5'	2 W E.side	0 - 3.5' S. of Centre Line. Mainly blue quartz. Well min. by iron, slight copper and lead sulphides. Narrow band of schist near footwall.	0.12	4.20

ALCONA MINES LIMITED

SAMPLE RECORD

Vein No.	Sample No.	Width	Trench	Remarks	Gold Ozs.	$
Central	377	4.5'	2 W E.side	3.5'-8.0' S. of Centre Line. Quartz carbonate well min. by finely disseminated pyrite. Slight copper & lead sulphides.	0.16	5.60
Central	378	3.5'	3 W W.side	3'-6.5' S. of Centre Line. Mainly quartz, slight iron, copper and lead sulphides.	0.03	1.05
Central	379	3.5'	3 W W.side	3' S.-0.5' N. of Centre Line Similar to No. 378	0.04	1.40
Central	380	3.2'	5 W	17.0'-20.2' S. of Centre Line. Quartz carbonate stringers in siliceous schist. Fair pyrite min. in proximity to stringers. Only streaky pyrite in silicified schist.	0.04	1.40
Central	381	3.0'	5 W	20.2'-23.2' S. of Centre Line. Quartz carbonate veining in silicified schist. Fair pyrite min. disseminated throughout. 4" band of white quartz near hanging wall.	0.05	1.75
Central	382	2.5'	4 W E.side	9.0'-11.5' S. of Centre Line. Mainly rotten schist with few quartz stringers. Very poor sample of badly oxidized material. Trench not blasted into.	0.02	0.70
Central	383	2.5'	4 W E.side	11.5'-14.0' S. of Centre Line. Mainly white quartz, scant pyrite. Badly oxidized.	0.04	1.40
Central	384	2.5'	4 W	14.0'-16.5' S. of Centre Line. Quartz ankerite stringers in silicified greenstone. Fair mineralization by pyrite. Badly oxidized.	0.04	1.40

Note: Trench No. 4 is 210 ft. westerly from tie-line between shft and Centre Line.

Trench No. 5 is 235 ft. " " " " " " "

Offshoot of Central 385	1.4'		60' westerly from 120' E. on Centre Line. Rose quartz, scant min. Scant pyrite. Odd speck of galena	0.26	9.1

SAMPLE RECORD

Vein No.	Sample No.	Width	Trench	Remarks	Gold Ozs.	$
Offshoot of Central	386	1.5'		50' westerly from 120' E. on Centre line. Rose quartz well min. by pyrite. Fair galena	1.20	35.70
Central	387	3.0'	6 W E.side	118'-121' S. from Centre Line. Quartz carbonate and silicified greenstone. Fair to poor pyrite min. Badly oxidized.	.01	.35
Central	388	2.5'	6 W E.side	112.5'-116' S. from Centre Line. Quartz carbonate in silicified greenstone. Poor sample - badly oxidized.	.04	1.40

Note
Trench 6 W is 676' west from tie-line between shaft & Centre Line.

Central Zone	389	4.5'	6 E	12' East of Sample No. 373. Quartz & schist. Quartz occurs as variable width stringers in schist. Slight lead, copper & iron sulphides. Trench not blasted into.	.28	9.80
Central Zone	390	4.0'	6 E	Mainly quartz and a little schist. 5' E. sample 373. Plentiful lead, copper & iron sulphides. Slight oxidation.	.50	17.50
Central Zone	391	2.0'	6 E	50' E. sample 375. Quartz, scant sulphides. Shows slight oxidation.	.08	2.80
100' E. and 150' S. of tie line of shaft. No. 3 Vein?	392	2.0'		Quartz & silicified schist. Quartz on hanging wall well mineralized by iron & lead sulphides. Schist well mineralized by iron. Foot-wall quartz barren.	.11	3.8

ALCONA MINES LIMITED

DIAMOND DRILL HOLE NO. 6

LOG:

D.D.H. No. 6
Location: Central Vein
Co-ordinates: 18.5' West; 263.0' South
Dip at Collar: 50°
at 125' : 53° (Corrected)
at 250' : 49° (Corrected)
Depth of Hole: 264.0 feet
Commenced: October 9th, 1939.
Completed: October 14th, 1939.
Logged by: C. R. Chataway, October 15th, 1939.

