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INDEX - METALLURGY

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2.0 SUMMARY

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Metallurgical test data from laboratory and pilot plant investigations was assembled and carefully analyzed. Following detailed analysis of all data, and based on the design performance of the modified concentrator and experience of northern operations, plant metallurgy for each ore type was predicted.

TABLE 1

PREDICTED METALLURGY BY ORE BODY

ORE SOURCE	CONC.	METALLURGY									
		ASSAYS						DISTRIBUTION			
		Pb	Zn	Au	Ag	Hg	As	Pb	Zn	Au	Ag
Faro-Zone III	Lead	67	*	0.75	600	40	0.03	87.5	*	33	65
	Zinc	*	53.5	*	*	300	0.01	*	88.5	*	*
Grum - Open Pit	Lead	60	*	3.5	750	90	0.10	80	*	33	65
	Zinc	*	55	*	*	650	0.05	*	83	*	*
Vangorda - Open Pit	Lead	50	*	6.5	575	60	0.25	80	*	35	55
	Zinc	*	52.8	*	*	300	0.10	*	77	*	*

Notes: a) Silver, Gold and Mercury assays in terms of g/tonne.

b) Data refers to predicted metallurgy at rated tonnages with design feed grades.

2.1 INTRODUCTION

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The determination of the metallurgical response of each of the three ore types was a long and complex process. Clearly the response of an ore is governed by two principal factors - mineralogical considerations and also treatment conditions.

Fortunately detailed mineralogical studies by the Exploration and Geological groups had revealed that each of the three deposits was made up of varying proportions of three major mineralogical components. These components were designated Type G indicating a baryte rich ore, Type E representing a pyritic specie and Type A which covered a range of mineralized quartzites with variable graphite contents.

Armed with this information, testwork proceeded with efforts being directed toward the determination of the metallurgical response of each of the three major mineralogical types. Then by using the geological model for each deposit, the proportion of each type present was determined, and hence the mean metallurgical response calculated.

As laboratory and pilot plant testwork proceeded it became apparent that certain treatment factors for optimum metallurgy were common to all ore types. A fine primary grind of at least 50 microns

P_{80} appeared quite critical and there were very strong indications that extremely fine regrinding was also necessary to the attainment of acceptable concentrate grades.

In the following section the derivation of metallurgical response by ore deposits using the technique outlined above is detailed. Some of the data requires further verification by means of additional testwork and other data, notably that from the Grum deposit is currently being verified by additional laboratory testwork on new samples of ore.

2.2 FARO ZONE III METALLURGY

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2.21 Preliminary Laboratory Testwork

Interest in the effects of finer primary grinding on Anvil metallurgy was stimulated by testwork performed at Faro during the fall of 1978 on a pyrrhotitic ore type. The indications from this and subsequent work coincided with data from other sources and suggested that optimum metallurgy for most Faro ore types would be obtained by primary grinding to at least the 50 micron P_{80} range. Table 2 shows the sources of data relating to work to determine optimum grind for Faro ore types.

TABLE 2

PRIMARY GRIND LEVEL FOR OPTIMUM METALLURGY

DATA SOURCE	TYPE OF TESTS	NO.OF TESTS	OPTIMUM GRIND P_{80}	OPTIMUM GRIND %-325#
Cyprus Anvil Test Laboratory	Rougher & Cleaner Tests	100	40-45	85
Kamloops Research Laboratory	Rougher & Cleaner Tests	120	45-50	80
Sachtleben - FDR	Cleaner Testing	30	52	78
Mitsui Mining & Smelting	Rougher & Cleaner Tests	30	37-70	80-93

Notes: a) Numbers of tests are estimates only.

b) Japanese work identified pyrrhotitic ore as benefiting most from fine grinding.

c) Tests are all laboratory scale work on 1.0 or 2.0 kg samples.

Detailed laboratory testwork by ore type was then planned and carried out at the Kamloops Research Laboratory in order to ensure that these preliminary results were representative of the majority of ore types (Ref. Report KM008 - June 10th 1979). Parallel work at the Cyprus Anvil laboratory on a different suite of samples, but encompassing the same test methods yielded the similar results. All data from this latter phase of the program is summarized in the Table 3 below; For completeness, the pilot plant data, described in detail later, is also shown.

TABLE 3

METALLURGICAL IMPROVEMENTS WITH FINER GRINDING

DATA SOURCE	GRIND RANGE INVESTIGATED (P_{80} microns)	GRIND RANGE INVESTIGATED (%-325#)	METALLURGICAL IMPROVEMENTS ACROSS RANGE LEAD GRADE	LEAD RECOVERY	ZINC GRADE	ZINC RECOVERY
Rougher Tests KRAL-78	135 - 40	50 - 85	7.0	*	2.50	*
Rougher Tests CAMC 78-79	125 - 37	45 - 90	11.3	*	5.9	*
Cleaner Tests KRAL-79	125 - 45	45 - 80	2.5	7.0	2.1	9.5
Cleaner Tests CAMC-79	125 - 40	45 - 85	3.2	5.2	4.4	4.0
Plant Test Nov.13-17/78	125 - 60	45 - 70	3.0	5.8	3.0	4.0
Cleaner Tests KRAL-79	145 - 30	30 - 90	*	9.6	*	12.0
Lakefield Pilot Plant I-79	130 - 45	40 - 85	5.9	3.6	3.6	5.9
Lakefield Pilot Plant II-79	140 - 55	35 - 80	4.5	1.0	4.5	4.5

Notes: a) * signifies that results were adjusted to a constant grade or recovery for the purposes of analysis of data.

b) Pilot plant data refers to the two separate samples tested.

c) CAMC - Cyprus Anvil Mining Corporation - Faro
KRAL - Kamloops Research and Assay - Kamloops
Lakefield - Lakefield Research Laboratory - Ontario

2.22 Pilot Plant Test Program

In order to verify the results of the laboratory test program, a pilot plant investigation was planned and executed. The work was performed at the testing facility of Lakefield Research of Canada, Limited, Lakefield, Ontario, during September and October of 1979.

During the course of the investigation at Lakefield, two bulk samples were subjected to a detailed investigation. The technical aspects of the work are described in Lakefield Reports LR 2202 Volumes I-IV. Supervision and direction of the testing was the responsibility of W. Muir, Plant Metallurgist, Cyprus Anvil Mining Corporation and P.J. Brown, Consulting Metallurgist, Met Engineers Ltd. At all times during the program, the senior Lakefield metallurgist, Serge Bulatovic provided the indespensible link between client and pilot plant operating personnel.

The work was performed using the standard reagent pattern currently employed at Anvil and with a flotation circuit designed to approximate to that proposed in the modified circuit at Anvil. Grinding and regrinding effects were the principle parameters investigated in the program. The program had two major objectives which were:

(a) The determination of the optimum economic grind level.

(b) Estimation of plant metallurgy at this optimum point.

2.23 Determination of the Optimum Grind

Using the results of the pilot plant program at various grind levels and by estimating the capital expenditure needed to achieve the various grind levels, it became possible to estimate optimum economic grind level. This calculation was performed by using the metallurgical data generated by the pilot plant work, a capital expenditure estimate generated by Kilborn Engineering Ltd. for various grinding configurations and an operating unit cost increment. The results of this study are summarized in table 4 below, and confirm that the economic and the metallurgical optima are near coincident at a P_{80} of about 50 microns.

TABLE 4

OPTIMUM ECONOMIC GRIND LEVEL

CASE	PRIMARY H.P.	REGRIND H.P.	TOTAL H.P.	GRIND TARGET P_{80}	CAPITAL COST $\$x10^6$	OPERATING COST INCREASE $\$x10^6$	RELATIVE NET PRESENT VALUE $\$x10^6$
1	11,700	1,350	13,050	50	43.0	6.8	153.9
2	9,200	2,350	11,550	70	43.1	5.1	129.8
3	7,700	1,850	10,550	100	38.6	4.3	112.7
4	5,200	1,350	6,550	130	0	0	117.4

Notes: a) Reference - Memo Brown to Taggart Dec. 16, 1979 "Optimum Grind Calculations".

b) Capital cost includes power plant: Clearly case 4 is with no expansion.

c) Operating cost includes manpower, steel and reagents.

d) Data assumes milling a 50:50 blend of Grum and Anvil Zone III ore.

