To J. F. Olk

Date December 4, 1975

From J. C. Devitt

Subject ANVIL ORE RESERVES AND PIT CONFIGURATION

An analysis of the historical background of our present pit configuration and reserves has been completed. A synopsis of the steps leading from the original R.M. Parsons' Feasibility Study to the present model follows. Detailed comparison between steps, with justifications, are included where applicable.

R.M. Parsons' Final Feasibility Study, February 1967:

Parsons designed a pit to mine a stated 41,000,000 ore tons from what was then known as the No. 1 zone. The separation of zones 1 and 3 by faulting and waste rock intrusion was not then known. In actuality, ore in this pit was not strictly zone 1 ore as now defined, but crossed the dike zone and encompassed part of zone 3 ore as well. Location of ore benches by zone is as follows:

Benches	Location
4205-3785	Zone 1
3750-3715	Zone 1 and dike zone
3680-3575	Zones 1 and 3 and dike zone
3540-3365	Not included in pit design

Parsons' pit design does not go below the 3575 bench. Their bench inventory lists 38,916,518 cumulative tons in benches down to and including 3575, instead of their reported 41,000,000. The addition of the 3540 bench does bring the total to 40,962,636 tons. It is speculated that they inadvertently read their tonnage distribution chart incorrectly and wrongly included one extra ore bench that was not, in fact, in the pit design.

Parsons were instructed to assume that at least 5,000,000 tons of ore at a grade equal to the No. 1 deposit could be obtained from the No. 2 and No. 3 deposits. No pits were actually designed to confirm this. On an assumed stripping ratio of 6.4 to 1, they calculated 10,000,000 BCY of waste to mine this ore.

Their inventories were:

	Ore	Waste	Stripping Ratio
Zone 1	41,000,000 Tons	62,000,000 BCY	4.8/1
Zones 2 and 3	5,000,000 Tons	10,000,000 BCY	6.4/1
Total	46,000,000 Tons	72,000,000 BCY	5.0/1

Pit parameters were:

Pit Slope	40°
Bench Height	35 ft.
Tonnage Factor (ore)	3.10 Tons/BCY
Ore Projection Method	Polygonal, with outside contacts determined by projections from cross-sections

J.F. Olk Mining Study, April 4, 1967:

Mr. Olk also designed a pit to mine the Faro No. 1 as then known. This pit has a very similar configuration to Parsons' pit, includes the same benches (to 3575), and involves the same ore projections across the dike area and into zone 3.

Mr. Olk also designed a pit for zone 2 ore. None was designed for what was then known as zone 3 because drilling was incomplete. However, a tonnage figure was calculated for the plus 5% material in zone 3.

Pit parameters were: (Zone 1 Pit)

Pit Slope	40°, except NW corner - 45°
Bench Height	35 ft.
Tonnage Factor (ore)	3.18 Tons/BCY
Ore Projection Method	Polygonal, with maximum projection beyond drill holes 141 ft.

Since Parsons' and Olk's pits for zone 1 are very similar in configuration data, and intent, a comparison by benches follows:

Bench	M Tons Parsons	M Tons Olk
4205	162.5	102.6
4170	444.5	285.6
4135	372.1	328.2
4100	540.9	666.5
4065	737.6	788.6
4030	683.1	754.7
3995	1,377.8	902.7
3960	2,073.9	2,022.8
3925	2,778.7	2,645.2
3890	3,165.7	2,592.6
3855	3,556.2	3,345.9
3820	3,448.7	3,264.4
3785	3,534.6	2,520.4
3750	2,718.0	2,125.2
3715	3,215.1	2,549.7
3680	2,989.9	2,762.4
3645	3,299.1	2,951.4
3610	2,456.6	1,864.0
3575	1,361.7	889.3
Dilution	-	211.0
Total	38,916.7	33,573.2
Waste	62,000.0	66,138.5
Ore (BCY)	12,238.0	10,557.6
Total Material	74,238.0	76,696.1
Stripping Ratio	5.1/1	6.3/1

Mr. Olk's volumes for the Zone 2 pit are:

Ore	3,916,000 Tons (1,231,447 BCY)
Waste	8,622,000 BCY
Stripping Ratio	7.0/1

In conclusion, Mr. Olk derived a pit of basically the same configuration as Parsons, but by using a different approach on ore interpretations, came up with 5,343,500 tons less. Even with the steepened wall in the NW corner of Mr. Olk's pit, his waste calculation still came out 4,138,500 BCY more, with the resulting significant increase in stripping ratio.

