CANADA

DEPARTMENT OF ENERGY, MINES AND RESOURCES

OTTAWA

IR 69-67

August 1969

PILOT PLANT INVESTIGATION OF A SILVER-LEAD ORE FROM DOLLY VARDEN MINES LTD., ALICE ARM, B.C.

by

T. F. BERRY AND R. W. BRUCE

Mineral Processing Division

NOTE: This report relates essentially to the samples as received. The report and any correspondence connected therewith shall not be used in full or in part as publicity or advertising matter.

COPY NO.

August 1969

Mines Branch Investigation Report IR 69-67

PILOT PLANT INVESTIGATION OF A SILVER-LEAD ORE FROM
DOLLY VARDEN MINES LTD., ALICE ARM, B.C.

by

T. F. Berry* and R. W. Bruce**

- - -

SUMMARY OF RESULTS

Daily belt-feed head samples of the ore gave an average analysis of 12.75 oz Ag/ton and 0.44% Pb.

A drum of silver-lead concentrate, weighing 350 pounds, which was produced during the pilot plant investigation, analyzed as follows:

Silver	(Ag)	-	416.54	oz/ton
Gold	(Au)	-	0.02	" "
Lead	(Pb)	-	12.68	per cent
Copper	(Cu)	-	1.02	" "
Arsenic	(As)	-	0.23	" "
Antimony	(Sb)	-	0.66	" "
Iron	(Sol Fe)	-	28.45	" "
Sulphur	(Tot S)	-	3.99	" "
Barium	(Ba)	-	11.66	" "
CaO		-	0.72	" "
Silica	(SiO_2)	-	14.14	" "
Insoluble		-	15.69	" "

A grind of 80% to 85% minus 200 mesh was necessary to ensure the liberation of the minerals.

Based on silver assays, unit-cell concentrates assaying in excess of 400 oz Ag/ton with recoveries of over 70% were readily achieved.

The best mill run (No. 5) gave a silver-lead concentrate assaying 424.0 oz Ag/ton and 12.78% lead with a recovery of 90% of the silver and 81.6% of the lead. An additional 5.5% of the silver and 6.9% of the lead was recovered in a pyrite concentrate. Batch cyanidation tests of a pyrite concentrate showed that about 70% of the silver in the concentrate was amenable to recovery by cyanidation.

---

*Technical Officer, ** Head, Non-Ferrous Minerals Section, Mineral Processing Division, Mines Branch, Department of Energy, Mines and Resources, Ottawa, Canada.

A comparative grindability test indicated that the ore was of the medium-hard variety with a work index of 16.6 kWh/short ton.

The settling rate of the silver-lead concentrate was very slow but the addition of small amounts of settling agent increased the rate markedly.

Bench filtration tests on the flotation concentrate showed that a settling agent is necessary to obtain an acceptable filtration rate.

CONTENTS

INTRODUCTION

On May 17, 1969, a pilot plant investigation on a silver-lead ore from Dolly Varden Mines Ltd. was started. This work, requested by Dr. A. C. Skerl, Managing Director, Dolly Varden Mines Ltd., 304-1033 West Pender Street, Vancouver 1, B.C., was the result of a feasibility study of the orebody and was designed to finalize a treatment procedure. Mr. R. C. Smith, Mill Superintendent of Dolly Varden Mines Ltd. was present during most of the pilot plant test and had overall responsibility for the work.

On April 26, 1969, a shipment of 78 sealed, steel drums containing 19.5 tons of lump ore was received from the property which is located at the head of Alice Arm inlet on the British Columbia coast at about 55°30' north latitude.

DETAILS OF INVESTIGATION

Ore Preparation

The ore, after crushing to 1/2 inch, was mixed with a front-end loader and was then reduced to minus 1/4 inch. An automatic sampler cut out a representative head sample for whatever additional work might be necessary at the conclusion of the pilot plant test. Neither a mineralogical examination of the ore nor a chemical head sample analysis of the ore was obtained during the pilot plant investigation.

Flowsheet and Bench Testing

The basic flowsheet used in the treatment of this ore is shown in Figure 1 of the Appendix. A No. 7 Denver flotation machine was used as a unit cell in the grinding circuit to recover a high-grade silver-lead concentrate. Because of the small amount of ore available for the investigation. An 80-mesh Sweco screen was used in closed circuit with the ball mill instead of a cyclone or a mechanical classifier to facilitate the rapid establishment of an optimum grind.

The remainder of the circuit was straight flotation of a silver-lead concentrate and a pyrite concentrate. An attempt was made in Run No. 7 to recover additional silver by grinding and flotation of the final pyrite concentrate.

The Appendix of this report contains the flowsheets used and the metallurgical balances obtained during the pilot plant testing. In addition, the results of a comparative work index determination on the ore are included and the results of settling and filtration tests on the final silver-lead concentrate and cyanidation tests on the final pyrite concentrate.

CONCLUSIONS AND DISCUSSION

The ore, which had the appearance of being very hard, had a comparative work index of 16.6 kWh/short ton and falls in the medium-hard range. During the pilot plant testing, the ore ground fairly easily to between 80% and 90% minus 200-mesh, at which grind an optimum recovery of the silver and the lead was obtained.

The No. 7 Denver flotation cell installed to act as a unit cell recovered a coarse, high-grade, silver-lead concentrate with a high recovery. In Run No. 3, based on the silver assays showing a ball-mill discharge of 10.40 oz Ag/ton, a unit cell concentrate of 513.06 oz Ag/ton and a unit-cell tailing of 3.00 oz Ag/ton, a calculated recovery of 71.6% of the silver was obtained. In plant operations this grade of unit-cell concentrate would be considered a finished product.

Batch, laboratory flotation tests done by a private metallurgical consulting company indicated that a complex arrangement of reagents, including the use of Na_2S and H_2SO_4, might be necessary to affect a high silver recovery. In the first test of the pilot plant investigation sulphuric acid was used, but the pH in the pyrite flotation circuit was difficult to control, and because an excessive amount of acid was necessary, this scheme was discontinued.

Generally, optimum flotation conditions were achieved by the simple expedient of using soda ash in the grind to maintain a flotation pH of about 10, together with a small mount of Aeroflot Promoter 242 as a collector-frother. Additional collector was added to the silver-lead conditioner and a small amount of pine oil was used as a supplementary frother. The pyrite was floated using amyl xanthate and pine oil. Copper sulphate, added to the pyrite-conditioner, may have been beneficial.

Various cell arrangements in the silver-lead and in the pyrite circuits were tried to obtain the best grade consistent with a high recovery. The best results were obtained in Run No. 5.

An attempt was made to recover additional silver from the pyrite concentrate. The concentrate was thickened and ground. Lime and sodium cyanide were used to depress the pyrite while the silver minerals were floated with Aeroflot Promoter 242. This treatment scheme, shown in Run No. 7, was only partially successful.

