HUMPHREYS SPIRAL BENEFICIATION OF SULPHIDE ORES FROM THE BOHEMIA DISTRICT MUSICK MINE

by

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Typed by Vivian Murbach

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The ore samples used were made available through Mr. Harold Barton of Lane Minerals Incorporated and the material was mined and transported by Mr. Kenneth O. Watkins.

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

LIST OF PLATES

HUMPHREYS SPIRAL BENEFICIATION OF SULPHIDE ORES FROM THE BOHEMIA DISTRICT, MUSICK MINE

OBJECTIVE

The principal objective of this investigation was to determine the feasibility of upgrading or concentrating the sulphide ores of an Oregon mine to an ultimite value which may be marketed profitably. It was proposed to accomplish this upgrading by use of the Humphreys Spiral concentrator.

The secondary objective of this investigation was to obtain operational data for concentration of this type of ore in the Humphreys Spiral. This data should include optimum flow rate, splitter settings, feed size, etc. To date, there is no information available on spiral concentration of this type of ore from Oregon mines.

The experimental work was performed on mine run samples from the Musick Mine in the Bohemia Mining District, township 23 south, range 1 east, Willamette Meridian.

INTRODUCTION

Since the early 1940's, the cost of mining labor has risen from $8.00 to more than $15.00 per day. This increase for mine labor, plus the relatively high capitalization and operational costs of conventional flotation plants and the availability of high-grade and consequently low cost base metals from foreign sources, has led to the closing of many base metal mines in the United States.

The Bohemia Mining District exemplifies the conditions of the problem outlined above. The district is located in a maturely dissected area of the western Cascade Mountains, 38 air miles southeast of Eugene, Oregon. The majority of the producing mines are located at elevations between four thousand and six thousand feet above sea level.

The district has a long record of intermittent production dating from 1880 to 1950 (4, p. 46).

Smelter payments have been made for the metals gold, silver, copper, lead, and zinc. At the present time the district is idle.

Development of new equipment with a lower initial cost and simplified techniques of operation may make reopening of this district possible.

The Humphreys Spiral[1] may be an economic solution to the problem of mineral beneficiation in the district.

The salient features of the Humphreys Spiral are the simplicity of operation and the low capitalization costs. There are no moving parts, floor space per ton of ore treated is very small, and installation, maintenance, and operational costs are very low in a gravity flow operation. Important variables which affect spiral operation are sand-water ratio (pulp density), concentration ratio, flow rate, and feed size.

The steep terrain of the Bohemia Mining District is ideal for construction of a gravity flow plant such as is shown in Figure 1, page 4. It would be possible to operate such a plant with a capacity of several hundred tons per day using only two men per shift. The operational costs of the plant would be limited to payroll, grinding, and minor maintenance.

1. The Humphreys Spiral is a patented device manufactured by the Humphreys Investment Company, 910 First National Bank Building, Denver, Colorado.