In reviewing the data in table 4, it is important to note that for the four alternatives considered that the difference in capital and installed power is very small indeed. The difference in predicted plant metallurgy for each case is significant.

The calculation of Relative Net Present Value was achieved by making certain assumptions about costs and prices: The values computed are appropriate for comparative purposes but may not be used as absolute values.

2.24 Predicted Metallurgy at Optimum Grind

Pilot plant data at the economic optimum grind level of 50 microns was averaged for each of the two bulk samples tested and the results used as a basis to predict the plant metallurgy at the optimum grind.

Both ore samples treated were taken from the Faro Zone I pit and were selected by the Engineering department of Cyprus Anvil as being representative of the ore type known as 4E. Type 4E is a predominately pyritic specie which comprises the bulk of the ore in the Zone III ore body. Since both samples suffered moderate to severe en route oxidation, some of the minerals were rendered non flotable - this was especially evident in sample No. 1. Shown below in Table 5 are the sulphide and non-sulphide contents of the two samples illustrating the degree of sample oxidation.

TABLE 5

PILOT PLANT BULK SAMPLES - HEAD ASSAYS

SAMPLE	ASSAYS %			
	Pb	Pb Ox.	Zn	Zn Ox.
Bulk Sample No.1	2.35	0.40	3.85	0.25
Bulk Sample No.2	2.40	0.35	4.34	0.22

Note: "Oxide components" encompass all non-sulphides including carbonates, hydroxides, etc.

Because of the oxidized condition of the samples, corrections were made to recorded data to compensate for the anamolous oxide contents. The corrections were based on experience, and established practice which required that a certain fraction of the reported oxide be removed from the head assay in the recovery calculations. These corrections had the effect of increasing the reported pilot plant lead recovery substantially; zinc metallurgy was only marginally influenced.

TABLE 6

CALCULATED SULPHIDE HEAD ASSAYS

SAMPLE	ASSAYS % LEAD ASSAY HEAD	ASSAY OXIDE	CALCULATED EFFECTIVE HEAD	ZINC ASSAY HEAD	ASSAY OXIDE	CALCULATED EFFECTIVE HEAD
Bulk Sample No. 1	2.35	0.40	2.05	3.85	0.25	3.72
Bulk Sample No.2	2.40	0.35	2.14	4.34	0.22	4.23

In keeping with conservative methods of predicting data used elsewhere in this study, it was assumed that, of the oxidized lead and zinc, only one quarter would be recoverable; (Table 6) Thus the calculated effective lead assays were derived. The results of the pilot plant data modified in this manner are shown below in table 7 and 8.

TABLE 7

LAKEFIELD PILOT PLANT TEST RESULTS

PILOT PLANT SAMPLE NO.1

50 MICRON GRIND

TESTS 9, 10 and 24

F Tail.
Pb Zn

RECOVERY

	WEIGHT %	ASSAYS Pb	Zn	DISTRIBUTION Pb	Zn
Feed	100.0	2.03	3.66	100.0	100.0
Lead Conc.	2.58	68.4	2.6	86.9	1.8
Zinc Conc.	6.35	0.38	52.1	1.2	90.4
Tails	91.71	0.27	0.31	11.9	7.8

Notes: a) See notes under table no. 8.

TABLE 8

PILOT PLANT SAMPLE NO.2

50 MICRON GRIND

TESTS 32, 33, 34 and 35

	WEIGHT %	ASSAYS Pb	Zn	DISTRIBUTION Pb	Zn
Feed	100.0	2.12	4.26	100.0	100.0
Lead Conc.	2.62	73.6	2.6	91.0	1.6
Zinc Conc.	6.99	0.48	55.2	1.6	90.6
Tails	90.39	0.17	0.37	7.4	7.8

Notes: a) Head assays were determined from the pilot plant grinding circuit overflow with a correction made for non-recoverable oxide species.

b) Weight percentages taken from assay calculations averaged for the pilot plant runs noted. Checked also by three product formula method.

c) Concentrate and minor element determined by assaying pilot plant composites.

In studying the overall results of the two test series two factors were considered significant:

i) The sample feed grades were quite low compared to predicted mill feed for Zone III: It would be quite reasonable to expect better results with the higher feed grade.

TABLE 9

COMPARISON OF HEAD ASSAYS

SAMPLE	HEAD ASSAYS Pb	Zn
Bulk Sample No. 1	2.35	3.85
Bulk Sample No. 2	2.40	3.85
Zone III Average Estimated	2.90	4.50

Note: a) Zone III data is from latest mine model and does not include phase 6 ore.

ii) The results from sample No.2 were considered to be quite exceptional. There were several indications to support this view, especially the high zinc concentrate grades which suggested a somewhat atypical zinc mineral structure, the rapid rate of flotation of the minerals, and the results from the Faro plant when treating similar material. Also the sample exhibited most unusual precious metal distributions.

The pilot plant data was analyzed and a conservative metallurgical balance constructed for actual plant operation. For comparison are shown some locked cycle test results generated on samples of Zone III drill core. (Table 10)

TABLE 10

PLANT METALLURGICAL BALANCE

DATA SOURCE	CONCENTRATE	ASSAYS Pb	ASSAYS Zn	DISTRIBUTION Pb	DISTRIBUTION Zn
Pilot Plant Work Sample No.1	Lead	68.4		86.9	
	Zinc		52.1		90.4
Pilot Plant Work Sample No.2	Lead	73.6		91.0	
	Zinc		55.2		90.6
Locked Cycle Tests	Lead	77.2		88.8	
	Zinc		53.8		83.5
Predicted Plant Results Faro Zone III	Lead	67.0		87.5	
	Zinc		53.5		88.5

Note: a) All data refers to metallurgy at a P_{80} of 50 microns.

b) Locked cycle tests performed at Cyprus Anvil during 1980 on samples of Zone III drill core. Data reported in January 1981 in a report by A. McIntyre.

2.25 Gold, Silver and Minor Elements

Based on assays of diamond drill composites, the Zone III ore was estimated to contain appreciable quantities of silver and minor amounts of gold. Averaged data indicated that the mean silver content would be 35 g/tonne Ag while gold would be about 0.065 g/tonne Au. The estimate of the predicted gold and silver metallurgy was conducted as follows:

i) Silver Metallurgy

Two pilot plant runs using sample No. 1 were assayed for silver; the results are shown below.

TABLE 11

SILVER RECOVERY AND GRADE AT 50 microns GRIND

TEST NO.	PRODUCT	WEIGHT %	ASSAYS %, g/tonne Pb	Zn	Ag	% DISTRIBUTION Pb	Zn	Ag
PP-9	Pb Cl. Conc. Zn Cl. Conc. Zn Comb. Tail.	2.58 6.56 90.86	69.1 0.43 0.59	2.4 52.5 0.41	501.13 38.32 6.62	75.8 1.2 23.0	1.6 88.8 9.6	60.3 11.7 28.0
	Cyclone O'Flow	100.00	2.35	3.88	21.46	100.00	100.00	100.00
PP-10	Pb Cl. Conc. Zn Cl. Conc. Zn Comb. Tail.	2.61 6.24 91.15	68.2 0.34 0.57	2.82 52.0 0.52	486.38 34.86 7.62	76.7 0.9 22.4	1.9 85.6 12.5	58.2 10.0 31.8
	Kason U'Size	100.00	2.32	3.79	21.82	100.00	100.00	100.00

Assuming that the silver recovery increases linearly with lead recovery as with most Faro area ore types, then increasing the lead recovery to 87.5% should result in some increase in silver recovery from the reported 58-60% level. Based on experience and the long term silver recovery data at Anvil, a silver recovery figure of 65% was selected.

Unfortunately the bulk samples were quite low in silver content - about 21.5 g/tonne Ag: correction for this was achieved by calculating the quantity of silver recovered into the lead concentrate at fixed recovery (e.g. 600 g/tonne silver in the lead concentrate) from a predicted mill feed of 34 g/tonne Ag.

ii) Gold Metallurgy

Some support for both the gold and silver predicted recovery levels were obtained by reassaying a locked cycle test on samples from Faro Zone III obtained in the 1978 drilling program. The data in Table 12 shows that a gold recovery of 35-40% may be anticipated in the lead concentrate. Using the predicted mine model feed grade and a gold recovery of 33% the gold concentration in the lead concentrate was calculated (Table 14).