Mr. Olk's pit, with modifications such as 45° slopes all around and 40 ft. benches, was used as a final pit design until 1970.

E.N. Pennebaker Reserve Estimate, October 3, 1967:

Mr. Pennebaker's estimate included all known plus 5% Pb + Zn material, without regard to any pit configuration. No attempt to differentiate between economic and non-economic "ore" was made. This estimate is summarized as follows:

Zone	Tons
1	34,206,995
3	24,106,611
2	5,159,335
Total	63,472,941

Zn Pb Ag
5.72 3.40 1.1 oz/ton
.9072 = metric "
311.2 gms/ton

Mr. Pennebaker's zone 1 estimate compares favourably with Mr. Olk's in-pit estimate. Zones 2 and 3 show substantial increases over both previous estimates. This is probably due to information from additional 1967 drilling as follows:

Zone 1	2 holes, total 1404 ft.
Zone 2	20 holes, total 4703 ft.
Zone 3	13 holes, total 9408 ft.

Ore Losses - Upper Benches, 1969:

As before stated, the working pit model up to 1970 was basically Mr. Olk's pit with steepened slopes and 40 ft. benches. Considerable amounts of predicted ore in upper benches failed to materialize. These losses, as compared to Mr. Olk's mining study, were:

Bench	Loss (tons)
4240	19,800 (gain)
4205	22,000
4170	161,700
4135	68,600
4100	345,500
4065	366,500
Total Ore Losses	944,500

1970 Ultimate Pit:

Bench plans were updated and revised to include all ore delineated by additional fill-in diamond drilling as follows:

No diamond drilling in 1968 and 1969

Year 1970 diamond drilling:

Zone 1 - 14 holes, total 5634 ft.

Zone 3 - 3 holes, total 2106 ft.

An ultimate pit was designed to recover all the then known ore in both zones 1 and 3.

Pit design parameters were:

Pit Slope	45°, double bench every other bench
Bench Height	40 ft.
Tonnage Factor (ore)	3.18 tons/BCY
Ore Projection Method	Rectangular Projection

Since the bench height, pit slope, and definitions of zones 1 and 3 had been changed since Mr. Olk's pit, no direct comparison by specific areas or benches is possible. This was a totally new pit concept, based on different parameters, including the lower lying and southerly parts of zone 3, which had been since drilled off and defined. This design is a major pit expansion beyond the scope of the earlier "zone 1" pits.

Material within this pit, projected to the start of mining, was calculated at:

Ore	
Zone	Tons
1	35,487,019
3	22,271,792
Total	57,758,811

Waste

119,102,152 BCY

Stripping Ratio

6.6/1

1973 Ultimate Pit:

In years 1971 and 1972, additional diamond drilling was done:

1971

Zone 3 - 5 holes, total 3080 ft.

1972

Zone 1 - 8 holes, total 2981 ft.

Zone 2 - 1 hole, 350 ft.

Zone 3 - 7 holes, total 3584 ft.

With this additional drilling, new bench maps and cross-sections were prepared. All prior data was edited in detail to eliminate errors. A systematized method of assigning areas of influence of tonnage and grade was developed which correlated with historical mining information, and full bench composite grades were adopted. The major effect of this was to delineate low grade pyritic sulfide lenses within the orebody. Most of these lenses were in the zone 3 portion, although some effect on the zone 1 portion was also noted. A loss of 4,304,396 tons in the zone 3 orebody from the 1970 estimate is derived from the delineation of these pyrite zones.

An ultimate pit was designed to fit the above orebody model. Parts of the projected ore were eliminated from the pit configuration due to excess stripping ratios. They are:

Zone	Tons
1	145,810
3	1,487,060
Total	1,632,870

Pit design parameters are:

Pit Slope	45°
Bench Height	40 ft. below 4030, 35 ft. above 4030
Tonnage Factor	3.18 Tons/BCY
Ore Projection Method	Rectangular, extended in direction of ore grade zonation

The measured quantities within the pit as of January 1, 1973 are:

Ore	
Zone	Tons
1	23,950,410
3	16,480,360
Total	40,430,770

Waste

90,252,560

Stripping Ratio

7.1/1

9.09.33104

Dike Losses - Zones 1 and 3, 1973:

Diamond drilling in 1973 in these zones was:

Zone 1 - 6 holes, total 2483 ft.