Batch cyanidation tests, which were carried out on the final pyrite concentrate, recovered a maximum of 69.5% of the silver in the pyrite concentrate. Thus, in the case of Run No. 5, only 3.8% of the silver in the ore would be recovered by this means. Because of the high capital cost of a cyanidation plant, it is doubtful if cyanidation of the pyrite concentrate would be economic at this time. However, the pyrite concentrate should be stock-piled for possible future treatment.

The final silver-lead concentrate was very slow in settling, but with the addition of a settling agent, the rate increased markedly. In the bench settling tests, M.G.L. Separan was used, although probably the same results would be obtained with most commercial settling agents.

With the use of a settling agent, adequate filtration rates will be obtained.

ACKNOWLEDGEMENTS

The authors wish to thank the personnel of the Analytical Chemistry Section of the Mineral Sciences Division of the Mines Branch for their contribution to this report.

APPENDIX

RUN NO: 1	FEED RATE: 510 lb/hr
DATE: 5-7-69	TIME OPERATED: 5 hr
ORE: Dolly Varden (Wolfe No.2)	SAMPLING PERIOD: 2 hr

PURPOSE OF RUN: Flotation of Ag-Pb concentrate in a lime circuit with Amyl Xanthate followed by pyrite flotation in an acid (H_2SO_4) circuit.

AVERAGE CONDITIONS DURING SAMPLING PERIOD

Point of Addition	Reagents - lbs/ton ore treated						Product	% S	pH
	Lime	Z-6	P.O.	H_2SO_4					
Ball Mill	4.1						Ball Mill Disch	6.0	
Unit cell		0.005	0.005				Ag-Pb Flot Feed	3.9	11.3
Ag-Pb cond		0.05					Pyrite " "	37.5	
Ag-Pb cell#1			0.009						
" " " #9		0.009							
Pyrite cond		.14		12.0					
" Ro cell #3			0.01						
" Scav cell #1		.06	Trace						

Screen Analysis	+65M	+100M	+150M	+200M	+325M	-325M	-200M
Ag-Pb Flot Feed	-	0.1	7.2	10.5			82.2

METALLURGICAL BALANCE

PRODUCT	WT.%	ASSAYS			DISTRIBUTION - %		
		Ag	Pb		Ag	Pb	
Unit-Cell Conc	2.3	333.00	10.71		63.5	58.7	
Ag-Pb Conc	1.5	164.00	5.03		20.6	18.2	
Pyrite conc	24.0	5.59	0.21		11.2	12.9	
Tailing	72.2	0.79	.06		4.7	10.2	
Head (calcd)	100.0	11.97	0.42		100.0	100.0	

REMARKS: The pH of the pulp during the pyrite rougher flotation was difficult to control. A large amount of acid was required.

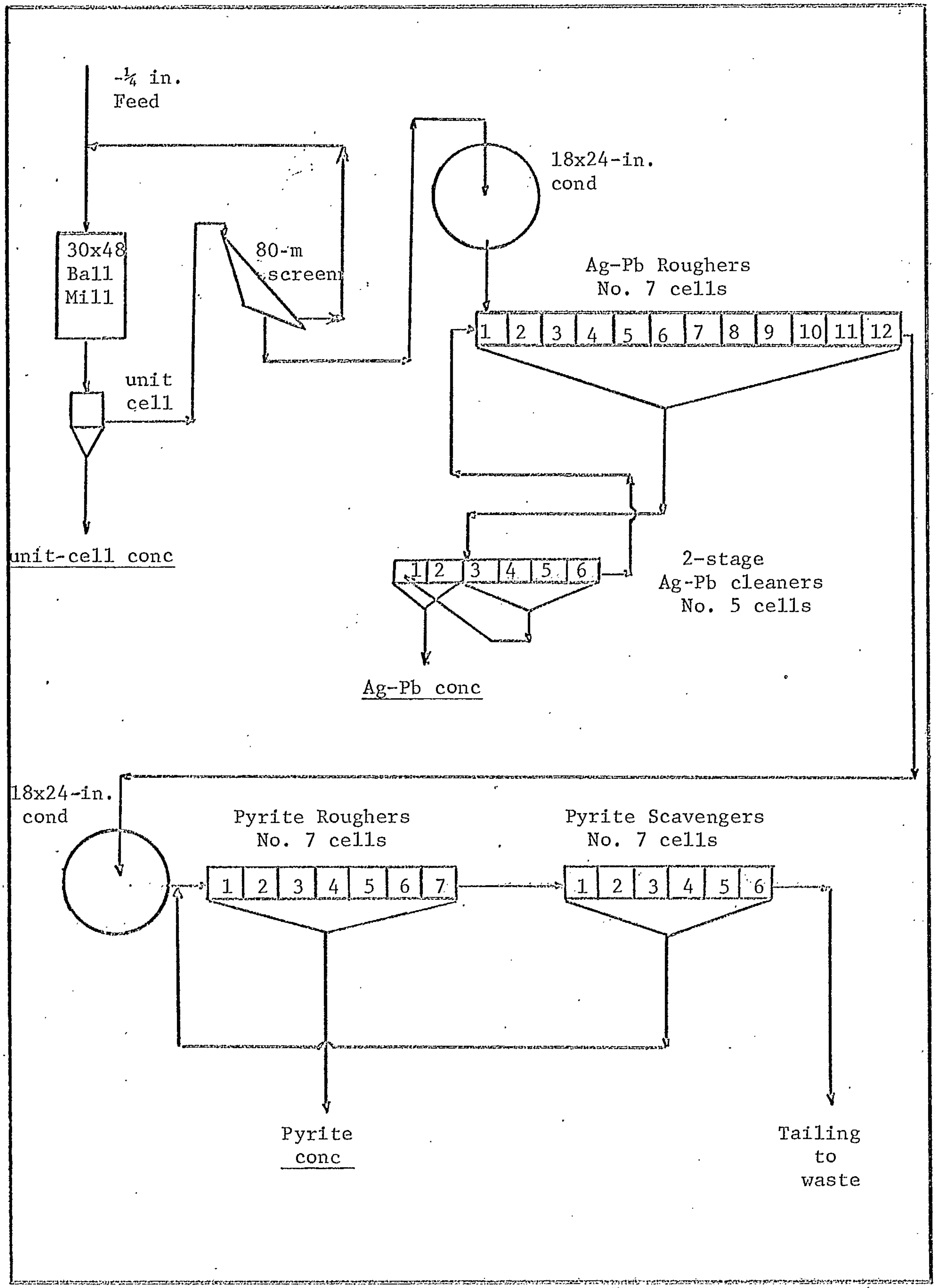

Figure 1: Flowsheet for Mill Run No. 1

RUN NO: 2	FEED RATE: 510 lb/hr
DATE: 5-8-69	TIME OPERATED: 7 hr
ORE: Dolly Varden (Wolfe No. 2)	SAMPLING PERIOD: 1½ hr

PURPOSE OF RUN: To investigate the flotation of the Ag-Pb concentrate with Aerofloat 242 in a soda ash circuit. The pyrite was floated using amyl xanthate and pine oil.