Footage	Remarks
0 - 16	Casing
16 - 37	Basalt, fairly massive. Occasional specks of pyrite throughout. From 20.5-20.7 - Carbonate schist. Mineralized by pyrite. From 20.7-21.0 - White quartz veinlet. Fair pyrite mineralization. At 22.7' - 1" blue quartz and pyrite veinlet.
37 - 141	Quartz feldspar porphyry. Odd cube of pyrite throughout. From 38.8-39.4 - Schist, slightly mineralized. 65.0-65.6 - Silicified section. 90.0-93.3 - Silicified section. 70.5-73.0 - Broken up.
141 - 187.5	Basalt; fairly massive. From 142.5-143.5 - White quartz. Barren. 161.5-164.0 - Mineralized carbonate schist with stringers of quartz. 182.3-183.8 - Sheared with occasional veinlet of quartz. Fair pyrite mineralization.
187.5- 190	Veined Zone From 187.5-188.0 - Sheared with occasional veinlet of quartz. Fair pyrite mineralization. 188.0-189.3 - White quartz. Fair mineralization by galena and pyrite. 189.3-190.0 - Sheared with occasional veinlet of quartz. Fair pyrite mineralization.
190 - 215.5	Basalt, fairly massive, medium-grained. From 192.0-193.0 - Sheared with occasional veinlet of quartz. Fair pyrite mineralization. At 204 - 3" mineralized section.

Log of D.D. Hole 6 (Continued)

190 - 215.5	Continued. At 206 - 3" sheared; blue quartz veinlets From 208.5-211.0 - Sheared section - mineralized. At 210.8 - 1" quartz and pyrite.
215.5 - 221	Veined Zone - Central Vein. From 215.5-217.0 - Mineralized schist. 217.0-217.5 - White quartz - scant sulphides. 217.5-221.0 - Well pyritized brownish schist, cut by numerous irregular veinlets of blue and white quartz.
221 - 264	Andesite; mainly fairly fine grained. 221 - 228.5 - Altered greenstone, hybrid type, phases with quartz phenocrysts. 222.0 - 2" of mineralized schist. 228.5-234.5 - Banded fine grained tuff? 234.8 - 3" of mineralized schist. 236.2 - 239.0 - Well mineralized schist and white quartz. 250.0 - 2" mineralized schist. 251.6 - 1" mineralized schist.

SAMPLES:

Core:		Ozs. Au.	$
Sample No. 451	161.5' - 164.0'	.06	2.10
" No. 452	187.5 - 189.8	.09	3.15
" No. 453	216.0' - 221.0'	.04	1.40
Sludge:			
Sample No. 454	180.0' - 190.0'	.06	2.10
" No. 455	210.0' - 220.0'	.03	1.05

cc: Messrs. W. B. Price
G. E. Harlinger

ALCONA MINES LIMITED

DIAMOND DRILL HOLE NO. 7

LOG

D.D. Hole No. 7
Location: Central Vein
Co-ordinates: 107.5' E; 215.0' S.
Bearing: N 42° E.
Dip at collar: 60°
at 220.0': 59° (Corrected)
Commenced: October 15th, 1939.
Completed: October 16th, 1939.
Logged by: C. R. Chataway, October 19 3th, 1939.

Footage	Remarks
0.0 - 7.5	Casing
7.5 - 9.5	Quartz porphyry, grey
9.5 - 39.0	Basalt; medium grained, becoming slightly finer grained.
	At 10' - 1" band white quartz, barren.
	From 25.3 - 26.0 - Brown schist (sharp contacts) with 2" band white vuggy quartz and pyrite (probably gold-bearing)
	From 26.1 - 26.8 - Brown schist with 2" band white quartz and pyrite.
	At 33.3 - 4" schisted
	At 38.0 - 1" pyrite and quartz
39.0 - 53.0	Andesite? fine-grained, massive
53.0 - 82.0	Basalt, medium coarse grained
	From 59.0-60.0 - Brownish schist with 1" blue quartz and pyrite veinlet.
	From 67.5-70.0 - Slightly sheared carbonated and pyritized with 2" white quartz veinlet at 68.2', slight galena.
	From 75.6-76.5 - Slightly sheared and pyritized. 1" white quartz veinlet.
82.0 - 88.5	Vein Material
	From 82.0-83.3 - Becoming sheared.
	From 83.3-84.0 - Brown schist and quartz. Plentiful pyrite.
	From 84.0-85.0 - White quartz. Scant galena and pyrite sulphides.
	From 85.0-85.8 - Mainly quartz ankerite. Very well mineralized by pyrite.
	From 85.8-87.0 - Brown schist and quartz. Well mineralized.
	From 87.0-88.5 - Becoming less sheared with 2" white quartz veinlet. Scant pyrite.