This drilling revealed the presence of a barren diorite dike approximately 100 ft. wide, standing almost vertically in the fault zone separating zones 1 and 3. Previously, ore had been projected up to the fault line. This projected ore was now seen to be waste for 50 ft. on either side of the fault line for the full depth of the orebody. Losses in reserves thus incurred are:

Zone	Tons
1	1,369,960
3	770,990
Total	2,160,950

Zone 2, 1973-1974:

Diamond drilling in 1973 was:

23 holes, total 3565 ft.

This drilling completed the grid pattern on zone 2 and supplemented some of the older holes that suffered from poor core recovery. This orebody was recalculated on the basis of 20 ft. benches to allow greater selectivity. In 1974, as part of a pit planning program, Laurich-Kennedy Associates designed a pit to recover all this ore. Materials in this pit are as follows:

Ore

3,424,003 Tons

Waste

5,609,776 BCY

Stripping Ratio

5.2/1

Laurich-Kennedy, 1974:

Laurich-Kennedy Associates of Seattle were commissioned to develop a lifetime mining schedule in the summer of 1974. They were given the pit configuration of the 1973 design, plus updated bench maps including the 1973 drilling (dike losses). Their pit volume measurements, as of July 1, 1974, are:

Zone	Ore Tons	Waste BCY	Stripping Ratio
1 and 3	34,660,508	90,597,642	8.3/1
2	3,392,236	5,609,778	5.3/1
Total	38,052,744	96,207,420	8.0/1

Reserves - January 1, 1975:

Diamond drilling in 1974 was as follows:

Zone 1 - 10 holes, total 3172 ft.

Zone 3 - 10 holes, total 4799 ft.

1) Cyprus Anvil:

Based on the above drilling, ore reserves were recalculated based on 1/1/75 status. They are:

	Tons	
	Inside L-K Pit	Outside L-K Pit
Zone 1	15,538,576	605,800
Zone 3	16,770,217	2,554,358
Total	32,308,793	3,160,158

2) E.N. Pennebaker:

E.N. Pennebaker also calculated the ore reserves during the year, also using the 1/1/75 status as a base. He got:

	Tons	
	Inside L-K Pit	Outside L-K Pit
Zone 1	15,245,076	1,663,075
Zone 3	19,032,207	5,993,363
Total	34,277,283	7,656,438

The tonnage factor used on all the ore reserve figures listed herein is 3.18 Tons/BCY. This factor has recently been investigated and changed to 3.5 Tons/BCY.

Present Zone 1:

A current look at the zone 1 pit shows the original ore volume contained as follows:

	Tons
Reserves remaining at 1/1/75 (Cyprus Anvil Estimate)	15,538,576
Milled prior to 1/1/75	13,799,034
Mined and stockpiled prior to 1/1/75	2,427,010
Total ore in zone 1 before mining	31,764,620

Comparing this to Mr. Olk's original estimate, less known ore losses:

	Tons
"Zone 1" Pit	33,572,900
Upper bench losses	– 944,500
Dike loss	– 2,160,950
Total	30,467,450

Therefore, the presently known zone 1 reserves compare favourably (within 1,300,000 tons) with Mr. Olk's original study when known losses are considered. By no means, however, could this zone ever have produced the 41,000,000 tons as called for in Parsons' feasibility study.

Zone 3 is a recently evaluated sector of the orebody, which has been taken into the new pit expanded from the original "zone 1" pit. At least half the zone 3 ore must be mined to produce the 41,000,000 tons as called for in the feasibility study, and this must be mined at a considerably higher stripping ratio than originally considered, because this ore is some 100 feet deeper than the main portion of the zone 1 orebody.

An ore reserve document book is being prepared, which will contain all details of the above outlined change sequences of the orebody and pit outline. One copy each will go to yourself and P. Taggart.

Also, to double check the total pit volumes remaining and to locate reasons for the sudden jump in stripping ratio between the 1/1/73 pit and the L-K pit (the same pit outline was used for both), pit status will be projected to 1/1/76 and total remaining ore tons and grade, plus remaining waste BCY will be calculated, checked, and tabulated. This will give a total pit inventory, by bench, as of 1/1/76. Results of this will be submitted by December 23, 1975.

JC Devitt

J. C. Devitt
Chief Mining Engineer

JCD/mm

cc. L. P. Taggart

SUMMARY

LARGE FARO DEPOSIT (ZONE 1/3)

EXCLUDING OXIDE RESERVES

NOT UPDATED WITH 1981 DRILLING.