AVERAGE CONDITIONS DURING SAMPLING PERIOD

Point of Addition	Reagents - lbs/ton ore treated: Na_2CO_3	Aero 242	DF 250	Z-6	P.O.		Product	% S	pH
Ball mill	3.2	0.6					Ball mill disch	61	
Unit cell			.03				Ag-Pb cond	35	10.0
Ag-Pb cell#9		-					Pyrite flot feed	36	9.7
Pyrite cond				0.30					
Pyrite cell #3					0.32				
Pyrite scav cell #1				.08	trace				

Screen Analysis	+65M	+100M	+150M	+200M	+325M	-325M	-200M
Ag-Pb Flot Feed	-						

METALLURGICAL BALANCE

PRODUCT	WT. %	ASSAYS Ag	ASSAYS Pb		DISTRIBUTION - % Ag	DISTRIBUTION - % Pb	
Unit-cell conc	0.27	596.00	13.92		22.7	12.5	
Ag-Pb final conc	0.63	567.48	18.93		50.5	39.2	
Pyrite final conc	21.0	6.02	0.25		17.9	17.4	
Flot tailing	78.10	0.82	0.12		8.9	30.9	
Head (calcd)	100.00	7.08	0.31		100.0	100.0	

REMARKS: Dowfroth 250 tended to produce a runny froth when used with Aerofloat 242.

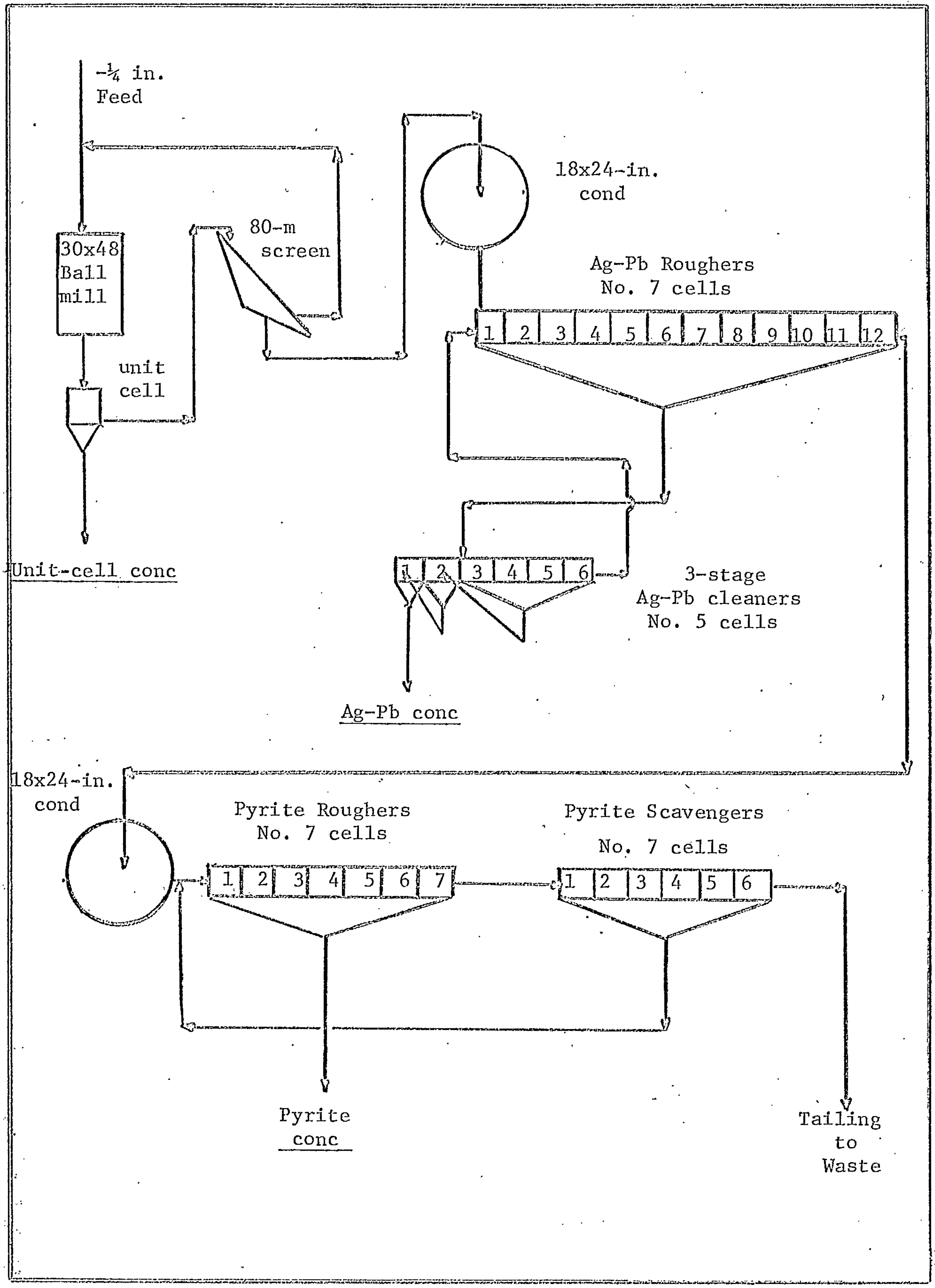

Figure 2: Flowsheet for Mill Run No. 2.

RUN NO: 3	FEED RATE: 510 lb/hr
DATE: May 12, 1969	TIME OPERATED: 5½ hr
ORE: Dolly Varden (Wolfe No. 2)	SAMPLING PERIOD: 2 hr

PURPOSE OF RUN: Unit-cell conc and Ag-Pb rougher conc to No. 1 cleaner (No. 7 cells) then to 3-stage cleaner circuit in No. 5 cells. Pyrite concentrate cleaned in two No. 7 cells.

AVERAGE CONDITIONS DURING SAMPLING PERIOD

Point of Addition	Reagents - lbs/ton ore treated						Product	% S	pH
	$NaCO_3$	Aero 242	P.O.	Z-6					
Ball mill	4.4	0.0012					Ball mill disch	60	
Unit cell			0.19				Ag-Pb cond	30	10.2
Ag-Pb cond		0.009					Ag-Pb cleaner		9.9
Ag-Pb Ro cell No. 7				-			Pyrite flot feed	26	9.8
Pyrite cond				0.32					
Pyrite Scav									
cell No.1			trace	0.08					

Screen Analysis	+65M	+100M	+150M	+200M	+325M	-325M	-200M
Ball mill discharge	4.3	6.0	9.5	10.3			69.9

METALLURGICAL BALANCE

PRODUCT	WT. %	ASSAYS			DISTRIBUTION - %		
		Ag	Pb		Ag	Pb	
Ag-Pb Final conc	0.94	808.00	25.82		79.9	70.2	
Pyrite conc	8.73	9.41	0.35		8.6	9.0	
Flot tailing	90.33	1.20	0.08		11.5	20.8	
Head (calcd)	100.00	9.50	0.35		100.0	100.0	
Head (assay)	-	12.84	0.46		-	-	

REMARKS: An increase in amount of Z-6 resulted in a lower float tailing.