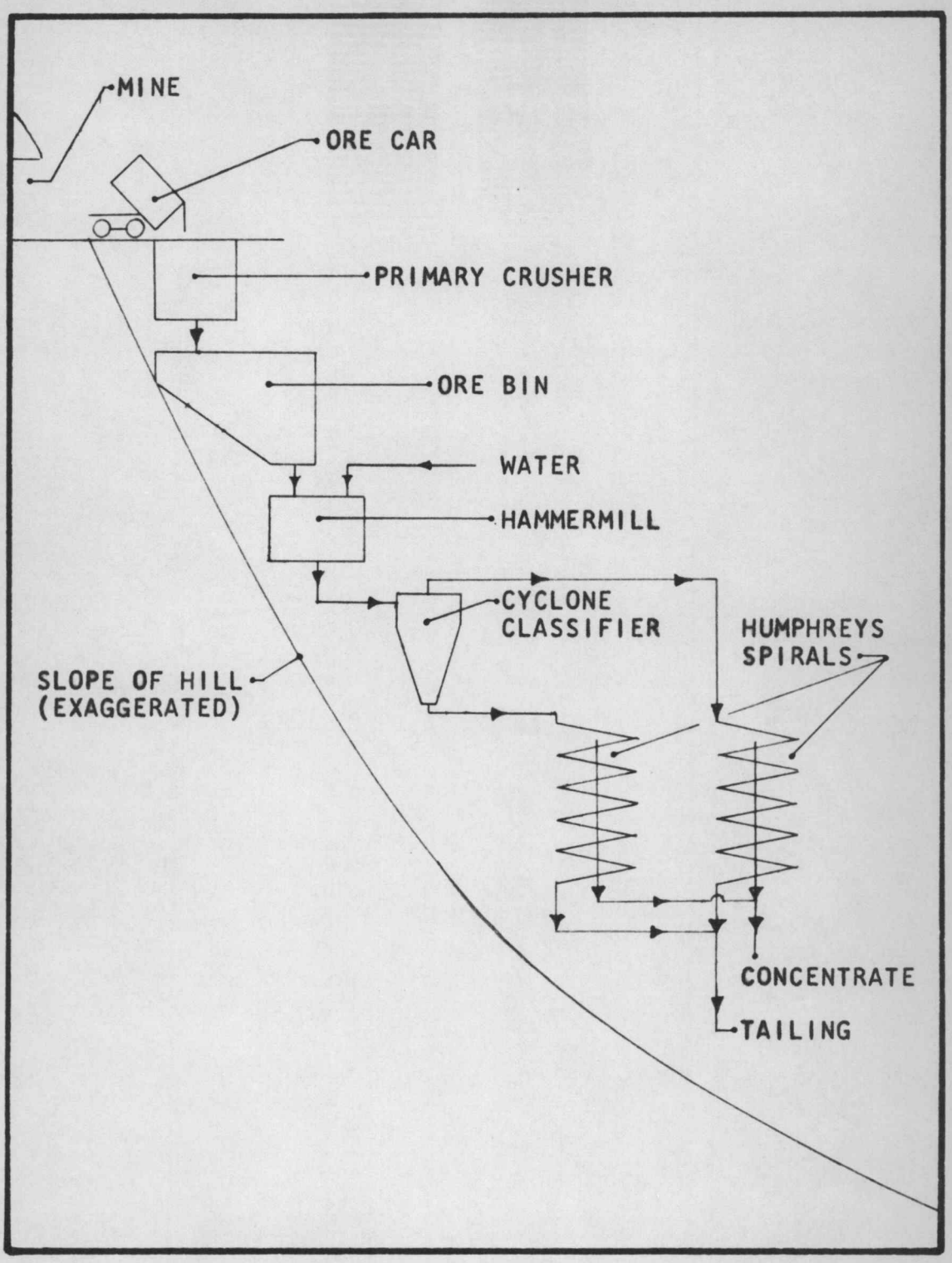

FIGURE 1: THE HUMPHREYS SPIRAL IN GRAVITY FLOW APPLICATION.

DESCRIPTION OF APPARATUS

The Humphreys Spiral is a trough of modified semi-circular cross-section which is wound helically about a vertical axis. As may be seen in Plate 1, page 7, the Spiral used consisted of five complete turns about a vertical axis, each turn having a pitch of 14 3/4 inches. The three turn Spiral shown at the left of Plate 1 was not used in this investigation.

The Spiral is constructed of 15 cast iron sections, each comprising 1/3 of a turn. Each section has a port for drawing off concentrates and, as may be seen in Plate 2, page 8, each port is equipped with a variable splitter which is formed from a "disc" and the size of the port. This "disc" has been bent in an L shape and rests on a countersunk flange inside the port. The portion of the "disc" parallel to the trough is flush with the surface of the trough. Special splitters with extensions were fabricated for operation at low flow rates. The special extensions for the splitters were fabricated by bolting strips of sheet metal to the factory-supplied splitters to extend the splitters farther into the stream. The splitters may be positioned to take specific width cuts from the stream flow. This positioning is accomplished by rotating the splitters in the ports.

A wash water trough is cast along the inside edge of the main spiral trough. This wash water trough carries a supply of clean water. A portion of this water is added to the main stream flow just downstream from each concentrate port.

While it is possible to extend the length of the Spiral by adding more sections, a maximum practical length of 5 turns has been established empirically (11, p. 6).

The closed circuit test spiral used on this project was assembled in accordance with the flow diagram shown in Figure 2, page 10. The system consisted of a 5 turn Humphreys Spiral of commercial size used in a closed circuit with a sump tank and a recirculation pump. The pump was a 1 3/4 inch, rubber lined, Warman Sand Pump. The pump was powered by a 3 horsepower electric motor.

Flow rate in the Spiral was controlled by a bypass line from the pump outlet to the sump tank.

Clean water was obtained from the slurry by installing a small cyclone in the pressure line between the pump and the Spiral. The overflow of the cyclone contained almost no solids greater than 400 mesh.

Plate 3, page 9, shows details of the wash water cyclone, the sump tank, and the sand pump.