TABLE 12

GOLD AND SILVER METALLURGY - LABORATORY TEST DATA

	ASSAYS (g/tonne) Au	Ag	DISTRIBUTION Au	Ag
Feed	0.17	44	100.0	100.0
Lead Concentrate	1.20	530	38.8	66.3
Zinc Concentrate	0.70	39	37.0	8.0

Notes: a) Composite obtained from a very limited drill program.

b) Head assays higher than mine model predicts.

c) Data from cycle test No.8 Ref.LR2082 using Faro Zone III ore.

iii) Mercury and Arsenic

Lacking specific assay data on Zone III lead and zinc concentrate contaminants, minor element assays for the year 1979 were averaged and used to predict mercury and arsenic concentrations. The averaged data for 1979 concentrations of minor elements is shown below in Table 13.

TABLE 13

MINOR ELEMENTS - FARO ZONE III

CONCENTRATE	ASSAYS g/tonne Hg	As
Lead Concentrate	40	300
Zinc Concentrate	300	100

2.26 Predicted Plant Metallurgy

TABLE 14

PREDICTED PLANT PERFORMANCE - ZONE III ORE

	WEIGHT	ASSAY				DISTRIBUTION			
		Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
Feed	100.0	2.9	4.6	35	*	100.0	100.0	100.0	100.0
Lead Conc.	3.79	67.0	3.0	600	0.75	87.5	2.5	65.0	33.0
Zinc Conc.	7.61	0.5	53.5	40	*	1.3	88.5	8.7	*
Tails	88.60	0.37	0.47	10	*	11.2	9.0	26.3	*

Note: a) The average gold content of the lead concentrates from Zone III will be below payable limits. However, due to the irregular occurrance of gold in the ore it is possible that payable gold will be encountered in some shipments.

2.3 GRUM METALLURGY

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2.31 Preliminary Laboratory Testwork

Metallurgical investigations of the Grum ore body by Kerr Addison Ltd. and Noranda Mines were preceded by extensive mineralogical studies in which the entire array of ore types, believed to number twelve, were individually characterized. Having identified the various ore types, metallurgical testwork commenced at Lakefield on individual samples of each ore type, and also on composites of ore types. This compositing of ore types was a relatively late step when it was realized that many of the so called ore types were in fact metallurgically indistinguishable. The concept of compositing samples for testing was very acceptable since the costs of individually testing each ore type were prodigious.

The testwork, which was performed exclusively at Lakefield Research Ltd., Lakefield, Ontario, commenced in 1976 and culminated in late 1977 with long series of pilot plant tests. The work was directed principally by Mr. K. Konigsmann, Chief Metallurgist, Noranda Mines, and by others in the Noranda Milling Committee. A very brief synopsis of the excellent work performed at Lakefield is given below.

2.32 Critical Results from Laboratory Testwork

The earliest work performed at Lakefield demonstrated the need for a fine primary grind on Grum ores in the range 50 microns P_{80}. This observation was strongly supported by the results of the meticulous microscopic studies of Dr. Carson of Noranda Mines, which indicated an unusually fine intergrowth of mineral crystals in most ore types.

Many laboratory test series were performed to illustrate the effect of variations of primary grind on metallurgy. Typical of the tests are the data shown below in tables 15 and 16; they refer to testwork on a composite made up of seven of the twelve ore types.

TABLE 15

EFFECT OF FINENESS OF PRIMARY GRIND

TEST NO.	TIME Min.	%-200 Mesh	PRODUCTS	WEIGHT %	ASSAY% Pb	Zn	% DISTRIBUTION Pb	Zn
153	30	87.1	Pb Cleaner Concentrate Pb Rougher Concentrate	3.64 17.93	65.9 15.3	3.69 7.54	78.2 89.8	2.2 22.2
179	25	79.2	Pb Cleaner Concentrate Pb Rougher Concentrate	3.30 19.21	63.3 14.0	4.04 7.65	70.0 89.9	2.2 24.1
180	20	68.1	Pb Cleaner Concentrate Pb Rougher Concentrate	3.01 18.81	65.9 14.0	3.89 7.66	67.1 88.8	2.0 24.2

Notes: a) Data Source L.R. 1991 Vol. 7.

b) Comparative data for the effect of grind on the zinc circuit not reported for the tests.

Having established that adequate mineral liberation could be obtained at a relatively fine primary grind, attention was then focused on improving the lead concentrate grade. In almost all samples the investigators found that an extremely fine regrind of the lead rougher concentrate was mandatory in order to achieve reasonable concentrate grades at acceptable recoveries. Typical of the results obtained are those shown below in Table 16. Data in this table refers to work on the pilot plant composite which encompassed components from most ore types. The effect of regrinding on lead final concentrate grade, at almost constant recovery, is obvious.

TABLE 16

EFFECT OF LEAD CONCENTRATE REGRIND ON LEAD CLEANING

TEST NO.	PRIMARY GRIND (MIN.)	Pb REGRIND TIME (MIN)	Pb REGRIND % PASS 10 u	PRODUCT	WEIGHT %	ASSAY % Pb	ASSAY % Zn	% DISTRIBUTION Pb	% DISTRIBUTION Zn
42	30	10	31.5	Pb Cleaner Concentrate Pb 1st Cleaner Conc. Pb Combined Tailing	12.12 18.63 81.37	38.6 27.3 0.85	12.9 13.7 9.19	81.0 88.0 12.0	15.6 25.5 74.5
43	30	20	37.2	Pb Cleaner Concentrate Pb 1st Cleaner Conc. Pb Combined Tailing	10.54 18.24 81.76	45.1 28.8 0.80	12.9 14.1 8.97	80.4 89.0 11.0	13.7 26.0 74.0
45	30	30	47.4	Pb Cleaner Concentrate Pb 1st Cleaner Conc. Pb Combined Tailing	8.75 16.08 83.92	51.6 32.4 0.87	11.3 14.0 9.14	76.1 87.7 12.3	9.9 22.6 77.4
46	30	40	58.0	Pb Cleaner Concentrate Pb 1st Cleaner Conc. Pb Combined Tailing	7.97 16.54 83.46	54.6 31.3 0.89	10.7 14.1 9.2	73.4 87.3 12.7	8.1 23.0 76.0
47	30	50	62.0	Pb Cleaner Concentrate Pb 1st Cleaner Conc. Pb Combined Tailing	7.96 14.56 85.44	59.7 35.8 0.86	9.53 13.6 9.32	80.0 87.7 12.3	7.6 19.9 80.1

Notes: a) Data source L.R. 1991 Vol. 10

b) Sample: Pilot plant composite.

The route to acceptable metallurgy with Grum ore types then was established; a fine primary grind (P_{80} 50-60 microns) followed by a very fine regrind of the lead rougher concentrate. (P_{80} 15-20 microns)

In the course of these detailed studies it was found that the lead regrind mill exerted a significant influence on the zinc metallurgy; probably due to the considerable quantity of zinc minerals reporting in the lead rougher concentrate. In general however, the zinc metallurgy did not pose a major problem. The Grum zinc mineral was observed to be quite low in interstitial iron and manganese and, apart from losses in the lead cleaner circuit products, zinc recovery was good. The laboratory results suggested that zinc concentrate grades in excess of 55% Zn could be fairly easily achieved provided that both the lead and zinc regrinding circuit were optimized.

TABLE 17

EFFECT OF ZINC REGRIND ON ZINC CLEANING

TEST No.	Zn CLEANER FEED %-20μ	%-10μ	PRODUCT	ASSAY Pb	Zn	DISTRIBUTION Pb	Zn
41,47	78	50	Zinc Cleaner Conc.	2.3	56.5	5.7	80.6
			Zinc Flotation Tail	1.2	1.6	16.8	12.6
42,43,44	39	22	Zinc Cleaner Conc.	2.5	51.0	6.3	73.5
			Zinc Flotation Tail	1.3	2.3	17.3	17.6

Note: a) Data source LR2027.

b) Data refers to laboratory batch cleaner tests on a composite of the major ore types.