Geological Reserves

As of April 1, 1981

Cut-Off Grade (Pb-Zn) %	~~3.0%~~	4.0%
Tonnage (000's ~~SDT~~ TONNES)	~~44,000~~ 37658	~~39,632~~ 34,156
Lead (% Pb)	~~2.8~~	3.0
Zinc (% Zn)	~~4.4~~	4.6
Silver (Ag g/tonne)	~~33~~	35

Adjusted Mine Reserves (Mill Feed)

As of April 1, 1981

Cut-Off Grade (Pb+Zn) %	-	4.0%
Waste (000's ~~YDS~~ BCM)		~~65,872~~ 50,366
Tonnage (000's ~~SDT~~ TONNES)		~~31,764~~ 28,816
Lead (% Pb)		2.9
Zinc (% Zn)		4.4
Silver (Ag g/tonne)		34

SUMMARY

FARO OXIDE STOCKPILE

CUT-OFF : N/A

TONNAGE (000's ~~SDT~~ TONNES)	~~1,800,000~~ 1,633,000
LEAD (% Pb)	2.9
ZINC (% Zn)	4.7
SILVER (Ag g/tonne)	33

SUMMARY

GRUM DEPOSIT

Note: ALL GRUM RESERVE FIGURES BASED BASED ON THE ~~NORCOMP~~—(NORANDA-~~COMPUTER CENTER~~) MODEL

CAMC INTERIM

Geological Reserves

Cut-Off Grade (Pb+Zn) %	-	4.0%
Tonnage (000's tonnes)		~~27,650~~ 30,781
Lead (% Pb)		3.1
Zinc (% Zn)		4.9
Silver (Ag g/tonne)		~~48~~ 49

Adjusted Mine Reserves (Mill Feed)

(Open Pit Only)

Cut-Off Grade (Pb+Zn) %	-	4.0%
Waste (000's M^3)		~~34,081~~ 45,688 *
Tonnage (000's tonnes)		~~15,583~~ 16,875
Lead (% Pb)		~~3.1~~ 3.0
Zinc (% Zn)		~~5.0~~ 4.9
Silver (Ag g/tonne)		47

* THIS FIGURE UNDERESTIMATES WASTE VOLUME. OVERBURDEN IN DUAL LAKE AREA HAS NOT BEEN FULLY TAKEN INTO ACCOUNT.

SUMMARY

VANGORDA DEPOSIT

BETWEEN SECTIONS 2W and 12E (EXCLUDES S.E. EXTENSION)

Geological Reserves

Cut-Off Grade (Pb+Zn) %	~~3.0%~~	4.0%
Tonnage (000's tonnes)	~~5,204~~	~~5,210~~ 4,950
Lead (% Pb)	~~3.0~~	3.3
Zinc (% Zn)	~~3.9~~	4.3
Silver (Ag g/tonne)	~~44~~	48
Gold (Au g/tonne)	~~0.76~~	0.74

Adjusted Mine Reserves (Mill Feed)

Cut-Off Grade (Pb+Zn) %	~~3.0%~~	4.0%
Waste (000's M^3)	~~7,307~~	7,542
Tonnage (000's tonnes)	~~5,323~~	~~4,520~~ 4,294 ~~4,[illegible]~~
Lead (% Pb)	~~3.0~~	3.3
Zinc (% Zn)	~~3.9~~	4.3
Silver (Ag g/tonne)	~~44~~	47
Gold (Au g/tonne)	~~0.75~~	0.73

GEOPHYSICAL EXPLORATION OF A LEAD-ZINC DEPOSIT IN YUKON TERRITORY

*by Edward O. Chisholm**

Abstract

Self-potential and magnetometer surveys were followed by a gravimetric survey in the northwestern Cordillera to outline successfully a flat-lying lead-zinc sulphide replacement deposit beneath 50 ft. of glacial overburden. This site of the survey is 125 miles northeast of Whitehorse, Yukon Territory, in mountainous terrain. Detailed diamond drilling verified the accuracy of the survey both as to boundaries and estimated tonnage of the deposit. Auxiliary surveys were carried out by aeromagnetic and geochemical methods. Graphitic schists interfered with the self-potential readings, but geochemical and magnetic results were helpful for indicating favourable terrain.

Location, Topography, Etc.