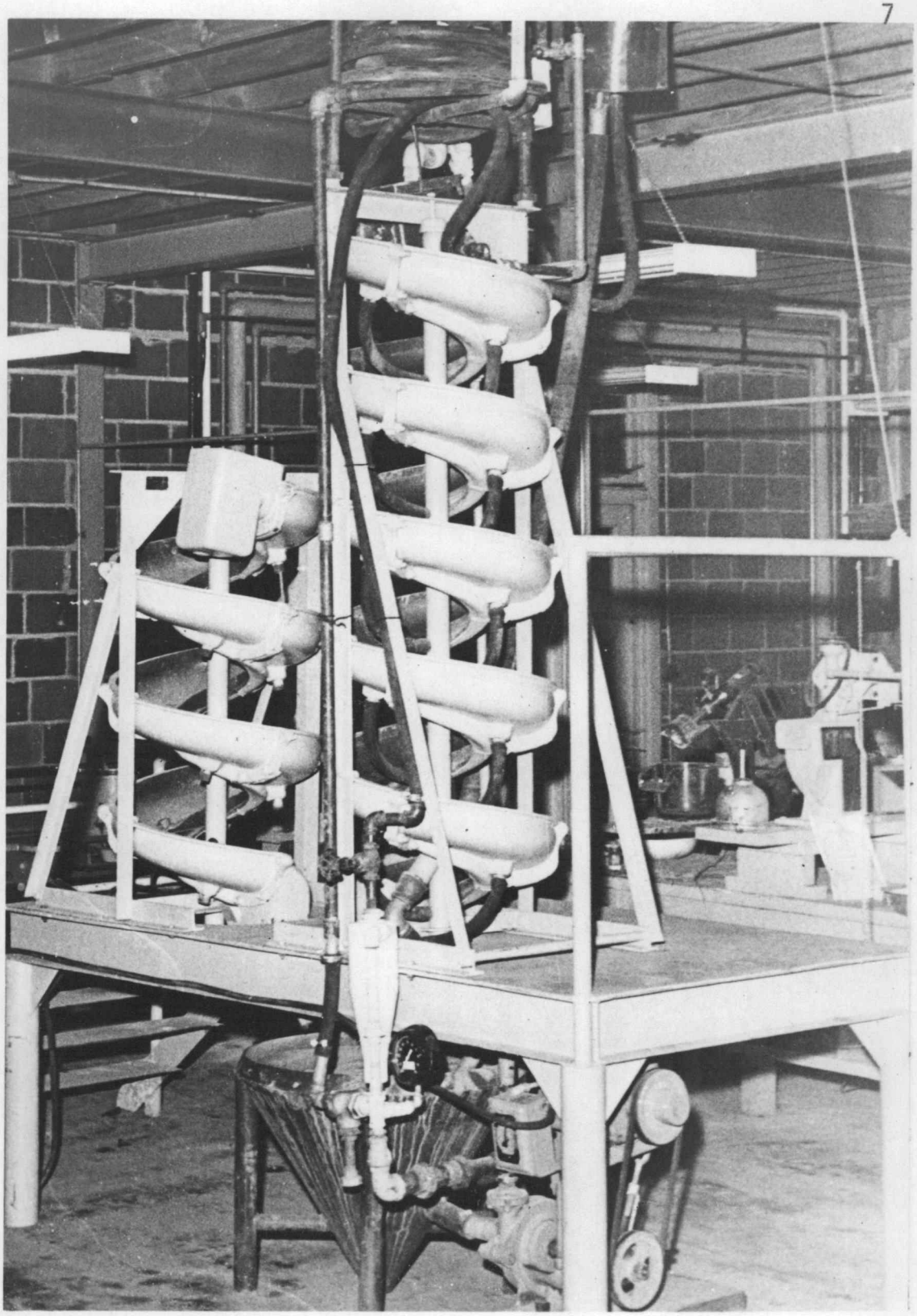

PLATE 1. Overview of the Humphreys Spiral.