2.33 Pilot Plant Test Program

As the laboratory testwork neared completion at Lakefield in mid 1977, underground operations at the Grum site were directed toward obtaining a bulk sample for pilot plant testwork. The development adit had by this time penetrated the main sulphide zone and the bulk sample was taken principally from this area. Unfortunately sample make-up was somewhat biased because of the limited availability of representative quantities of each specific ore type. The sample bias was reflected in the anomalously high metal contents of the pilot plant composite.

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TABLE 18

HEAD ASSAY - PILOT PLANT COMPOSITE

	ASSAYS %					ASSAYS g/tonne		
	Cu	Pb	Zn	Fe	As	Au	Ag	Hg
Sample	0.13	6.1	10.0	20.5	0.23	1.4	98	80

Despite the problems with the sample bias there was no alternative but to commence testwork and to attempt to compensate, or correct the test results for the high grade sample when predicting probable plant metallurgy.

To assist in the prediction of plant metallurgy and as a guide in the extrapolation of results several locked cycle tests were carried out on various composites of Grum ores by Lakefield Research.

TABLE 19

LOCKED CYCLE TEST DATA - GRUM COMPOSITES

TEST NO.	COMPOSITE NO.	PRODUCT	WEIGHT %	ASSAYS %		% DISTRIBUTION	
				Pb	Zn	Pb	Zn
207	1	Pb Cleaner Conc. Zn Cleaner Conc. Zn Flot.Tailing	4.77 9.18 86.05	66.7 0.58 0.95	4.56 57.40 1.43	78.4 1.3 20.3	3.2 78.5 18.3
		Head (Calc.)	100.0	4.06	6.71	100.0	100.0
217	1	Pb Cleaner Conc. Zn Cleaner Conc. Zn Flot. Tailing	6.12 8.91 84.07	62.2 0.40 0.40	5.67 59.70 1.49	91.0 0.9 8.1	5.0 76.7 18.3
		Head (Calc.)	100.0	4.18	6.93	100.0	100.0
230	3	Pb Cleaner Conc. Zn Cleaner Conc. Zn Flot. Tailing	5.82 13.75 80.43	62.5 0.39 1.02	6.16 57.80 1.25	80.6 1.2 18.2	3.9 85.4 10.7
		Head (Calc.)	100.0	4.51	9.31	100.0	100.0
237	2	Pb Cleaner Conc. Zn Cleaner Conc. Zn Flot. Tailing	9.12 14.31 76.57	59.7 0.56 0.41	7.47 55.7 0.78	93.3 1.4 5.3	7.4 86.2 6.4
		Head (Calc.)	100.0	5.84	9.25	100.0	100.0
241	2	Pb Cleaner Conc. Zn Cleaner Conc. Zn Flot. Tailing	9.06 14.49 75.87	59.1 0.46 0.44	7.24 56.6 0.57	91.4 1.2 5.7	6.9 86.6 4.5
		Head (Calc.)	100.0	5.86	9.53	100.0	100.0

Notes: a) The composites 1, 2, and 3 are described in compositional detail in L.R. 1991 Vol. 8.

b) Tests carried out at 50-60 micron grind.

After about thirty preliminary tests, the regrinding circuits were optimized, the reagent scheme balanced and the optimum primary grind established at between 50-55 microns P_{80}. In the course of the work, and subsequent to the preliminary tests, several interesting observations were made.

a) Initially it appeared that lead regrind was not quite as critical as the laboratory work had indicated. Presumably this was because much of the laboratory work was carried out with samples averaging 9–10% combined metal; the higher grade pilot plant samples would naturally require less regrinding.

b) The collector consumption in the pilot plant was considerably lower than had been observed in the laboratory. Again this could be related to the relatively coarse mineralization associated with the high grade sample.

c) The zinc circuit first cleaner pH was found to be critical. Even very small variations ($\pm$ 0.5 units at pH 11.5) exercised enormous effects on the zinc concentrate grade.

d) High cyanide additions, in total about 250 g/tonne were found to be critical to attainment of good metallurgy.

The pilot plant program was directed by K. Konigsmann of Noranda Mines who was aided by D.M. Wyslouzil of Lakefield Research. A total of 53 tests were performed using a flotation reagent scheme very similar to that employed at Anvil (and incidentally almost identical to that used in the Anvil Pilot Plant program). The results are shown below in Table 20.

TABLE 20

GRUM PILOT PLANT TEST DATA

TEST NO.	METALLURGY LEAD GRADE	LEAD RECOVERY	ZINC GRADE	ZINC RECOVERY
1	49.6	75.9	58.9	48.9
2	53.1	74.3	56.0	73.5
3	44.7	80.7	56.0	70.9
4	44.1	80.6	50.2	66.2
5	47.6	75.6	55.8	66.1
6	59.0	81.8	54.3	78.5
7	46.6	83.1	52.6	72.3
8	49.9	79.7	49.6	78.0
9	62.4	77.8	55.1	73.8
10	44.9	79.2	36.9	74.0
11	42.2	81.6	42.6	72.7
12	47.9	74.5	38.2	76.4
13	46.4	77.8	46.0	77.6
14	52.8	77.3	47.0	78.6
15	57.2	74.8	42.3	78.7
16	56.2	76.2	49.3	75.8
17	59.2	76.2	43.3	81.4
18	57.5	74.8	50.0	79.9
19	57.9	74.2	51.5	76.7
20	59.7	70.8	48.3	74.9
21	53.6	78.1	53.9	76.1
22	62.3	71.7	54.2	78.1
23	52.3	71.0	56.3	69.1
24	58.4	63.0	52.0	81.1
25	59.3	77.4	52.1	81.1
26	57.0	76.0	56.4	76.0
27	58.4	77.7	48.3	79.8
28	51.9	78.0	46.8	75.6
29	56.4	77.6	54.5	74.2
30	54.9	75.4	50.0	78.0
31	53.7	76.3	55.0	75.4
32	65.4	78.1	54.5	79.8
33	59.2	80.4	51.8	79.0
34	64.8	77.6	53.5	77.4
35	66.6	75.7	57.4	72.0
36	54.1	77.5	52.6	78.0
37	63.1	78.3	50.2	82.3
38	60.6	77.0	53.3	79.9
39	56.9	75.3	50.0	76.6
40	57.2	75.2	56.0	72.9
41	62.7	77.2	57.4	79.6
42	60.9	75.6	48.4	79.9
43	60.7	75.9	52.1	75.5
44	54.1	77.4	52.4	65.1
45	46.8	76.5	53.8	71.7
46	55.2	76.8	50.8	73.6
47	64.6	77.9	55.5	81.5
49	62.7	76.0	56.5	78.3
50	60.8	78.1	53.6	80.6
51	65.8	74.4	52.6	82.6
52	58.3	77.5	54.0	78.0
53	60.9	75.5	54.7	78.9

Notes: a) Data source progress report No.11 Vol. I-IV L.R. 2027.

b) All results from tests using a standard flowsheet and one composite ore sample.

2.34 Analysis of Pilot Plant Results

The data from the tests in the latter part of the program was examined in minute detail by the Noranda Milling Committee. Noting that the sample tested comprised principally of massive sulphide ore types, the committee elected to produce two metallurgical balances: One representing the massive sulphide ore metallurgy while the other indicated their estimate of the overall average Grum metallurgy. This latter estimate was based on the results of laboratory cleaner and the locked cycle tests on various samples.

The committee were of the opinion that the true average metallurgy of all Grum ores would be somewhat better than for the sulphide zone ores. Accordingly they modified their best pilot plant using the results of the locked cycle tests to reflect this belief and increased both lead and zinc recoveries with the same concentrate grades. The two balances are shown below in Table 21.

TABLE 21

PREDICTED METALLURGY FROM PILOT PLANT RESULTS

ORE TYPE	CONCENTRATE	ASSAYS				DISTRIBUTION			
		Pb	Zn	Au	Ag	Pb	Zn	Au	Ag
Massive Sulphide	Lead	62	10	4.8	925	77	*	33	72
	Zinc	2.5	56	*	*	*	81	*	*
Average	Lead	62	8	5.1	950	80	*	33	72
	Zinc	2.0	56	*	*	*	84	*	*

Notes: a) Data source - Noranda Milling Committee Report Dec. 1977.

b) Au and Ag in g/tonne.