Log of D.D. Hole No. 7 (Continued)

Footage	Remarks
88.5 - 175.0	Basalt, medium coarse grained, massive. From 133.0-135.0 - Schisted material with veinlets of white quartz. Fairly poor pyrite mineralization. From 186.0-174.0 - Small phenocrysts of feldspar present. At 173.5 - 2" brown schist and quartz. Scatn pyrite and galena.
175.0 - 170.5	Basic dyke? fine-grained, compact, dark green
170.5 - 182.5	Feldsper porphyry, light grey, fine grained.
182.5 - 194.5	Basalt, medium-grained.
194.5 - 195.5	Feldspar porphyry, light grey, fine grained
195.5 - 209.0	Andesite, fine-grained rom 195.5-196.5 - Schisted with quartz veinlets. at 198.0 - 2" schisted with quartz veinlets.
209.0 - 220.0	Feldspar porphyry, dark grey
220.0 - 223.2	Andesite, fine-grained, massive with occasional veinlet of barren white quartz.
223.2 - 226.2	Veined Zone: From 223.2-224.9 - Vein Material. Mainly white quartz. Scant sulphides, mostly pyrite, scant filmy galena. Last 4" well pyritized. From 224.9-226.2 - Slightly sheared and mineralized. 1" white quartz at 226.0.
226.2 - 243.0	Andesite. Fine grained, massive. From 231.0-243.0 - Very fine grained with numerous blotches of grey silica. From 235.2-236.9 - White quartz. Very scant sulphides. Last 5" schist & quartz.
243.0 - 255.0	Basalt, fine grained. From 248.0-250.0 - Brown schist with numerous veinlets of mineralized white quartz. From 253.0-255.0 - Slightly sheared. 1" blue quartz veinlet at 254'.
255.0 - 260.0	Feldspar porphyry, medium coarse grained, light green, andesitic.
260.0	END OF HOLE.

ALCONA MINES LIMITED

SAMPLING RECORD

D.D. HOLE 7

Sample No.	Footage		Width	Gold Ozs.	Gold $
456	63.3 - 67.0		3.7	.02	.70
457	133.0 - 135.0		2.0	Nil	---
458	223.2 - 224.9		1.7	.01	.35
459	235.2 - 236.9		1.7	Nil	---
460	80.0 - 90.0	(Sludge)		.04	1.40
461	220.0 - 230.0	(Sludge)		.03	1.05
462	248.0 - 250.0		2.0	.02	.70
470	67.6 69.6		2.0	.01	.35

52J04SE0021 52J04SE0014 ZARN LAKE

900

M-22

SYLVANITE GOLD MINES, LIMITED

ASSAY SHEET

From _____ Date _____

SAMPLE No.	DESCRIPTION	WIDTH	VALUE PER TON

Assayer

December 2, 1939.

Mr. E. K. Fockler,
80 Richmond Street West,
TORONTO,
Ontario.

Dear Mr. Fockler:

Enclosed please find the copy of your Alcona report which you so kindly left here for our consideration. I am returning your maps under separate cover.

We have gone carefully over the information contained and I do not think that at this time we would be interested in undertaking any development work on the property.

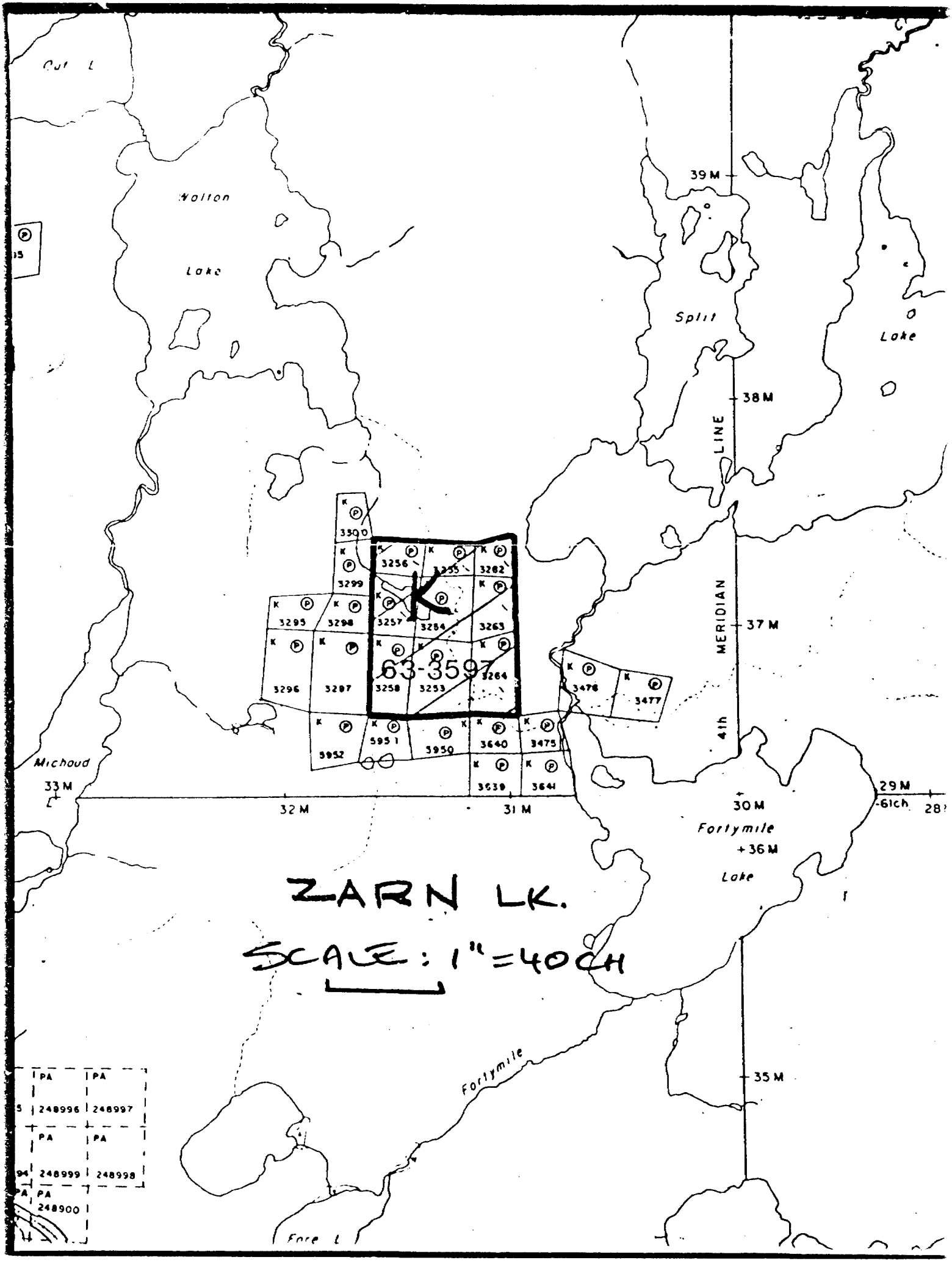
Thank you very much for bringing the property to our attention.