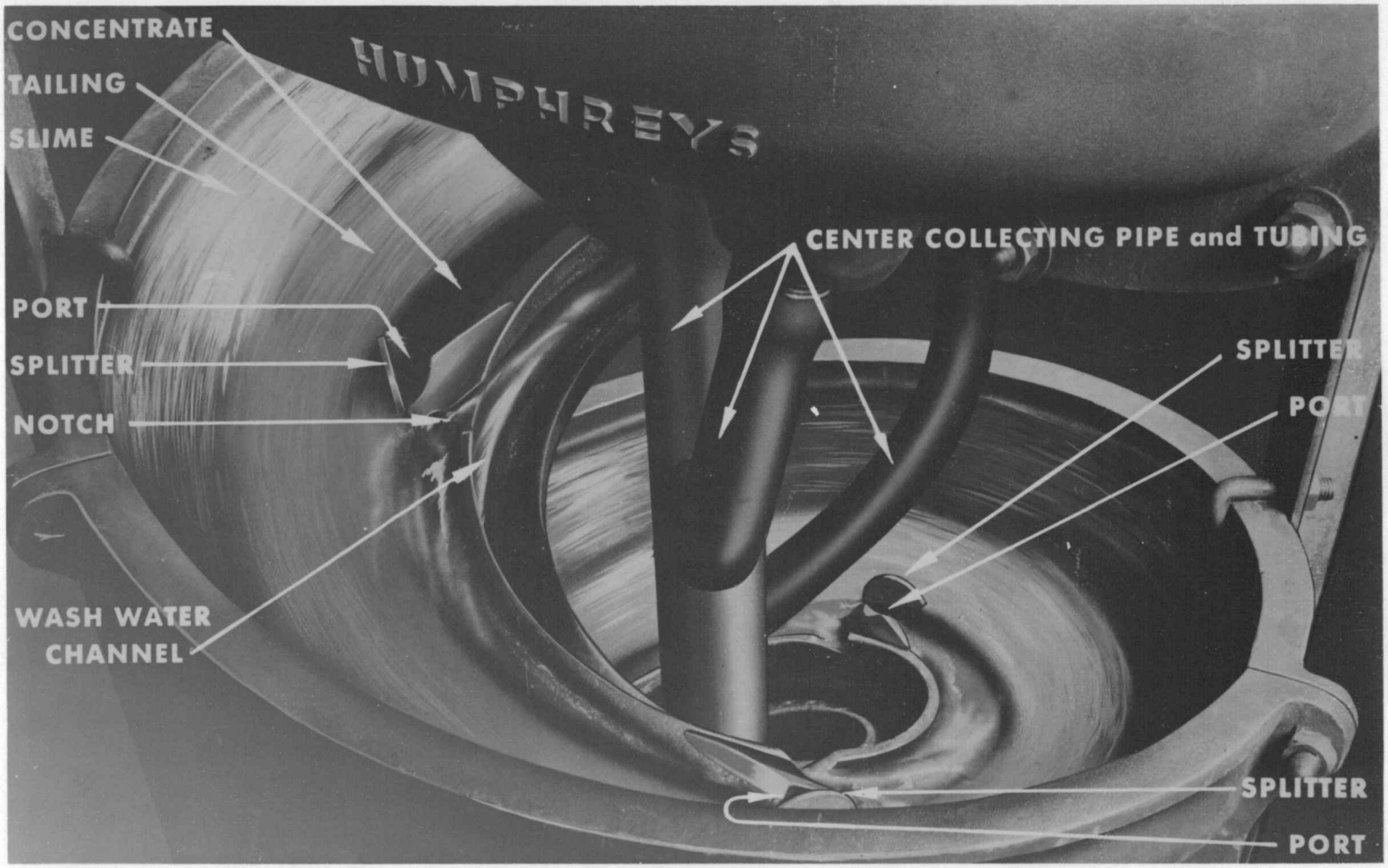

PLATE 2. Photograph illustrating concentrating action and position of products in the Spiral. Courtesy Humphreys Investment Company