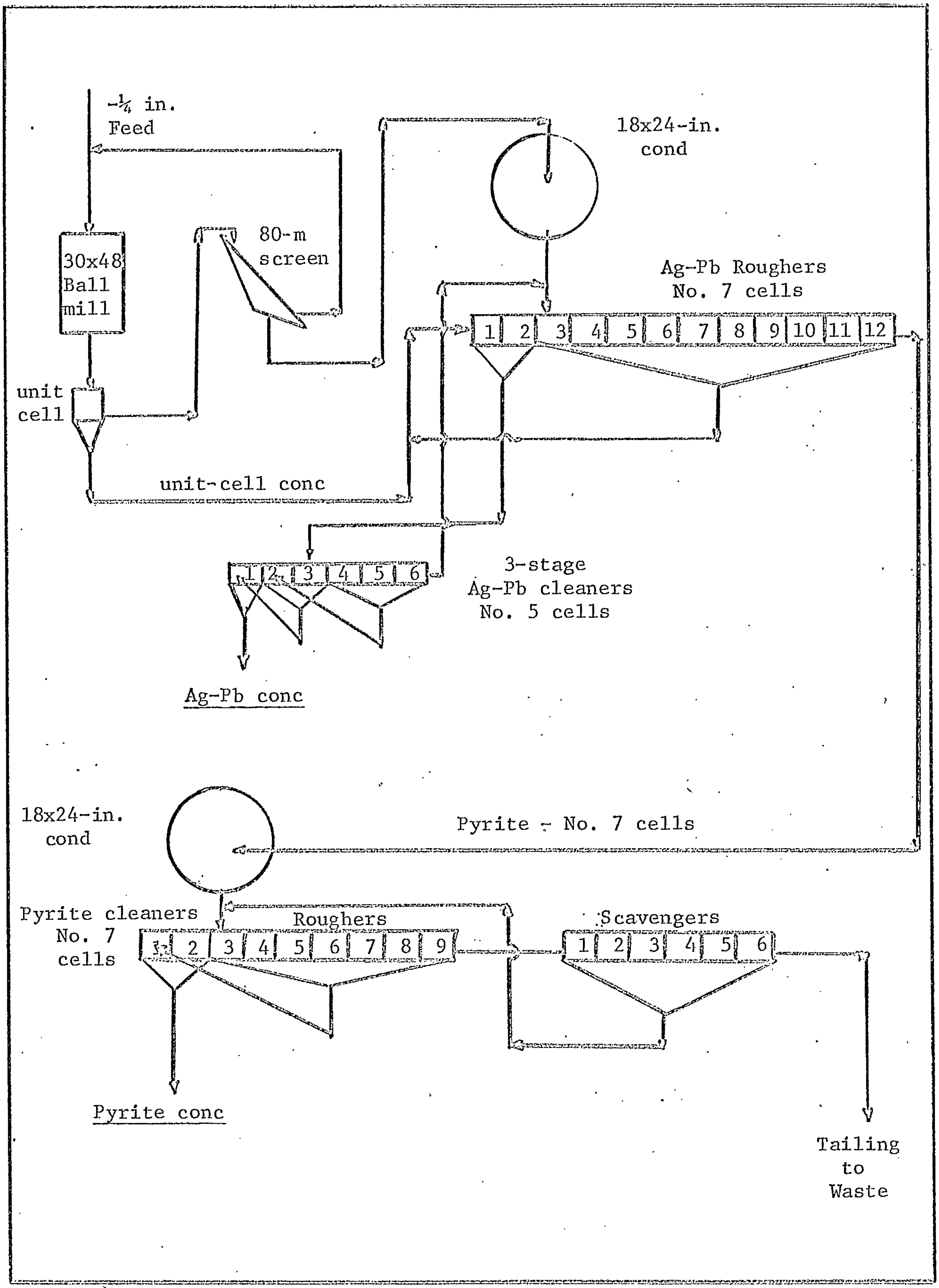

Figure 3: Flowsheet for Mill Run No. 3.

RUN NO: 4	FEED RATE: 504 lb/hr
DATE: May 13, 1969	TIME OPERATED: 4 hr
ORE: Dolly Varden (Wolfe No. 2)	SAMPLING PERIOD: 2 hr

PURPOSE OF RUN: Unit-cell conc to No. 1 cleaner stage.
Pyrite conditioner to pyrite cleaner circuit to remove finished concentrate. Increased Z-6 in pyrite float to lower tailing.

AVERAGE CONDITIONS DURING SAMPLING PERIOD

Point of Addition	Reagents - lbs/ton ore treated						Product	% S	pH
	$NaCO_3$	Aero 242	Z-6	P.O.					
Ball mill	4.4	0.0012					Ball mill disch	60	
Unit cell				0.18			Ag-Pb cond	33	10.1
Ag-Pb cond		0.0018					Pyrite cond	22	9.7
Pyrite cond			0.34				Pyrite cl		9.7
Pyrite Ro #4			0.09						
Pyrite Scav No. 1			0.08	trace					

Screen Analysis	+65M	+100M	+150M	+200M	+325M	-325M	-200M

METALLURGICAL BALANCE

PRODUCT	WT.%	ASSAYS			DISTRIBUTION - %		
		Ag	Pb		Ag	Pb	
Ag-Pb conc	1.1	745.0	24.3		80.1	71.6	
Pyrite conc	12.1	10.06	0.37		11.9	12.1	
Flot tail	86.8	0.945	0.07		8.0	16.3	
Head (calcd)	100.0	10.23	0.37		100.0	100.0	
Head (assay)	-	12.10	0.38		-	-	

REMARKS: Good flotation, no control problems.