Yours very truly,

SYLVANITE GOLD MINES, LIMITED,
(No Personal Liability)
EXPLORATION DEPARTMENT,

GLH:LO
Encl.

Superintendent.


Cut L
Wolton
Lake
39 M
Split
Lake
38 M
LINE
MERIDIAN
37 M
4th
3300
3256
3255
3282
3299
3295
3298
3257
3284
3263
3296
3297
3258
3253
3264
63-3597
3476
3477
5951
3950
3640
3475
3952
3639
3641
Michoud
33 M
32 M
31 M
30 M
29 M
61ch
28
Fortymile
+36 M
Lake
ZARN LK.
SCALE: 1" = 40CH
35 M
Fortymile
PA
248996
248997
PA
248999
248998
PA
248900
Fork L

FOR ADDITIONAL INFORMATION SEE MAPS:

52T/04SE-0014 #1

August 16th, 1939.

Mr. E. K. Fockler,
60 Richmond St. W.,
Toronto, Ont.

Dear Mr. Fockler:-

In compliance with your request, I am forwarding the assays received from samples taken on the Alcona Mines property.

From the heavily mineralized section of No. 1 Vein, we obtained the following from three channels:

26.00 dwts/1.6'
40.40 dwts/2.4'
16.00 dwts/2.6'

West along the vein at distances of 50 feet and 200 feet respectively, from the west end of this section, we obtained channels returning 1.60 dwts/1.2' and 3.20 dwts/1.9'.

We would be extremely interested to learn the results you obtained from diamond drilling if this information will be available.

Yours truly,

D.K.B.

D. K. Burke

DKB:EB.
C.C. to G. E. Farlinger.

K 3300 | K 3807 | K 3808 | K 3809

K 3289 | K 3256 | K 3255 | K 3262 | K 3763

K 3298 | K 3257 | K 3254 | K 3263 | K 3764

K 3297 | K 3258 | K 3253 | K 3264 | K 3765

K 3269 | K 3270 | K 3271 | K 3640 | K 3475

K 3639 | K 3641

LAKE

SPLIT LAKE OR LAKE OF BAYS

K 3394 | K 3397 | K 3850

K 3395 | K 3396 | K 3851 | K 3853 | K 3858

CREEK

K 3476 | K 3877 | K 3852 | K 3854 | K 3857

POWELL SYNDICATE

LINE OF FOURTH MERIDIAN

K 3854 | K 3356

FORTY MILE LK.

BASELINE

200

31

SCALE: 1" = 10 CHAINS

52J04SE0021 52J04SE0014 ZARN LAKE

52J/045E-0014-#1

ARCONA GOLD MINES LTD.
CLAIMS NO. K.3253-3258
K.3262-3264 K.3850-K.3852
POWELL SYNDICATE
CLAIMS NO. K.3853-3858

PLAN OF PROPERTY OF ALCONA GOLD MINES LTD.
& CONTIGUOUS MINING CLAIMS NEAR ALCONA, ONTARIO.