THE lead-zinc sulphide replacement body investigated by geophysical techniques described here is located on the headwaters of Vangorda Creek, a tributary of Pelly River, on the Vangorda Mines property of Prospectors Airways Company Limited. The surrounding country is mountainous but the deposit is in the centre of an intermontane valley approximately 2½ miles wide. Differences of elevation within the area of the survey are less than 100 ft., a feature which facilitated gravity work.

The replacement body lies beneath a glacial sand and boulder ridge varying in thickness from 25 to 80 ft. The ridge is traversed by Vangorda Creek, a small glacial stream, and a smaller tributary. Outcrop is exposed in the area of the survey at one location only on the bank of the stream, and consists of massive sulphides 100 ft. in length, 10 ft. in height. Permafrost, though usual in this latitude, was not encountered. Overlying the glacial material is a persistent layer of volcanic ash approximately 4 inches thick. The climate is sub-arctic and the soil profile is immature with zones of leaching and oxides penetrating less than a foot below surface. This results in a smaller, residual base-metal content of the soil and water than is normal to central latitudes.

The geophysical methods were started in advance of drilling, but were broadened in kind and magnitude as the drilling progressed and as different characteristics of the deposit were revealed. Initially the simplest methods, the self-potential and the magnetometer, were used. When it was apparent that graphitic schist was present, the electrical methods were discarded and magnetometer work was increased. The detail of this method was completed during the second season in time to indicate possible extensions of the deposit.

Further drilling indicated, however, that the magnetometer could not be relied upon solely to indicate sulphides because of the presence of other magnetic zones. At this stage gravimetric work was initiated and was completed over the deposit at the same time as the drill program. The results coincided so well that further drilling to extend the margins of the deposit was considered unnecessary. The excess mass calculation agreed so closely with the tonnage figure arrived at by drilling that it was decided also that further deep holes to explore the possibility of underlying zones were unnecessary.

Geology, Mineralization, Alteration

The deposit comprises an overlapping series of horizontal lenses of sulphides that appear to replace a favourable sedimentary bed; longitudinal section of the body is shown in Figure 1. Seventy-three diamond drill holes indicate a length of 3,200 ft. with an average width of 490 ft., and 9,400,000 tons of sulphide containing 3.16% Pb, 4.96% Zn, 0.27% Cu, 1.76 ozs. Ag, and 0.02 oz. Au; also, an additional 12,600,000 tons of low-grade to barren sulphides. The total mass of sulphides is estimated from diamond drilling to be in the order of 22,000,000 tons. The mineralized body extends from bedrock surface to a depth of 300 ft. Drilling to 1,000 ft. encountered no underlying body.

9.4
12.6
22.0

The host rocks comprise a flat-lying sedimentary assemblage which can be divided into two main zones, namely, one predominantly chloritic sericite schist, and the other predominantly graphitic schist. They are intimately associated with much intercalation at the edges of the graphitic horizon. The graphitic schist is minutely crumpled, breaks easily along cleavage planes, and contains narrow (up to 1 m.m.) quartz stringers

*Chief Geologist, Prospectors Airways Co. Ltd., Toronto, Canada.

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KERR ADDISON MINES LIMITED

(FOR INTER-OFFICE USE ONLY)

To W. M. Sirola From P. M. Kavanagh

Subject <u>Ore Reserves Calculations, Swim Lakes "A" Group, Yukon.</u> Date December 27th, 1966.

Attached is a data sheet I have worked out using yours and Fred's interpretation on the cross sections. If you refer to your copies of the sections you will note how I handled the various intersections, etc.

Using a tonnage factor of 8, which was the tonnage factor used in the Vangorda property calculations, the resulting figures of interest are the following:

4,750,000 tons of 1.5 oz. Ag, 3.8% Pb, 4.7% Zn

Those figures are not very different from the 5,000,000 tons of 1.5 oz. Ag. and 9% Pb-Zn which I presented at our last Board meeting, but of course they are based on a much more careful interpretation of our data.

The tonnage figure can very probably be expanded in the area east of A-30 and A-31 and south of A-19 and A-17, also possibly south of A-24 and A-20, and east of A-37. The tonnage figure does not all represent open-pitable reserves; I intend to make a calculation of the indicated open-pitable portion as soon as I have time.

I would appreciate any comments you or Fred may have.

Paul M. Kavanagh
Chief Geologist - Exploration.