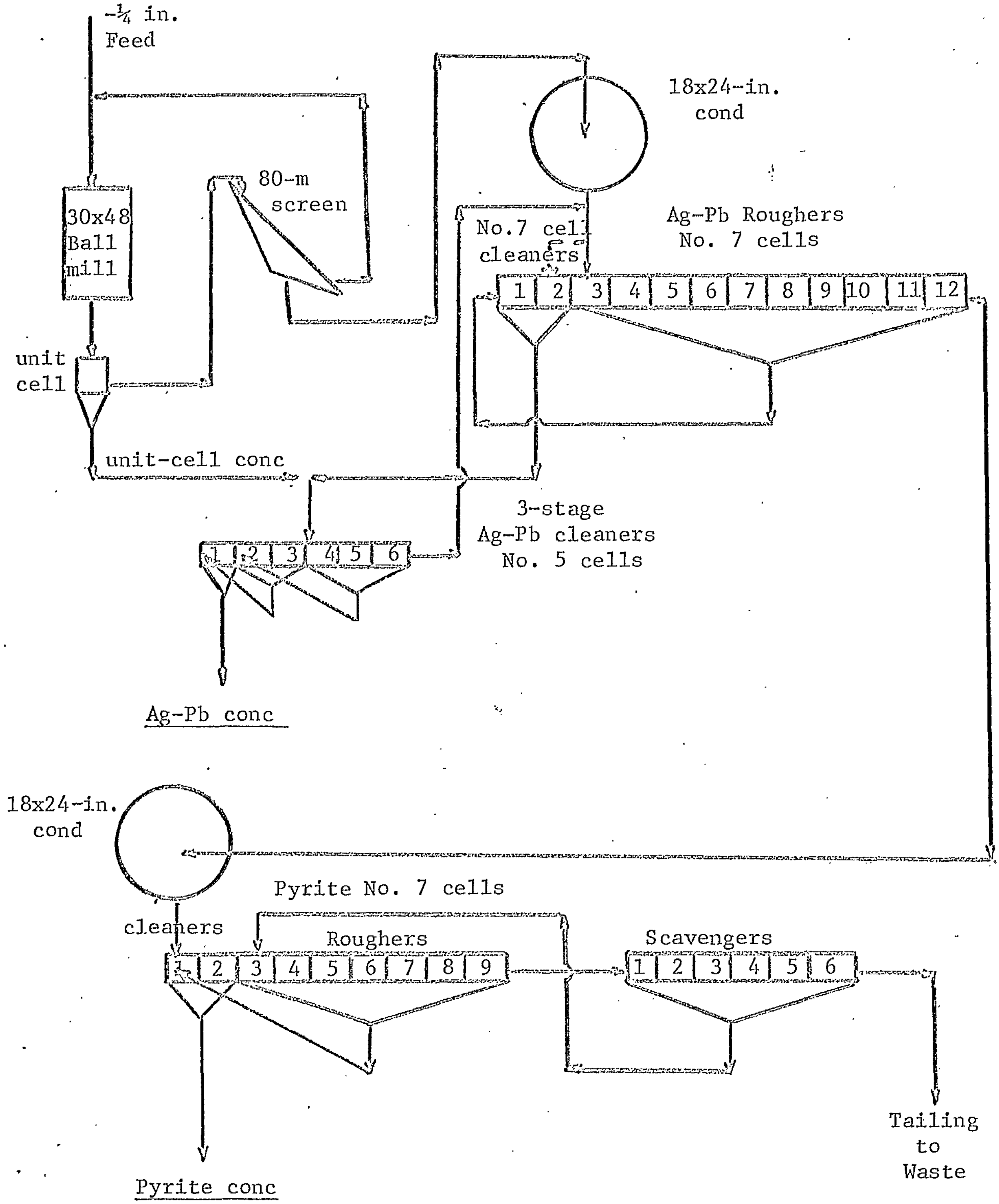

Figure 4: Flowsheet for Mill Run No. 4

RUN NO: 5	FEED RATE: 504 lb/hr
DATE: May 14, 1969	TIME OPERATED: 5 hr
ORE: Dolly Varden (Wolfe No.2)	SAMPLING PERIOD: 2 hr

PURPOSE OF RUN: Pyrite conditioner discharged to first cell of 3 No. 7 pyrite cleaners. $CuSO_4$ in pyrite float.

AVERAGE CONDITIONS DURING SAMPLING PERIOD

Point of Addition	Reagents - lbs/ton ore treated: Na_2CO_3	Aero 242	P.O.	Z-6	$CuSO_4$		Product	% S	pH
Ball mill	4.4	0.0012					Ball mill disch	60	
Unit cell			0.18				Ag-Pb cond	34	10.0
Ag-Pb cond		0.0018					Pyrite cond	30	9.6
Pyrite cond				0.38	0.22				
Pyrite Ro #3				0.11					
Pyrite Scav No. 12			trace	0.10					

Screen Analysis	+65M	+100M	+150M	+200M	+325M	-325M	-200M

METALLURGICAL BALANCE

PRODUCT	WT.%	ASSAYS Ag	Pb		DISTRIBUTION - % Ag	Pb	
Ag-Pb conc	3.4	424.0	12.78		90.0	81.6	
Pyrite conc	12.0	7.42	0.31		5.5	6.9	
Flot tailing	84.6	0.85	0.06		4.5	11.5	
Head (calcd)	100.0	16.02	0.53		100.0	100.0	
Head (assay)	-	12.82					

REMARKS: The copper sulphate seemed to be advantageous in the pyrite circuit.

RUN NO: 6	FEED RATE: 500 lb/hr
DATE: May 20, 1969	TIME OPERATED: 5 hr
ORE: Dolly Varden (Wolfe No. 2)	SAMPLING PERIOD:

PURPOSE OF RUN:

To increase recovery by using finer grind.

AVERAGE CONDITIONS DURING SAMPLING PERIOD

Point of Addition	Reagents - lbs/ton ore treated						Product	% S	pH
	NaCO	Aero 242	P.O.	Z-6	$CuSO_4$				
Ball mill	4.4	0.0012					Ball mill disch	60	
Unit cell			0.18				Ag-Pb cond	32	9.8
Ag-Pb cond		0.0018					Pyrite cond	28	9.2
Pyrite cond				0.38	0.22				
Pyrite Ro #3				0.11					
Pyrite Scav			trace	0.10					
No. 1-2									

Screen Analysis	+65M	+100M	+150M	+200M	+325M	-325M	-200M
Ball Mill Discharge	3.5	5.6	9.0	9.9			72.0

METALLURGICAL BALANCE

PRODUCT	WT. %	ASSAYS			DISTRIBUTION - %		
		Ag	Pb		Ag	Pb	
Ag-Pb Final conc	2.0	468.0	15.0		86.3	75.8	
Pyrite Final conc	11.2	8.0	0.31		8.2	8.8	
Flotation tailing	86.9	0.69	0.07		5.5	15.4	
Head (calcd)	100.0	10.85	0.40		100.0	100.0	
Head (assay)		13.40	0.44				

REMARKS: Finer grind did not improve the recovery over Test 5.