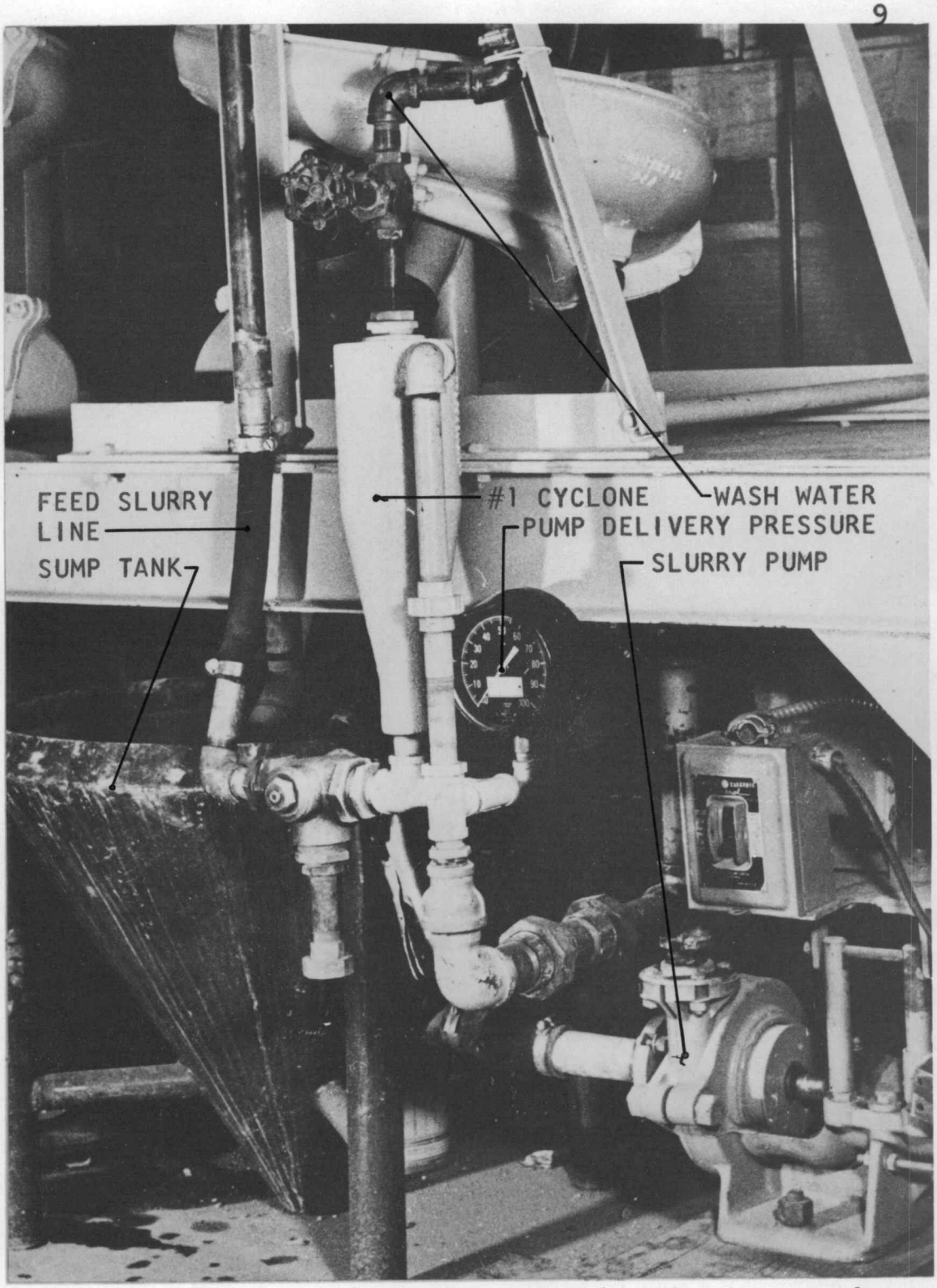

PLATE 3. Sump tank, slurry pump, and wash water cyclone details.

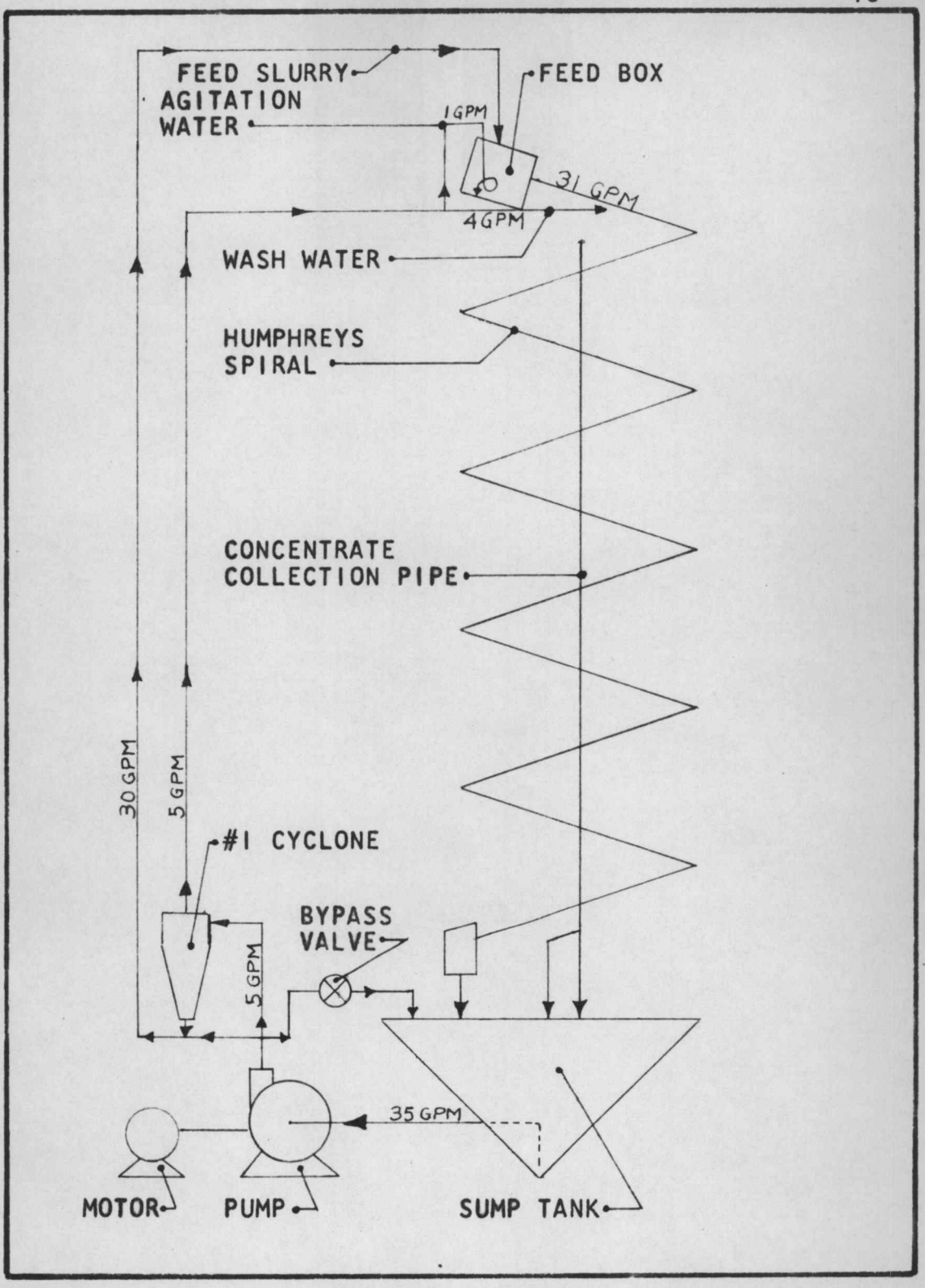

FIGURE 2: FLOW DIAGRAM, HUMPHREYS SPIRAL TEST CIRCUIT.