PMK:sw

Section No.	Hole No.	Footage	Length	Width	Thickness	Ag.oz	Pb%	Zn%	Volume Cu.Ft.	Vol.x Ag	Vol. x Pb	Vol. x Zn
106+50SE	A-24	151.7 - 200.5	200'	200	40	1.8	3.7	4.8	1,600,000	2,880,000	5,920,000	7,680,000
108+50SE	A-20	160.0 - 187.0	175	230	25	1.3	3.1	4.9	1,010,000	1,313,000	3,131,000	4,949,000
		328.6 - 360.5	175	250	30	0.9	2.1	4.6	1,310,000	1,179,000	2,751,000	6,026,000
	A-10	356.2 - 432.0	175	220	60	1.7	2.3	5.6	2,320,000	3,944,000	5,336,000	12,992,000
	A-18	465.0 - 483.0	175	200	15	0.6	2.2	5.0	530,000	318,000	1,166,000	2,650,000
110 SE	A-31	140.0 - 160.0	175	150	20	1.3	3.2	3.7	530,000	689,000	1,696,000	1,961,000
		175.0 - 195.0	175	150	20	1.6	3.3	4.5	520,000	832,000	1,716,000	2,340,000
		232.5 - 242.5	175	150	10	1.8	3.2	5.8	260,000	468,000	832,000	1,508,000
	A-30	406.0 - 483.0	175	210	70	1.2	3.4	4.0	2,570,000	3,084,000	8,738,000	10,280,000
112 SE	A-19	83.0 - 113.0	200	200	30	1.6	4.7	4.0	1,200,000	1,920,000	5,640,000	4,800,000
		210.0 - 230.0	200	200	20	1.1	3.2	4.9	800,000	880,000	2,560,000	3,920,000
	A-15	334.0 - 352.9	200	200	19	0.7	2.3	4.5	760,000	532,000	1,748,000	3,420,000
114 SE	A-12	320.5 - 399.0	200	200	79	1.2	3.8	4.0	3,160,000	3,792,000	12,008,000	12,640,000
116 SE	A-23	125.0 - 175.0	200	160	80	1.5	4.1	4.3	2,560,000	3,840,000	10,496,000	11,008,000
	A-35	406.0 - 421.0	200	200	10	1.2	3.5	4.0	400,000	480,000	1,400,000	1,600,000
	A-6A	154.0 - 232.0	200	200	78	2.5	7.8	7.2	3,120,000	7,800,000	24,336,000	22,464,000
	A-14	423.3 - 437.3	200	220	10	1.5	3.5	4.3	440,000	660,000	1,540,000	1,892,000
118 SE	A-28	200.0 - 252.0	200	190	50	1.7	3.3	5.6	1,900,000	3,230,000	6,270,000	10,640,000
	A-4	193.0 - 237.0	200	190	43	1.1	3.5	4.2	1,630,000	1,793,000	5,705,000	6,846,000
		279.0 - 314.0	200	190	35	1.5	1.0	5.2	1,330,000	1,995,000	1,330,000	6,916,000
		375.0 - 395.0	200	170	20	1.1	4.5	4.3	680,000	748,000	3,060,000	2,924,000
		425.0 - 472.0	200	170	47	1.6	2.9	4.2	1,600,000	2,560,000	4,640,000	6,720,000
120 SE	A-5	276.0 - 320.0	200	190	43	2.1	5.6	4.6	1,600,000	3,360,000	8,960,000	7,360,000
		356.4 - 367.8	200	180	10	1.4	5.1	4.1	360,000	504,000	1,836,000	1,476,000
	A-13	472.0 - 482.0	200	190	10	1.5	3.5	5.0	380,000	570,000	1,330,000	1,900,000
122 SE	A-29	130.5 - 181.0	200	140	45	1.1	4.9	3.5	1,260,000	1,386,000	6,174,000	4,410,000
		205.0 - 230.7	200	150	20	1.7	3.6	5.4	600,000	1,020,000	2,160,000	3,240,000
	A-25	179.7 - 188.0	200	150	9	1.8	3.6	4.7	270,000	486,000	972,000	1,269,000
	A-16	452.7 - 464.5	200	200	12	1.1	3.0	4.5	480,000	528,000	1,440,000	2,160,000
124 SE	A-37	73.0 - 113.0	200	200	40	1.2	3.0	assumed 4.0	1,600,000	1,920,000	4,800,000	6,400,000
		402.0 - 432.0	200	200	30	1.8	4.2	3.6	1,200,000	2,160,000	5,040,000	4,320,000
									37,980,000 Cu.Ft.			