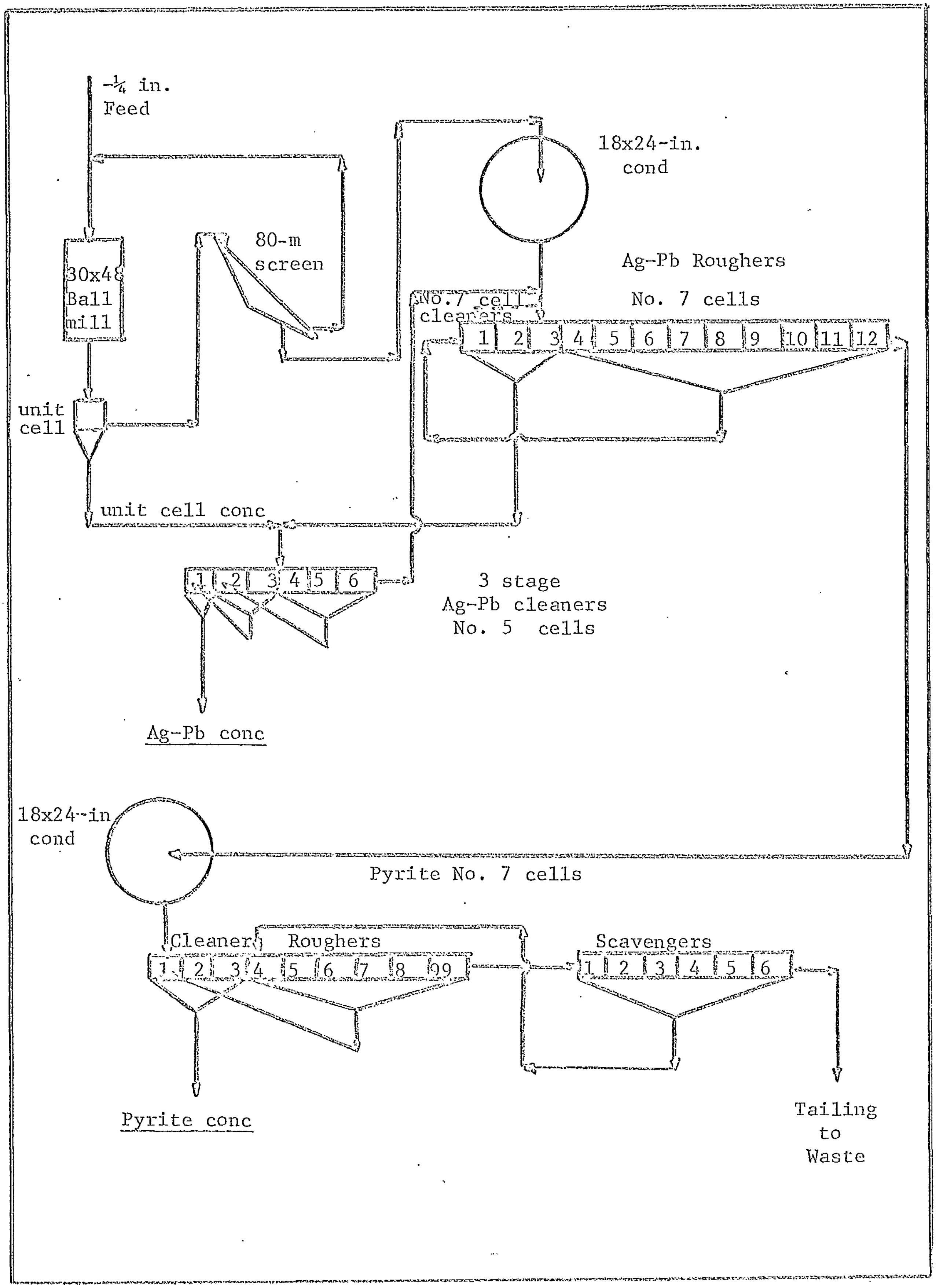

Figure 5: Flowsheet for Mill Run No. 5 and 6.

RUN NO: 7	FEED RATE: 500 lb/hr
DATE: May 23, 1969	TIME OPERATED: 5 hr
ORE: Dolly Varden (Wolfe No. 2)	SAMPLING PERIOD: 2 hr

PURPOSE OF RUN: Same grind as in Run No. 6.

Regrind of final pyrite concentrate and reflotation for additional silver recovery.

AVERAGE CONDITIONS DURING SAMPLING PERIOD

Point of Addition	Reagents - lbs/ton ore treated						Product	% S	pH
	$NaCO_3$	Aero 242	P.O.	$CuSO_4$	Z-6				
Ball mill	4.3	0.001					Ball mill disch	64	
Unit cell			0.018				Ag-Pb cond	36	9.6
Ag-Pb cond		0.002					Pyrite cond	32	9.2
Pyrite cond				0.22	0.40		Regrind mill disch	30	12.1
Pyrite ro #3					0.11				
Pyrite scav #1			trace		0.10				
Regrind mill									
Regrind flot		0.001							

Screen Analysis	+65M	+100M	+150M	+200M	+325M	-325M	-200M

METALLURGICAL BALANCE

PRODUCT	WT. %	ASSAYS			DISTRIBUTION - %		
		Ag	Pb		Ag	Pb	
Ag-Pb conc	2.6	431.02	12.25		87.1	75.0	
Regrind conc	3.5	9.40	0.44		2.6	3.6	
" tail	8.8	6.89	0.26		4.7	5.4	
Flot tail	85.1	0.845	0.08		5.6	16.0	
Head (calcd)	100.0	12.86	0.42		100.0	100.0	
Head (assay)		13.60	0.49				

REMARKS: Note: 4.1 lb lime added to regrind mill and 2.0 lb NaCN to regrind flotation to depress the pyrite.