THEORY OF OPERATION

When a slurry of sand and water flows down a spiral channel, each element is subjected to a centrifugal force. This force is directly proportional to the square of the velocity of the flow and inversely proportional to the radius at which the element is located. This centrifugal force piles the slurry up on the outer rim of the spiral until an equilibrium between centrifugal force outward and gravitational force downward is reached (7, p. 85).

The "fluid film" concept of hydrodynamics has established the presence of a stagnant layer of liquid on the bottom of any moving stream. In the Humphreys Spiral, velocity and centrifugal force are reduced in this stagnant layer and gravitational force is allowed to pull the heavier particles down toward the axis of the Spiral trough and thus toward the concentrate ports (7, p. 85). Therefore, a cross flow exists within the stream, the direction of the cross flow at the bottom of the stream being toward the concentrate ports, and the direction of the cross flow at the top of the stream being toward the outer edge of the spiral trough. The upper cross flow is supplemented by the addition of wash water to the top of the stream while the lower cross flow is depleted by removal of the concentrate. Thus the over-all volume of the stream remains fairly constant.

The wash water tends to carry the lighter particles of sand toward the outer edge of the stream. An idealized view of the cross flow is shown in Figure 3, page 13.

A concentration of heavy material may be effected by removal of the portion of the stream flow within which the cross flow has concentrated the heavier elements of the slurry.

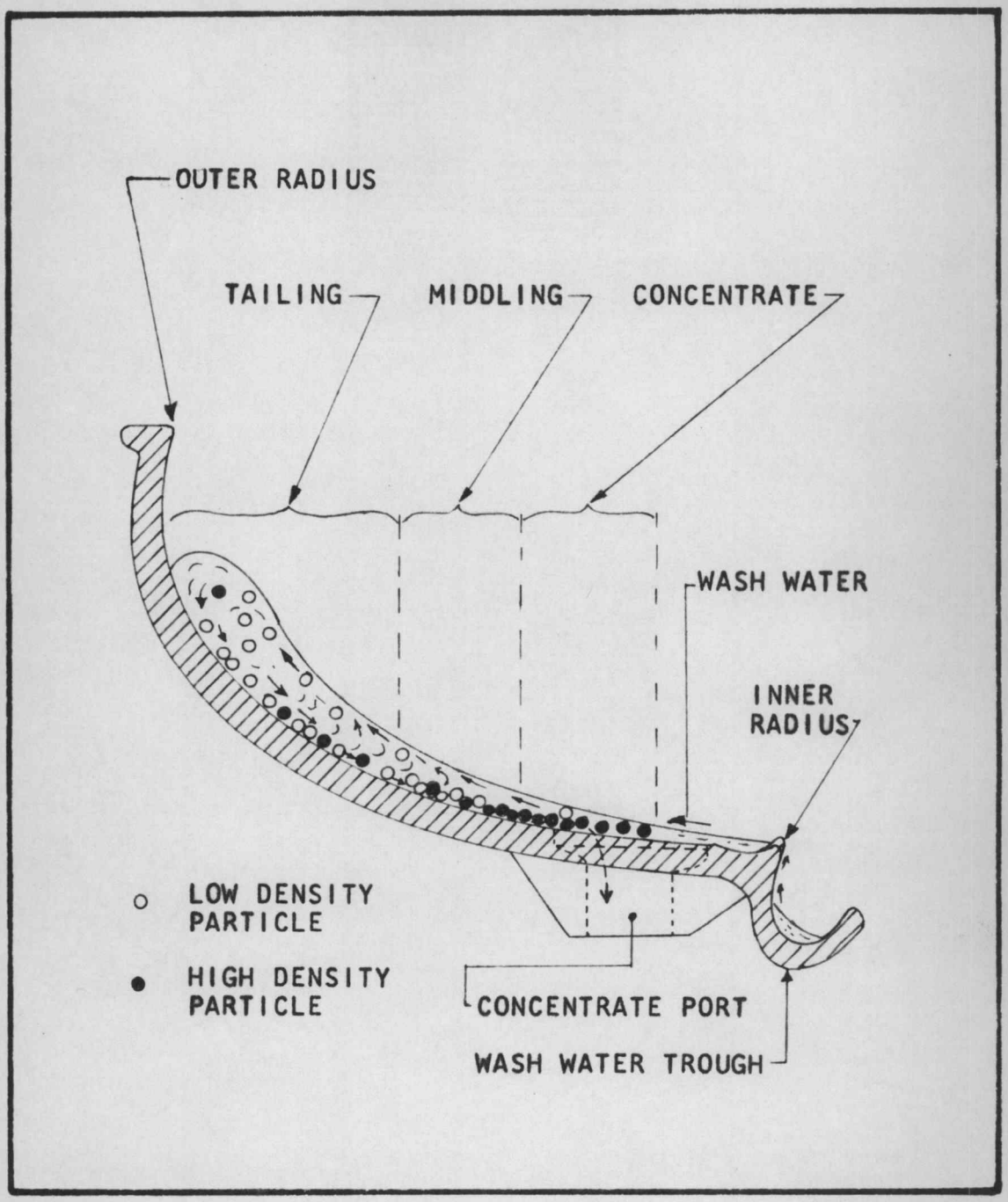

FIGURE 3: CROSS SECTION OF SPIRAL STREAM SHOWING RELATIVE POSITION OF SPIRAL PRODUCTS.