2.35 Gold, Silver and Minor Elements

Some assays on three pilot plant runs yielded some interesting results on precious metal concentrations and distributions. Concentrates collected during various pilot plant runs were assayed for mercury and arsenic. The results are discussed below in section 2.36.

TABLE 22

GOLD & SILVER METALLURGY - GRUM PILOT PLANT

TEST NO.	PRODUCT	WEIGHT %	ASSAYS (g/tonne) Au	Ag	% DISTRIBUTION Au	Ag
PP25	Pb Cleaner Concentrate	8.02	3.40	857	48.8	72.5
	Zn Cleaner Concentrate	16.04	0.15	73	4.9	12.4
	Zn Combined Tailing	75.94	0.30	19	46.3	15.1
	FLOTATION FEED (calc)	100.00	0.68	95	100.0	100.0
PP35	Pb Cleaner Concentrate	6.77	4.46	950	50.6	74.8
	Zn Cleaner Concentrate	12.30	0.15	69	3.4	9.8
	Zn Combined Tailing	80.93	0.30	16	46.0	15.4
	FLOTATION FEED (calc.)	100.00	0.68	86	100.0	100.0
PP37	Pb Cleaner Concentrate	7.32	3.77	950	46.6	77.2
	Zn Cleaner Concentrate	16.72	0.30	74	9.8	13.8
	Zn Combined Tailing	75.96	0.30	11	43.6	9.0
	FLOTATION FEED (calc.)	100.00	0.68	90	100.0	100.0

Notes: a) Data source - Progress Report No.11 Vol.I L.R.2027. Gold & Silver assays reported only on these tests.

b) Each data set based on an 8 hour duration pilot plant run.

2.36 Predicted Plant Metallurgy

Because of the significant effect which mill feed grade exercises on metallurgy and since the pilot plant sample was known to be biased, there are reasons to believe that the Grum average metallurgy, predicted by the Noranda Milling Committee, may be optimistic. The principal reasons are as follows:

a) Lead grade was difficult to achieve in the pilot plant, and lead recovery was quite low in those tests in which concentrate grades in excess of 60% Pb were obtained.

b) Silver recovery at 72% probably reflects the anomalously high silver content in the pilot plant feed. Silver recovery into the lead concentrate at Cyprus Anvil averages somewhat less than 60% and is expected to reach 65% only at very high lead recoveries.

Based on actual experience of the operation of a metallurgical plant in the Faro area with the constraints of climate, limited manpower availability and variable ore types, a conservative approach would indicate that a somewhat more pessemistic balance be adopted. Accordingly the lead concentrate grade was reduced from 62% to 60% Pb, but the lead recovery was kept constant. Silver recovery was reduced to 65% to reflect lower silver grade expected in the diluted ore from the Grum deposit.

The gold in the mill feed was assumed to be proportional to the lead content; hence the diluted feed grade required that the pilot plant gold head assay be reduced by about 60%. A gold recovery figure of 33% was used in the calculations - identical to that assumed for Zone III ore.

TABLE 23

PREDICTED METALLURGY - GRUM ORE

CONCENTRATE	ASSAYS						DISTRIBUTION			
	Pb%	Zn%	Au	Ag	As	Hg	Pb%	Zn%	Au%	Ag%
Average Mill Feed-Lead Conc.	60	11	3.5	750	100	90	80	*	33	65
-Zinc Conc.	2.5	55	*	*	50	650		83		

Notes: a) Au, Ag, As, and Hg assays in terms of g/tonne.

b) Silver recovery recalculated on the basis of approximately 50 g/tonne Ag in the mill feed.

c) Average mill feed including dilution, will be about 9% combined lead and zinc metal.

2.4 VANGORDA METALLURGY

2.4 VANGORDA METALLURGY

2.41 Preliminary Laboratory Work

Since its discovery in the mid fifties the Vangorda ore body remained, until quite recently a metallurgical enigma. Early metallurgical work produced erratic results and even the most optimistic metallurgist could predict only the production of a bulk lead-zinc concentrate. Probably the reason for the very poor initial metallurgical results was that the early diamond drill core was of small diameter and hence core recovery was poor, incidentally negating the chance of an accurate ore body model, and ensuring the rapid oxidation of the minerals, thus preventing the generation of reproducible laboratory test data.

Redrilling in the late sixties, and recognition of at least three ore types by the various investigators led to the generation of a few groups of reasonably encouraging metallurgical results. After about five years of sporadic testwork in many laboratories, there emerged a picture of an ore which depicted an extremely fine mineral crystal intergrowth, very markedly activated iron and zinc minerals, and a prediliction for rapid and deleterious mineral oxidation.

The Table 24 below is a synopsis of the early published work on Vangorda ore – unfortunately very few of the investigators noted the ore type being tested. The testwork covers a wide range of metallurgical conditions varying from lime to soda ash modulated circuits, various collectors and depressants and several cleaning schemes. All work exhibited one common factor however – Vangorda ores required an extremely fine primary grind of the order 30–50 μ P_{80} to permit any sort of separation to be achieved.

TABLE 24

PRELIMINARY LABORATORY TEST DATA - VANGORDA ORE

REF.	FEED		LEAD CONC.		ZINC CONC.		TAILINGS		GRIND	SOURCE
	% Pb	% Zn	GRADE	REC.	GRADE	REC.	% Pb	% Zn	MESH #	
1	4.0	4.9	49.9	77.0	49.4	60.4	0.15	0.66	99%-200	Dowa Mining Company Report. March 25, 1969.
2	4.6	4.7	56.6	83.8	54.4	79.7	0.63	0.54	64%-325	Galligher Company Report. July 17,1969. Series 2 tests.
3	4.1	5.2	51.7	89.7	55.4	78.1	0.38	0.65	82%-325	Brunswick Mining & Smelting Report. 1969
4	3.2	5.2	46.1	77.6	53.0	49.5	-	-	-	Brunswick Mining & Smelting attachment "Refractory Ores"
5	4.0	4.9	49.8	77.0	49.4	60.4	-	-	-	Brunswick Mining & Smelting attachment "Refractory Ores"
6	3.9	5.1	56.5	72.5	50.8	51.5	-	-	-	Brunswick Mining & Smelting attachment "Refractory Ores"
7	4.4	5.0	22.7	85.2	24.2	60.0	0.52	0.44	82%-325	Noranda Report "Vangorda & Brunswick M & S Ore Samples" December 9-13, 1969
8	5.8	6.1	29.1	85.4	11.0	60.1	0.54	1.22	80%-325	Noranda Report "Preliminary Test work on Vangorda Cores" April 2, 1970.
9	3.4	3.7	22.3	75.6	12.5	64.2	0.58	0.70	80%-325	Noranda Report "Preliminary Test work on Vangorda Cores" April 2, 1970.
10	1.6	2.7	37.4	77.9	49.2	77.4	0.34	0.36	55%-325	Noranda Report "Preliminary Locked Tests" Feb. 26, 1975
11	3.6	4.1	53.6	77.9	51.8	77.3	0.64	0.63	63%-325	Noranda Report "Preliminary Locked Tests" Feb. 26, 1975
12	3.7	6.8	48.7	78.4	54.4	80.4	0.51	0.74	77%-325	Noranda Report "Preliminary Locked Tests" Feb. 26, 1975
13	6.2	10.5	25.0	93.2	48.6	64.6	0.34	0.72	72%-325	Noranda Report "Preliminary Locked Tests" Feb. 26, 1975
14	3.1	4.4	61.7	78.3	55.3	86.6	0.19	0.14	80%-325	Dowa Mining Company "Metallurgical Test of the Vangorda Ore" May 1975.

Notes: a) All data shown in this table was abstracted from laboratory reports. Various schemes were employed but the soda ash - cyanide reagent scheme was most favoured.

b) The importance of a fine primary grind and the need for regrinding was repeatedly stressed by the investigators.