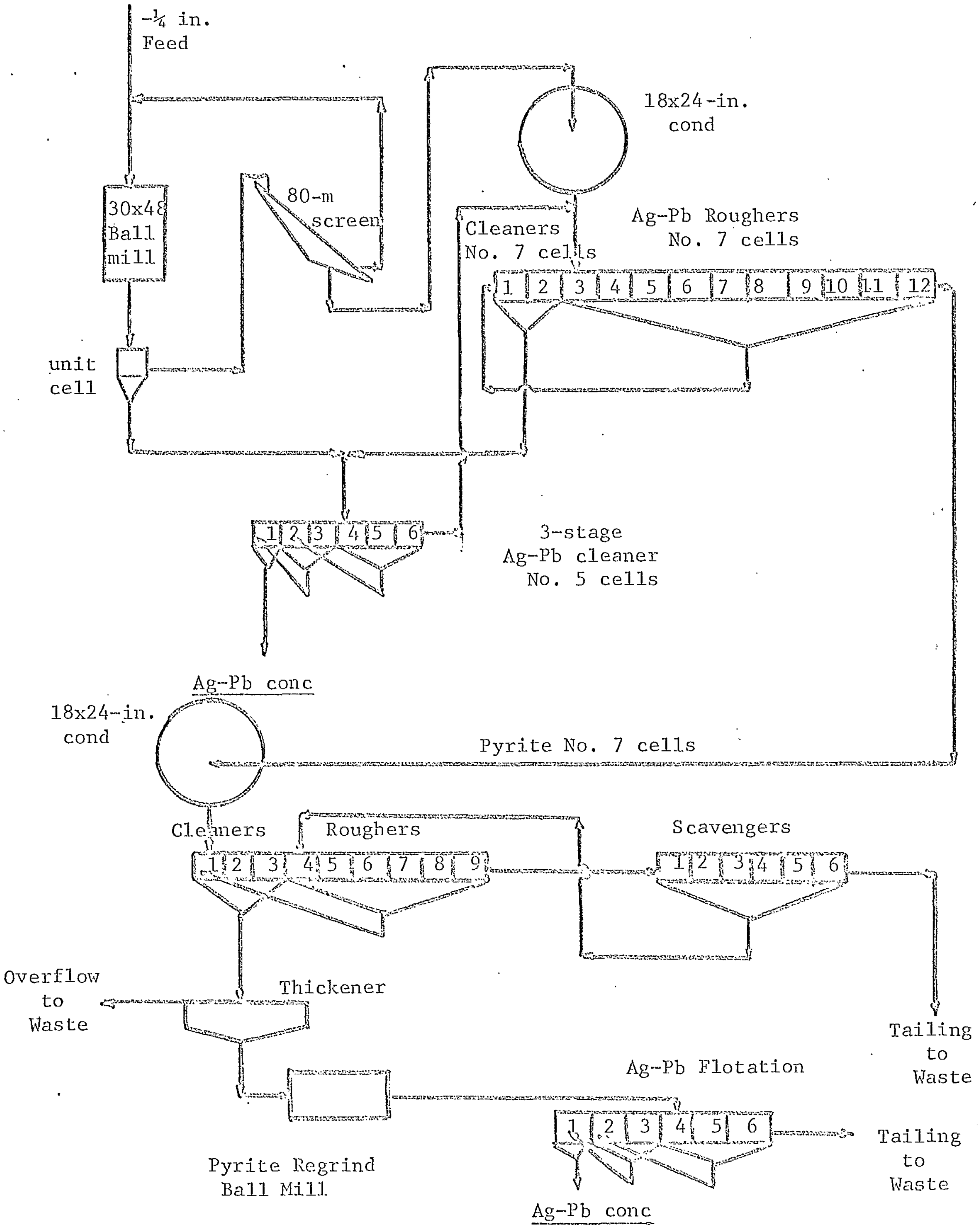

Figure 6: Flowsheet for Mill Run No. 7.

Work Index Determination

The grindability of the Dolly Varden ore was compared with that of an ore of known work index. Following a procedure (1) developed at the Mines Branch, 2000-gram samples of the Dolly Varden ore and the comparison ore were ground under identical conditions in the same batch ball-mill. The screen and cyclosizer results of the ball mill feed and the ground ore were recorded. The results were plotted on log-log graph paper (particle size in microns vs per cent passing) from which 80% passing points in microns were obtained. The particle size analysis is shown in Table 1, and is graphically illustrated in Figure 7.

TABLE 1

Results of Screen and Cyclosizer Tests

Particle Size	Dolly Varden				Comparison Ore			
	Feed		30 min grind		Feed		30 min grind	
	% ret	% pass	% ret	% pass	% ret	% pass	% ret	% pass
+10 mesh	0.4	99.6			0.6	99.4		
14 "	11.2	88.4			11.5	87.9		
20 "	22.2	66.2			19.0	68.9		
28 "	16.6	49.6			18.4	50.5		
35 "	10.4	39.2			13.5	37.0		
48 "	7.7	31.5		100.0	8.6	28.4		100.0
65 "	7.6	23.9	0.7	99.3	7.4	21.0	0.2	99.8
100 "	5.4	18.5	1.4	97.9	5.1	15.9	2.1	97.7
150 "	5.0	13.5	6.7	91.2	3.7	12.2	8.7	89.0
200 "	4.7	8.8	16.3	74.9	3.0	9.2	14.5	74.5
-200 "	8.8				9.2			
+250 "			3.7	71.2			2.4	72.1
40.6 micron			10.3	60.9			7.6	64.5
30.9 "			14.5	46.4			15.1	49.4
22.5 "			12.3	34.1			11.0	38.4
15.5 "			9.4	24.7			8.4	30.0
11.9 "			5.9	18.8			6.0	24.0
-11.9 "			18.8				24.0	
Total	100.0		100.0		100.0		100.0	