PHYSICAL CHARACTERISTICS AND MINERAL COMPOSITION OF ORE TESTED

The ore deposits of the Musick Mine are hydrothermal open space fillings with subsequent replacement deposition. The veins are found in altered andesite of probable Miocene age (4, p. 40). Pyritization is the most common alteration of the wall rock.

Samples of the ore were examined in hand specimen, reflecting section, and reflecting briquette.

The hand specimen shown in Plate 4a, page 16, shows the typical sharp delineation between the vein filling and the wall rock. Alteration of the wall rock by pyrite replacement is also visible in the photograph. In Plate 4b, page 16, replacement of an included fragment of wall rock by chalcopyrite is clearly visible. Plate 5b, page 17, is a photograph of the characteristic open space filling.

From examination of the reflecting sections, it appears that a multi-stage open space filling has taken place. The apparent sequence of filling is as follows: Stage one is partial open space filling of galena and pyrite followed by pyrite and sphalerite, with deposition of quartz completing the first filling. Subsequent to this deposition, there was extensive brecciation of the first stage filling. The interstices later were filled

with sphalerite and chalcopyrite and then by quartz. Following this second filling, there was apparently extensive replacement of included wall rock fragments by chalcopyrite (Plate 4b, page 16). The deposit shows definite evidence of both open space filling and replacement.

The ore tested was a sample of mine run ore mined in 1950 by Taylor and Watkins. It has been stored in a bin at the Musick Mine from 1950 to 1959. This material was subject to minor oxidation.

A representative sample of the ore was crushed to -10 mesh and divided into six fractions between 10 mesh and 100 mesh. The division was accomplished by dry screening. These fractions were bonded into briquettes utilizing polyester resin as the bonding agent (Plate 5a, page 17). The briquettes were polished for examination by reflected polarized light. From examination of the briquettes, it was determined that liberation of the sulphides from the gangue material is essentially complete at a grind size of 65 mesh.

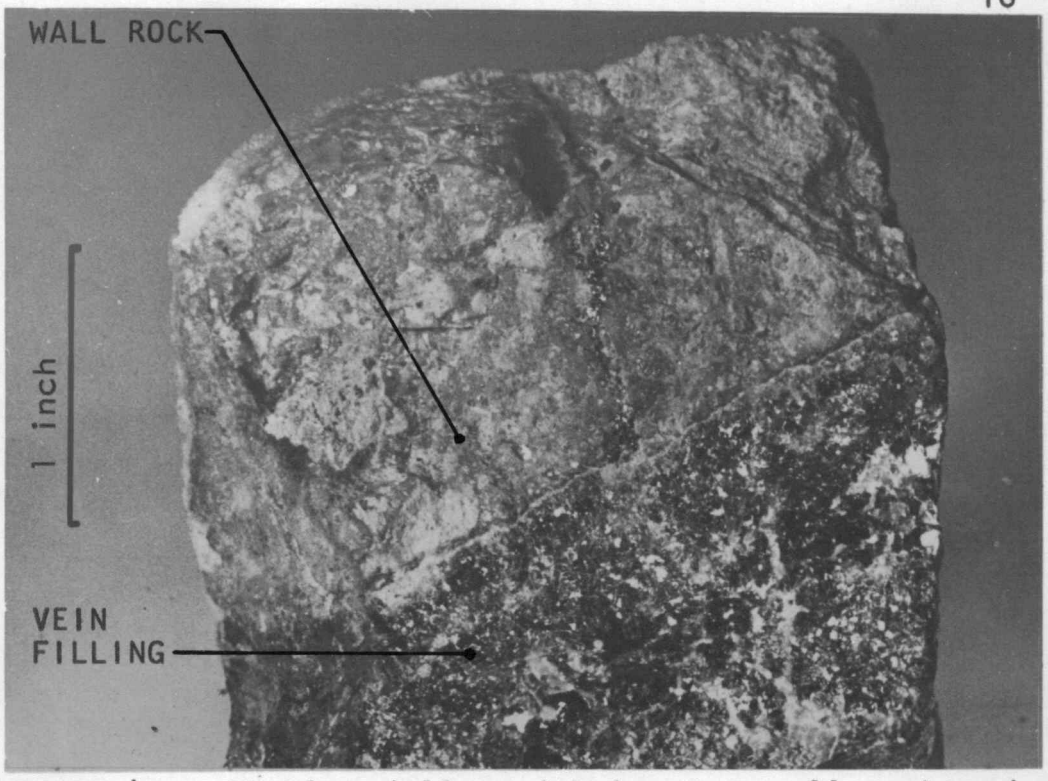

PLATE 4a. Showing delineation between wall rock and vein filling.