2.42 Detailed Type Testing

In the Summer of 1979 the Vangorda deposit was redrilled by Cyprus Anvil with relatively large diameter NQ drill bits and the core subjected to a meticulous geological examination and logging. The core was divided into three main geological species, type 4G, 4E and 4A, and then subdivided again according to assay range of lead and zinc.

Each group of samples was then subjected to detailed metallurgical testwork at the Kamloops Research Laboratory: The tests being designed to determine the effect of primary grind and regrinding on metallurgy when the particular ore type was treated with the Anvil reagent scheme. During the period June to November a total of 55 open circuit cleaner tests were completed on the three major ore types under study.

The results, are shown in summary form below in Table 25 as generated and also in a corrected form based on established laboratory procedures for the evaluation of metallurgical test data. The corrections involve the redistribution of the cleaner tailings to one or other of the concentrates according to experience.

Of special note are the extreme finess of grind employed in most tests ranging from 20-50 microns P_{80} in the primary grind. That these very fine grinds can be achieved is due to the extreme friability of the Vangorda ores.

Derivation of the metallurgy by ore type was fairly simple and was achieved by averaging the best corrected results obtained in the laboratory testwork. Almost invariably the best results occurred at very fine grind levels coupled with fine regrinding. Since the tests covered a wide range of test conditions for each major ore type, considerable attention was addressed to selection of the results for inclusion in each data set.

At no time during this phase of the testwork, nor in subsequent work performed in 1980, was a point detected beyond which fine grinding exercised a deleterious effect on metallurgy. These latter studies were conducted in the range 15-40 microns primary grind, with samples of various ore types (Ref. KM032 October 1980). As a point of general interest more and more complex sulphide operations are employing extremely fine primary grind levels. Meggen and Brunswick grind to 30-40 while at Huelva and Aznalcolla in Spain the primary grind is performed at less than 30 μ .

TABLE 25

VANGORDA METALLURGY BY ORE TYPE

TEST NO.	TEST METALLURGY LEAD GRADE	TEST METALLURGY LEAD RECOVERY	TEST METALLURGY ZINC GRADE	TEST METALLURGY ZINC RECOVERY	CORRECTED METALLURGY LEAD GRADE	CORRECTED METALLURGY LEAD RECOVERY	CORRECTED METALLURGY ZINC GRADE	CORRECTED METALLURGY ZINC RECOVERY	GRIND TIME (min.)	GRIND % 325 MESH	ORE TYPE
1	45.5	43.9	54.1	69.6	45.5	58.0	54.1	74.7	15	94	1A4G
2	36.4	88.4	55.7	66.0	36.4	90.3	55.7	68.6	15	97	1A4G
3	55.7	85.6	56.7	76.4	55.7	87.1	56.7	79.0	15	96	1A4G
4	54.6	87.2	54.9	76.2	54.6	89.2	54.9	78.7	15	98	1A4G
5	57.8	83.2	56.6	75.2	57.8	85.7	56.6	77.7	15	97	1A4G
6	40.3	84.2	55.1	63.3	40.3	86.7	55.1	67.4	10	86	1A4G
7	48.2	85.0	57.5	62.3	48.2	87.5	57.5	67.9	10	88	1A4G
8	60.8	77.5	54.2	61.8	60.8	81.3	54.2	69.3	10	89	1A4G
13	57.3	83.1	55.9	71.7	57.3	83.4	55.9	75.2	15	94	1A4G
14	61.4	79.7	55.9	77.9	61.4	81.4	55.9	80.6	15	94	1A4G
9	36.4	72.5	47.7	63.3	36.4	76.5	47.7	69.3	10	91	1B4G
10	45.9	76.2	51.4	72.3	45.9	79.2	51.4	75.3	15	94	1B4G
11	49.6	67.9	52.4	75.9	49.6	73.6	52.4	81.6	15	97	1B4G
12	41.8	73.6	54.1	71.5	41.8	77.2	54.1	74.5	15	95	1B4G
15	41.3	71.3	51.6	76.0	41.3	74.4	51.6	78.3	15	95	1B4G
16	50.0	77.4	55.0	78.9	50.0	80.2	55.0	81.6	25	99	1B4G
17	49.3	80.8	55.1	74.8	49.3	82.1	55.1	76.8	15	95	1B4G
18	53.6	69.6	56.8	46.5	53.6	72.6	56.8	61.5	15	93	1B4G
19	56.9	67.9	58.1	59.8	56.9	72.5	58.1	67.4	20	98	1B4G
34	43.3	78.2	53.0	57.6	43.2	81.2	53.0	64.5	20	97	2A4E
30	33.7	79.6	37.2	68.7	33.7	81.3	37.2	71.7	10	80	2B4E
31	39.9	77.7	41.2	67.5	39.9	80.7	41.2	72.5	15	91	2B4E
32	44.4	77.5	51.0	65.8	44.4	79.5	51.0	70.9	20	97	2B4E
33	29.7	74.6	40.6	59.9	29.7	77.6	40.6	65.0	5	57	2B4E
35	50.7	78.8	49.8	65.7	50.7	82.8	49.7	72.0	20	99	2B4E
20	37.9	51.9	44.9	17.7	37.9	56.3	44.9	43.2	15	93	2C4E
21	42.9	53.6	50.4	43.1	42.9	61.1	50.4	55.6	20	97	2C4E
22	45.2	74.2	52.3	69.5	45.2	76.3	52.3	74.3	20	99	2C4E
23	33.4	72.5	42.7	74.6	33.4	75.5	42.7	77.6	15	91	2C4E
24	33.7	73.3	44.6	63.2	33.4	76.3	44.6	69.2	15	92	2C4E
25	38.4	58.0	42.4	73.1	38.4	64.0	42.4	77.0	15	98	2C4E
36	34.2	59.9	45.8	57.3	34.2	65.9	45.8	67.3	10	84	2C4E
41	36.5	55.5	41.4	55.3	36.5	62.5	41.4	67.0	5	60	2C4E
37	23.3	66.9	32.1	50.4	23.3	72.9	32.1	59.4	5	49	2D4E
38	37.2	74.7	46.7	53.3	37.2	77.7	46.7	61.0	20	97	2D4E
39	37.7	65.6	45.3	57.8	37.7	68.6	45.3	63.8	15	86	2D4E
40	42.5	60.6	42.6	61.2	42.4	64.0	42.6	67.2	10	71	2D4E
42	38.9	76.3	46.8	69.9	38.9	76.3	46.8	75.9	5	37	3A4A
43	58.9	70.0	52.4	58.8	58.9	76.0	52.4	69.8	10	56	3A4A
44	67.2	62.6	49.9	76.2	67.2	70.6	49.9	82.2	15	76	3A4A
45	58.6	78.8	56.5	69.8	58.6	81.8	56.5	75.8	25	88	3A4A
46	48.8	80.5	51.5	77.2	48.8	83.5	51.5	81.2	25	88	3B4A
47	33.4	82.8	50.3	69.9	33.4	84.8	50.3	77.9	15	73	3B4A
48	31.9	82.8	44.8	72.2	31.9	83.8	44.8	75.2	10	54	3B4A
49	40.2	71.8	45.5	79.4	40.2	74.8	45.5	83.4	10	62	3C4A
50	36.9	75.0	46.3	76.0	36.9	79.0	46.3	80.0	15	74	3C4A
51	31.0	72.8	52.2	73.1	31.0	77.8	52.2	78.1	20	85	3C4A
56	30.6	74.2	52.1	74.2	30.6	79.2	52.1	79.2	25	89	3C4A
52	35.2	80.4	53.9	66.1	35.2	82.4	53.9	72.1	25	90	3D4A
53	27.2	72.7	45.5	69.8	27.2	74.7	45.5	74.8	10	59	3D4A
54	21.8	63.0	42.3	73.8	21.8	68.0	42.3	75.5	15	70	3D4A
55	20.9	75.4	48.6	66.6	20.9	79.3	48.6	72.5	20	87	3D4A

Notes: a) Test metallurgy is recovery at final cleaner concentrate grade.

b) Corrected metallurgy was arrived at by redistribution of the cleaner tailings.

c) Grind time refers to time in the laboratory test mill; Product P_{80} is proportional to grind time.