TOTALS

VOL.	VOL. X AG.	VOL. X PB	VOL. X ZN.
37,980,000	56,871,000	144,731,000	178,711,000.
	1.5	3.8	4.7
	1.497	3.81	4.705

$8 \overline{)\,37,980,000}$
4,747,500

TOTAL WIDTH
1,015'
DOUBLE CHECKED
RCM.

A-4	90
A-5	65
A-6A	90
A-9	10
A-10	70
A-12	80
A-13	20
A-14	25
A-15	10
A-16	15
A-18	20
A-19	50
A-20	35
A-23	75
A-24	50
A-28	60
A-29	95
A-30	65
A-31	30
A-33	30
A-35	30
A-37	50

$$\frac{1065 \times 40{,}000}{8} = 1065 \times 5{,}000 = 5{,}325{,}000$$

1015 x 5

5,000,000 Tons 1.5 Ag, 4% Pb, 5% Zn.

				WIDTH x ASSAY.		
FOOTAGE	AG	PB	ZN.	AG	PB	ZN.
44.0'	1.1	3.5	4.2	48.4	154.0	184.8
20.0'	1.1	4.5	4.3.	22.0	90.0	86.0.
25.0'	1.8	4.3	4.1	45.0	107.5	102.5.
44.0'	2.1	5.6	4.6.	92.4	246.4	202.4.
11.0'	1.4	5.1	4.1	15.4	56.1	45.1.
7.0'	1.7	3.3	6.0.	11.9	23.1	42.0.
78.0'	2.5	7.8	7.2.	195.0	608.4	561.6.
10.0'	1.5	5.5	5.5	15.0	55.0.	55.0.
5.0'	1.1	3.7	5.1	5.5	18.5	25.5.
5.0'	0.6	2.9	5.5	3.0.	14.5	27.5.
70.0'	1.7	2.3	5.6.	119.0	161.0.	392.0.
80.0'	1.2	3.8	4.0.	96.0.	224.0.	320.0.
10.0'	1.5	3.5	5.0.	15.0.	35.0.	50.0.
5.0'	1.6	4.6	5.2	8.0.	23.0.	26.0.
14.0'	1.5	3.5	4.3	21.0.	49.0.	60.2.
10.0'	2.0	3.7	5.3.	20.0	37.0.	53.0.
10.0'	1.3	3.6	6.3	13.0.	36.0.	63.0.
12.0'	1.1	3.0	4.5	13.2	36.0.	54.0.
18.0'	0.6	2.2	5.0.	10.8	39.6.	90.0.
30.0'	1.6	4.7	4.0	48.0.	141.0.	120.0.
20.0'	1.1	3.2	4.9	22.0.	64.0.	98.0.
27.0'	1.3	3.1	4.9	35.1	83.7.	132.3.
5.0'	1.0	2.3	8.6.	5.0	11.5.	43.0.
75.0'	1.5	4.1	4.3.	112.5	307.5	322.5
50.0'	1.8	3.7	4.8.	90.0.	185.0.	240.0.
52.0'	1.7	3.3	5.6.	88.4.	174.9	296.8.
5.0'	0.8	2.1	6.0	4.0.	10.5	30.0.

WIDTH X ASSAY

WIDTH	AG	PB	ZN	AG	PB	ZN
50.0'	1.1	4.9	3.5	55.0	245.0	175.0
26.0'	1.7	3.6	5.4	44.2	93.6	140.4
5.0'	1.0	3.4	8.1	5.0	17.0	40.5
10.0'	2.1	3.7	5.1	21.0	37.0	51.0
35.0'	1.4	4.4	5.3	49.0	154.0	185.5
27.0'	1.2	3.8	4.4	32.4	102.6	118.8
20.0'	1.6	3.3	4.5	32.0	66.0	90.0
10.0'	1.8	3.2	5.8	18.0	32.0	58.0
10.0'	1.9	5.8	7.4	19.0	58.0	74.0
5.0'	1.4	2.5	5.5	7.0	12.5	27.5
15.0'	1.2	3.5	4.0	18.0	52.5	60.0
30.0'	1.2	3.0	4.0	36.0	90.0	120.0
30.0'	1.8	4.2	3.6	54.0	126.0	108.0
40/1015.				1565.2	4078.4	4971.9
25.4				1.54	4.02	4.90