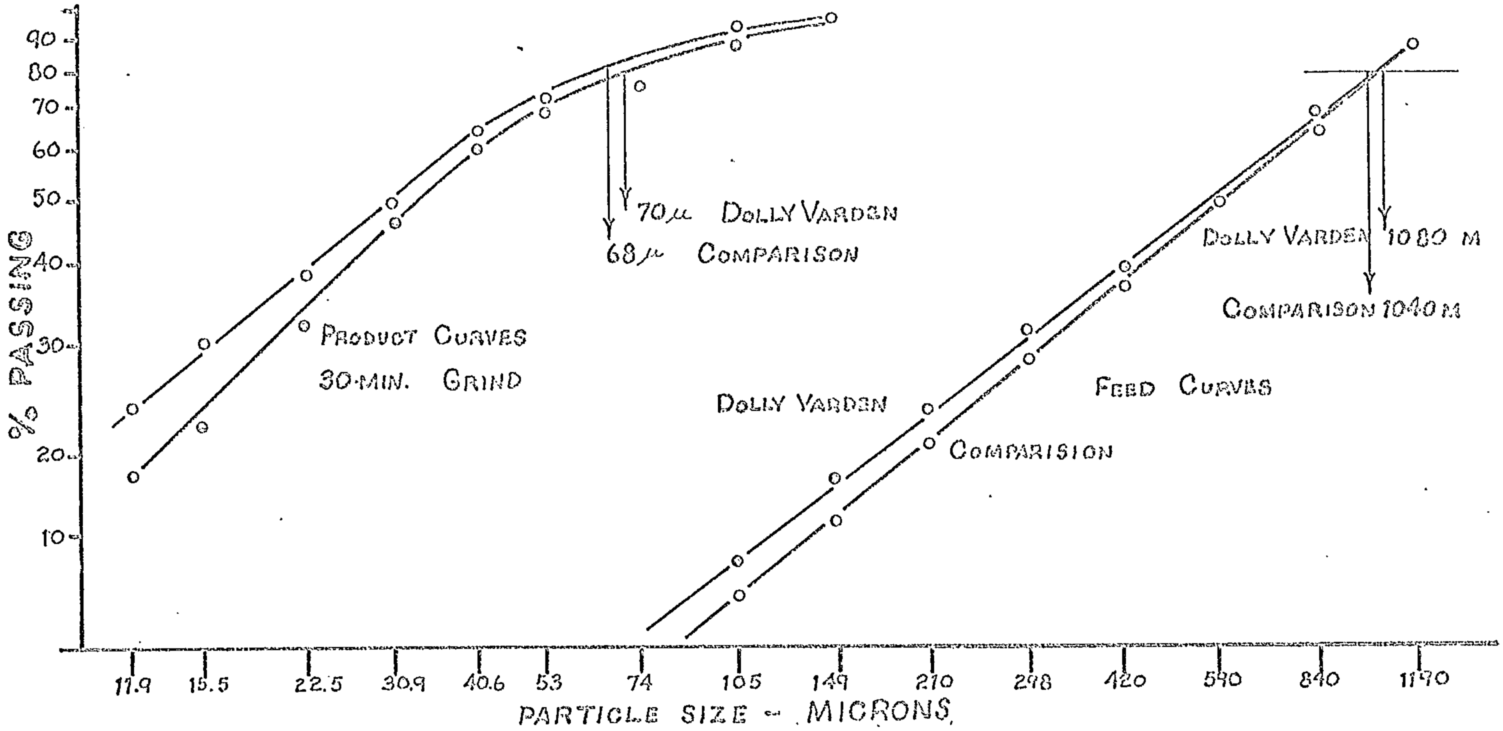

Figure 7 - Size Distribution of Dolly Varden and Comparison Ore

From the curves, the 80% passing points in microns were obtained. These were as follows:

80% Passing Points in Microns

Product	Dolly Varden Ore	Comparison Ore
Feed	1080	1040
30-min grind	70	68

The Bond work index for the comparison ore is 16.4 kWh/short ton. Substituting this figure and those in the above table in the formula developed by F.C. Bond, as shown on the following page,

$$Wi\left(\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right) = Wi\left(\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right)$$

we have
$$Wi\left(\frac{10}{\sqrt{70}} - \frac{10}{\sqrt{1080}}\right) = 16.4\left(\frac{10}{\sqrt{68}} - \frac{10}{\sqrt{1040}}\right)$$

and
$$Wi \text{ (Dolly Varden ore)} = 16.6 \text{ kWh/short ton,}$$

The ball mill horsepower required and from it the ball mill size needed may be determined by using the work index figure.

Assume a mill capacity of 700 tons/24 hour day or 29.2 ton/hour

Assume 3/8-in. ore feed crushed to 80% passing 9500 microns

Assume a product ground to 80% passing 70 microns.

The work required $W = \frac{10\ Wi}{\sqrt{P}} - \frac{10\ Wi}{\sqrt{F}}$

and $W = 16.6\left(\frac{10}{\sqrt{70}} - \frac{10}{\sqrt{9500}}\right)$

and $W = 18.14$ kWh

Required mill power = 18.14 x 29.2 x 1.34 = 709.8 H.P.

Approximate mill sizes required - 9' x 16', 10' x 12', or 10' x 14'.

Settling Tests

A series of settling tests was done on the final concentrate which was shipped to Vancouver. In all of the tests an initial pulp pH of about 10 was achieved through the addition of soda ash in order to approximate the pH of the final Ag-Pb concentrate produced during the pilot plant testing. In the following table the settling rates achieved through the use of Separan MGL are shown at pulp densities of 4:1, 3:1, and 2:1. Also shown are the settling rates achieved using varying amounts of H_2SO_4.

TABLE 2

Results of Settling Tests on Final Concentrate

Reagents lb/ton conc		Test pH	Settling Rate ft/hr at Pulp Densities			Final Pulp Density
Separan	H_2SO_4		4:1	3:1	2:1	
-	-	10.2	4.0	2.75	1.50	1.5
0.0025	-	9.5	9.4	6.25	2.50	1.5
0.005	-	10.0	21.38	15.62	6.88	1.5
0.010	-	9.8	32.5	20.62	7.60	1.5
0.025	-	9.9	47.25	25.0	10.0	1.5
0.050	-	9.8	60.0	37.5	11.25	1.5
-	1.84	9.5	4.90	3.60	1.90	1.5
-	3.68	9.3	4.50	4.00	2.40	1.5
-	5.52	9.0	5.50	4.50	2.25	1.5
-	9.20	8.5	5.50	4.25	2.10	1.5
-	18.4	6.9	6.80	4.60	2.00	1.5

Filtration Tests

A series of filtration tests were carried out on the Ag-Pb final concentrate. The pulp density in all of the tests was 1.5:1 which was the final settled-pulp density in the settling tests. Table 3 shows the test conditions and the results of the tests.

TABLE 3

Results of Filtration Tests on Samples of Ag-Pb Concentrate

Pulp pH	Reagents, lb/ton core		Cake Time, secs		Vacuum, psi		Average thick-ness, in.	Moist-ure, %	Weight, lbs	Rate, lb/sq ft/hr
	Separan	H_2SO_4	Form-ing	Drying	Form-ing	Drying				
10.0			90	180	28	23	3/16	23.3	0.12	13.7
10.0	0.05		30	60	25	22	1/4	23.8	0.17	58.3
10.0	0.005		30	60	27	23	3/16	21.3	0.16	54.8
10.0	0.025		30	60	27	20	1/8	20.1	0.11	37.7
10.0	0.025		60	120	28	22	3/16	20.5	0.19	32.6
8.5		0.01	30	60	27	22	1/8	22.2	0.10	34.3
8.5		0.01	60	120	28	24	3/16	21.6	0.16	27.4
8.5		0.01	90	180	29	25	1/4	21.8	0.18	20.6
6.5		0.025	30	60	27	27	1/8	20.8	0.15	51.4
6.5		0.025	60	120	30	25	1/4	20.4	0.22	37.1

Cyanidation Tests

The pyrite concentrate produced during the pilot plant tests assayed as high as 10.06 oz Ag/ton (Mill Run No. 4). A series of cyanidation tests were carried out on some of the pyrite concentrates produced to determine whether the contained silver was amenable to recovery by cyanidation.

Table 4 shows the test conditions and the results which were obtained.

TABLE 4

Results of Cyanidation Tests on Pyrite Concentrates

Solids %	Grind %-200m	Solution lb/ton		Time hr	Consumption lb/ton Conc		Assay * oz/ton Ag		Extraction Ag %
		NaCN	CaO		NaCN	CaO	Feed	Residue	
33	82.2	1.0	1.0	24	2.0	4.6	8.0	4.32	46.0
11	"	"	"	48	2.8	7.32	"	4.16	48.0
11	"	"	"	72	3.5	7.2	"	4.07	49.1
20	"	"	"	24	4.0	9.2	"	4.63	42.1
11	"	"	"	48	5.28	13.7	"	4.11	48.6
11	"	"	"	72	6.4	14.32	"	3.94	50.8
33	"	4.0	"	"	9.2	8.72	"	3.73	53.7
20	"	"	"	"	8.0	14.6	"	3.80	52.5
33	95.0	1.0	"	"	3.72	8.2	9.4	3.20	66.0
20	"	"	"	"	6.24	16.6	"	3.66	61.1
33	"	4.0	"	"	5.76	7.6	"	2.86	69.5
20	"	"	"	"	8.24	10.8	"	2.98	68.3

* From Internal Report MS AC 69-531

/am