2.43 Gold, Silver and Minor Elements

In addition to the normal assays used for the computation of recoveries, the concentrates were assayed for silver, gold, mercury and arsenic. This data for selected tests is shown below in table 26 which shows average gold and silver concentrations and estimated recoveries. Averaged mercury and arsenic concentrations are reported in Tables 27 and 28 by type and for the Vangorda orebody.

TABLE 26

GOLD AND SILVER METALLURGY - VANGORDA ORE TYPES

ORE TYPE	TEST NO.	RECOVERY GOLD	RECOVERY SILVER	GRADE GOLD	GRADE SILVER
1A4G	3	54.6	70.6	4.8	692
1A4G	4	47.4	71.7	3.8	641
1A4G	5	42.2	63.3	4.1	685
AVERAGE	-	48.1	68.5	4.2	673
1B4G	17	28.9	62.7	7.5	584
1B4G	16	33.3	70.9	8.9	679
1B4G	11	23.5	56.3	7.2	588
1B4G	18	30.6	51.5	10.3	620
1B4G	19	23.1	50.3	8.9	696
AVERAGE	-	27.9	58.3	8.6	633
2A4E	34	59.7	81.5	7.5	638
2B4E	32	57.1	53.9	8.2	582
2B4E	35	60.2	64.4	8.9	714
2C4E	21	39.4	36.2	8.2	384
2C4E	22	39.0	42.5	6.5	360
2D4E	38	60.2	51.3	11.7	353
2D4E	39	52.8	42.7	12.3	353
AVERAGE	-	52.6	53.2	10.3	483
3A4A	44	7.0	46.4	2.4	822
3A4A	45	10.4	59.9	2.7	802
3B4A	46	9.6	61.0	2.2	579
3C4A	49	13.5	51.9	2.1	381
3D4A	52	17.0	67.9	2.5	415
AVERAGE	-	11.5	57.4	2.4	600

Notes: a) Recoveries calculated from composite head assay and unadjusted test distribution data.

b) It is estimated that silver and gold recoveries are accurate to within ±10%.

c) Au and Ag in g/tonne.

2.44 Predicted Plant Metallurgy

An approximate mine model based on preliminary cross sections generated by D. Hanson indicated the relative occurrance of the three major ore types. This data was then used to produce a weighted metallurgical balance for all elements of interest in the Vangorda ore body. See tables 27 and 28 below.

TABLE 27

VANGORDA METALLURGY BY ORE TYPE

ORE TYPE	RELATIVE OCCURRENCE IN DEPOSIT %	PREDICTED METALLURGY									
		ASSAYS						DISTRIBUTION			
		Pb	Zn	Au	Ag	Hg	As	Pb	Zn	Au	Ag
4G Baritic	31	53 *	* 55	6.0 *	650 *	80 440	50 20	84 *	* 80	40 *	65 *
4E Pyritic	39	49 *	* 51	10.0 *	480 *	60 250	200 50	77 *	* 72	50 *	50 *
4A Quartzitic	30	48 *	* 53	2.4 *	600 *	40 250	500 200	81 *	* 79	15 *	55 *

Notes: a) Data from best adjusted test data from type testing program.

b) Relative occurrence data calculated from the mine model available in 1980.

c) Au, Ag, As, and Hg, in g/tonne.

TABLE 28

PREDICTED PLANT METALLURY - VANGORDA ORE

CONCENTRATE	ASSAYS						DISTRIBUTION			
	Pb	Zn	Au	Ag	Hg	As	Pb	Zn	Au	Ag
Lead	50	*	6.5	575	60	250	80	*	35	55
Zinc	*	52.8	*	*	300	100	*	77	*	*

Notes: a) Metallurgy calculated from weighted average of type testing data.

b) Au, Ag, As, and Hg, in g/tonne.

2.5 COMPATABILITY TESTING

2.5 COMPATABILITY TESTING

2.51 Laboratory Testwork

Having established the metallurgy for each ore type and then deducing the overall metallurgy for each ore body, the next point to consider was the feasibility of milling the various ores concurrently. As discussed earlier in this section, both pilot plant studies and all recent laboratory testwork have been designed to emulate achievable conditions in the modified Cyprus Anvil mill.

Since the Vangorda material is only a small fraction of the total remaining ore in the area, interaction effects were studied only between Grum and Cyprus Anvil ore. The testwork described here was designed in cooperation with W. Muir, Plant Metallurgist at Faro, and various members of the Feasibility and Development Group. The work was performed at the Lakefield Research Laboratory in 1979.

The testwork comprised several open circuit cleaner tests utilizing the standard flowsheet, with grinding at about 50 microns P_{80}, and a reagent pattern approximating to that employed at Cyprus Anvil. The samples consisted of drill core from the Faro deposit and the screened remnants of the pilot plant sample treated at Lakefield during the 1977 Grum testwork. Sample composition was as follows:

TABLE 29

HEAD ASSAYS OF COMPOSITES

SAMPLE	ASSAYS %				
	Cu	Pb	Zn	Fe	S
Grum	0.14	4.98	9.13	33.10	33.9
Cyprus Anvil	0.18	3.15	4.43	34.80	32.2

Notes: a) Grum sample originated from materials stored at Lakefield.

b) Cyprus Anvil sample was provided by W. Muir plant metallurgist and comprised principally Type 4E Pyritic species.

2.52 Base Metallurgy by Ore Type

First standard open circuit cleaner tests were performed on Grum and Cyprus Anvil samples to obtain data about the base conditions. The results of two test pairs are shown below in Table 30.

TABLE 30

BASE METALLURGY FOR GRUM & CYPRUS ANVIL SAMPLES

PRODUCT	WEIGHT %	ASSAYS % Pb	Zn	% DISTRIBUTION Pb	Zn
CYPRUS ANVIL ORE					
Pb Cleaner Concentrate	3.01	74.9	1.81	81.7	1.3
Pb 1st Cleaner Conc.	4.61	54.8	4.31	91.3	4.6
Zn Cleaner Concentrate	6.04	0.36	52.80	0.8	74.3
Zn Rougher Concentrate	30.81	0.31	12.9	3.9	92.9
Zn Flotation Tailing	64.68	0.20	0.17	4.6	2.5
CYPRUS ANVIL ORE					
Pb Cleaner Concentrate	3.06	72.8	2.03	79.8	1.4
Pb 1st Cleaner Conc.	5.68	44.9	5.21	91.4	6.8
Zn Cleaner Concentrate	5.48	0.34	53.8	0.7	67.8
Zn Rougher Concentrate	13.00	0.48	29.5	2.2	88.3
Zn Flotation Tailing	81.32	0.22	0.26	6.4	4.9
GRUM ORE					
Pb Cleaner Concentrate	6.49	60.3	7.52	81.3	5.4
Pb 1st Cleaner Conc.	20.65	21.6	12.20	92.5	27.9
Zn Cleaner Concentrate	10.27	0.50	51.70	1.1	58.5
Zn Rougher Concentrate	27.25	0.65	22.80	3.7	68.4
Zn Flotation Tailing	52.10	0.36	0.65	3.8	3.7
GRUM ORE					
Pb Cleaner Concentrate	6.85	56.2	9.12	80.6	6.9
Pb 1st Cleaner Conc.	15.12	28.8	12.4	91.2	20.7
Zn Cleaner Concentrate	10.21	0.63	52.8	1.3	59.6
Zn Rougher Concentrate	21.60	0.70	31.1	3.2	74.1
Zn Flotation Tailing	63.28	0.43	0.73	5.6	5.2

2.53 Tests with Ore Mixtures

Two more tests were then performed with a 50:50 mixture of Grum and Anvil ores. The test procedures were the same and the results are reported below in Table 31.