Swim Lakes "A" ORE Reserves 4,750,000 Tons. Ag - 1.5 oz Pb - 3.8% Zn - 4.7%

Section No.	Hole No.	Footage	Length	Width	Thickness	Ag.oz	Pb%	Zn%	Volume Cu.Ft.	Vol.x Ag	Vol. x Pb	Vol. x Zn
106+50SE	A-24	151.7 - 200.5	200'	200	40	1.8	3.7	4.8	1,600,000	2,880,000	5,920,000	7,680,000
108+50SE	A-20	160.0 - 187.0	175	230	25	1.3	3.1	4.9	1,010,000	1,313,000	3,131,000	4,949,000
		328.6 - 360.5	175	250	30	0.9	2.1	4.6	1,310,000	1,179,000	2,751,000	6,026,000
	A-10	356.2 - 432.0	175	220	60	1.7	2.3	5.6	2,320,000	3,944,000	5,336,000	12,992,000
	A-18	465.0 - 483.0	175	200	15	0.6	2.2	5.0	530,000	318,000	1,166,000	2,650,000
110 SE	A-31	140.0 - 160.0	175	150	20	1.3	3.2	3.7	530,000	689,000	1,696,000	1,961,000
		175.0 - 195.0	175	150	20	1.6	3.3	4.5	520,000	832,000	1,716,000	2,340,000
		232.5 - 242.5	175	150	10	1.8	3.2	5.8	260,000	468,000	832,000	1,508,000
	A-30	406.0 - 483.0	175	210	70	1.2	3.4	4.0	2,570,000	3,084,000	8,738,000	10,280,000
112 SE	A-19	83.0 - 113.0	200	200	30	1.6	4.7	4.0	1,200,000	1,920,000	5,640,000	4,800,000
		210.0 - 230.0	200	200	20	1.1	3.2	4.9	800,000	880,000	2,560,000	3,920,000
	A-15	334.0 - 352.9	200	200	19	0.7	2.3	4.5	760,000	532,000	1,748,000	3,420,000
114 SE	A-12	320.5 - 399.0	200	200	79	1.2	3.8	4.0	3,160,000	3,792,000	12,008,000	12,640,000
116 SE	A-23	125.0 - 175.0	200	160	80	1.5	4.1	4.3	2,560,000	3,840,000	10,496,000	11,008,000
	A-35	406.0 - 421.0	200	200	10	1.2	3.5	4.0	400,000	480,000	1,400,000	1,600,000
	A-6A	154.0 - 232.0	200	200	78	2.5	7.8	7.2	3,120,000	7,800,000	24,336,000	22,464,000
	A-14	423.3 - 437.3	200	220	10	1.5	3.5	4.3	440,000	660,000	1,540,000	1,892,000
118 SE	A-28	200.0 - 252.0	200	190	50	1.7	3.3	5.6	1,900,000	3,230,000	6,270,000	10,640,000
	A-4	193.0 - 237.0	200	190	43	1.1	3.5	4.2	1,630,000	1,793,000	5,705,000	6,846,000
		279.0 - 314.0	200	190	35	1.5	1.0	5.2	1,330,000	1,995,000	1,330,000	6,916,000
		375.0 - 395.0	200	170	20	1.1	4.5	4.3	680,000	748,000	3,060,000	2,924,000
		425.0 - 472.0	200	170	47	1.6	2.9	4.2	1,600,000	2,560,000	4,640,000	6,720,000
120 SE	A-5	276.0 - 320.0	200	190	43	2.1	5.6	4.6	1,600,000	3,360,000	8,960,000	7,360,000
		356.4 - 367.8	200	180	10	1.4	5.1	4.1	360,000	504,000	1,836,000	1,476,000
	A-13	472.0 - 482.0	200	190	10	1.5	3.5	5.0	380,000	570,000	1,330,000	1,900,000
122 SE	A-29	130.5 - 181.0	200	140	45	1.1	4.9	3.5	1,260,000	1,386,000	6,174,000	4,410,000
		205.0 - 230.7	200	150	20	1.7	3.6	5.4	600,000	1,020,000	2,160,000	3,240,000
	A-25	179.7 - 188.0	200	150	9	1.8	3.6	4.7	270,000	486,000	972,000	1,269,000
	A-16	452.7 - 464.5	200	200	12	1.1	3.0	4.5	480,000	528,000	1,440,000	2,160,000
124 SE	A-37	73.0 - 113.0	200	200	40	1.2	3.0	assumed 4.0	1,600,000	1,920,000	4,800,000	6,400,000
		402.0 - 432.0 95417	200	200	30	1.8	4.2	3.6	1,200,000	2,160,000	5,040,000	4,320,000

37,980,000 Cu.Ft.