TABLE 31

METALLURGY OF MIXTURE 50:50 GRUM & ANVIL

PRODUCT	WEIGHT %	ASSAYS % Pb	ASSAYS % Zn	% DISTRIBUTION Pb	% DISTRIBUTION Zn
Pb Cleaner Concentrate	4.72	68.6	4.57	85.1	3.2
Pb 1st Cleaner Conc.	12.27	28.8	10.70	92.8	19.6
Zn Cleaner Concentrate	8.38	0.51	52.5	1.1	65.9
Zn Rougher Concentrate	23.80	0.53	21.60	3.3	76.9
Zn Flotation Tailing	63.93	0.23	0.36	3.9	3.5
Pb Cleaner Concentrate	4.41	69.8	5.30	80.5	3.5
Pb 1st Cleaner Conc.	9.30	37.3	9.65	91.2	13.5
Zn Cleaner Concentrate	8.01	0.38	53.50	0.8	64.9
Zn Rougher Concentrate	20.87	0.56	25.8	3.1	81.7
Zn Flotation Tailing	69.83	0.31	0.45	5.8	4.8

By rearranging and averaging the data in Tables 29 and 30 it is possible to show that the results for the mixture of ores falls, as expected, between that recorded for the two ore sources when separately tested.

An interesting point about these results is that the blended mixture produced results which were slightly better than the arithmetic average of both ore types individually. This is a peculiar but not unique occurrance with blends of ores and is probably due to preferential grinding of the relatively softer Grum ore.

TABLE 32

SUMMARY OF TEST DATA

CONDITION		ASSAYS Pb	ASSAYS Zn	DISTRIBUTION Pb	DISTRIBUTION Zn
Anvil - Base Metallurgy	Lead	73.9	*	80.8	*
	Zinc	*	53.3	*	71.1
Grum - Base Metallurgy	Lead	58.3	*	81.0	*
	Zinc	*	52.3	*	59.1
50:50 Blend Metallurgy	Lead	69.2	*	82.8	*
	Zinc	*	52.8	*	65.1

Notes: a) All data from LR 2176 No. 5

b) Test data refers to batch tests which are not directly compable to the predicted plant data.

TABLE 33

COMPARISON OF

ANTICIPATED AND ACTUAL METALLURGY FOR BLENDED ORES

CONDITION		ASSAYS Pb	ASSAYS Zn	DISTRIBUTION Pb	DISTRIBUTION Zn
Anticipated Metallurgy (Arithmetic Average)	Lead	66.1	*	80.9	*
	Zinc	*	52.8	*	65.1
Actual Metallurgy	Lead	69.2	*	82.8	*
	Zinc	*	53.0	*	65.4

Notes: a) Again data refers to batch cleaner test results, unadjusted for redistribution of cleaner tails.

2.6 SPECIAL CHARACTERISTICS OF THE CONCENTRATES

2.6 SPECIAL CHARACTERISTICS OF CONCENTRATES

There are several characteristics of mineral concentrates which are of considerable significance in shipping and sales which have not yet been discussed. The three most important factors, aside from major metal contents, are concentrate flow moitures, moisture content and trace element analyses.

2.61 Flow Moistures

Flow moisture is an approximate physical test method which permits an estimate to be made of the point at which plastic flow of concentrates might occur. Clearly the onset of plastic flow will depend on mean particle size, particle shape and the range of various sizes present. The effect of plastic flow of large concentrate masses on ship stability and rolling moment are apparent.

To determine the effect of finer grinding of the concentrates on flow moisture, samples originating from the fine grind tests at Lakefield Research were subsequent to a flow moisture test. The results of these standard tests are summarized below.

TABLE 34

FLOW MOISTURE

	NORMAL	FINE GRIND
Lead	8.1	9.7
Zinc	9.8	11.2

Notes: a) Normal - Nov. 1979 test result for a six month certificate.

b) Normal P_{80} for concentrates 30-40 microns.

c) Fine grind P_{80} 10-20 microns.

d) The increase in flow moisture point with finer grinding was unexpected. Usually flow moistures tend to decrease with finer particle size.

2.62 Moisture Contained in the Concentrates

The upgraded dewatering plant will accomodate all the planned concentrate production and produce concentrates containing 4.5% moisture under ideal conditions. However, these ideal conditions assumed in the theoretical calculations of dryer capacity seldom exist in practice. Therefore, as shown in the table below the moisture contents will decrease from present levels but will probably not immediately reach the target moisture content of 4.5%. The probable initial plant performance is shown below in Table 35.

TABLE 35

MOISTURE CONTENTS OF CONCENTRATES

CONDITION	LEAD CONC. % WATER	ZINC CONC. % WATER
1978-1979 Averages	5.2	6.6
Theoretical Plant Performance	4.5	4.5
Initial Predicted Performance	4.7	5.0

2.63 Spectrographic Analyses

Some trace elements exercise considerable influence on the smelter process and as such, incur significant penalties to the seller of concentrates. The detection of these minor or trace elements is usually achieved by performing a spectrographic analysis of the concentrates.

TABLE 36

SPECTROGRAPHIC DATA - FINAL CONCENTRATES

(Results from Can-Test Vancouver)

ELEMENT		VANGORDA						GRUM		ANVIL ORE	
		TYPE 4A		TYPE 4E		TYPE 4G		PILOT PLANT COMPOSITE		PILOT PLANT COMPOSITE	
		ZINC	LEAD	ZINC	LEAD	ZINC	LEAD	ZINC	LEAD	ZINC	LEAD
Aluminum	Al	1.	1.	1.	0.1	0.2	0.2	0.1	0.05	N.D.	N.D.
Antimony	Sb	N.D.	0.1	N.D.	0.05	N.D.	0.3	N.D.	0.15	N.D.	N.D.
Arsenic	As	TRACE	*	N.D.	0.03	N.D.	0.05	0.15	N.D.	0.01	0.0002
Barium	Ba	0.1	0.3	*	*	*	*				
Beryllium	Be	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Bismuth	Bi	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Boron	B	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Cadmium	Cd	TRACE	N.D.	TRACE	N.D.	TRACE	N.D.	0.05	0.1	0.05	N.D.
Calcium	Ca	1.	0.5	2.	1.	1.	1.	0.10	0.07	0.05	0.07
Chromium	Cr	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Cobalt	Co	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Copper	Cu	0.3	0.1	*	*	0.1	0.3	0.2	0.5		
Gallium	Ga	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Iron	Fe	MAJOR	MAJOR	MAJOR	MAJOR	MAJOR	MAJOR	MAJOR	MAJOR	MAJOR	MAJOR
Lead	Pb	*	MATRIX	*	MATRIX	*	MATRIX	MAJOR	MATRIX	MAJOR	MATRIX
Magnesium	Mg	1.	0.1	2.	0.3	1.	0.5	0.02	0.02	0.1	0.01
Manganese	Mn	0.1	0.07	0.3	0.2	0.2	0.3	0.03	0.01	0.02	0.01
Molybdenum	Mo	N.D.	N.D.	TRACE	TRACE	N.D.	TRACE	N.D.	N.D.	N.D.	N.D.
Niobium	Nb	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Nickel	Ni	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Potassium	K	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Silicon	Si	3.	5.+	0.5	1.	0.3	1.	1.5	1.5	0.1	0.1
Sodium	Na	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Strontium	Sr	TRACE	0.001	0.03	0.05	0.01	0.05	N.D.	N.D.	N.D.	N.D.
Tantalum	Ta	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Thorium	Th	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Tin	Sn	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Titanium	Ti	0.1	0.3	0.01	0.01	0.01	0.01	N.D.	0.01	N.D.	N.D.
Tungsten	W	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Uranium	U	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Vanadium	V	0.001	0.005	0.001	0.001	0.001	0.001	N.D.	N.D.	N.D.	N.D.
Zinc	Zn	MATRIX	*	MATRIX	*	MATRIX	MAJOR	MATRIX	*	MATRIX	*

Notes: a) Percentages of the various elements expressed in these analyses may be considered accurate to within plus or minus 35 to 50% of the amount present.

b) Semi-quantitative spectrographic analytical results for gold and silver are normally not of a sufficient degree of precision to enable calculation of the true value of ores. Therefore, should exact values be required, it is recommended that these elements be assayed by the conventional Fire Assay Method. Quantitative and Fire Assays may be carried out on the retained pulp samples.

c) Silicon, aluminum, magnesium, calcium and iron are normal components of complex silicates.

d) MATRIX - Major constituent
MAJOR - Above normal spectrographic range
TRACE - Detected by minor amounts
N.D. - Not detected
* - Suggest assay (above 0.3%)