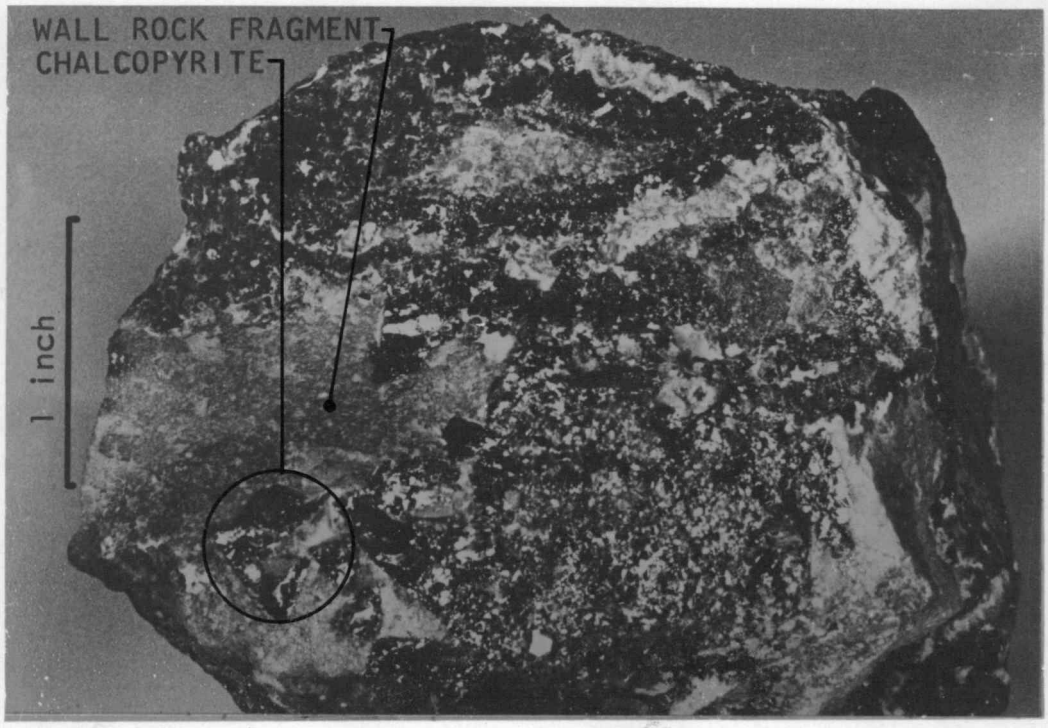

PLATE 4b. Replacement of an included fragment of wall rock by chalcopyrite

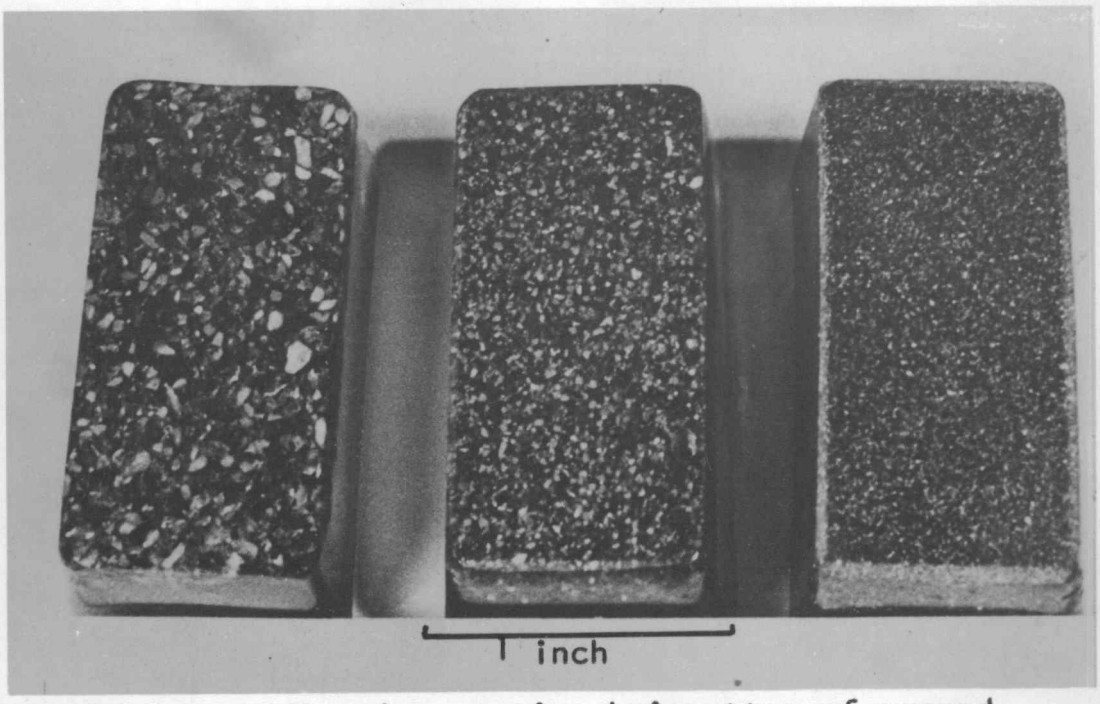

PLATE 5a. Reflecting section briquettes of ground and sized ore.

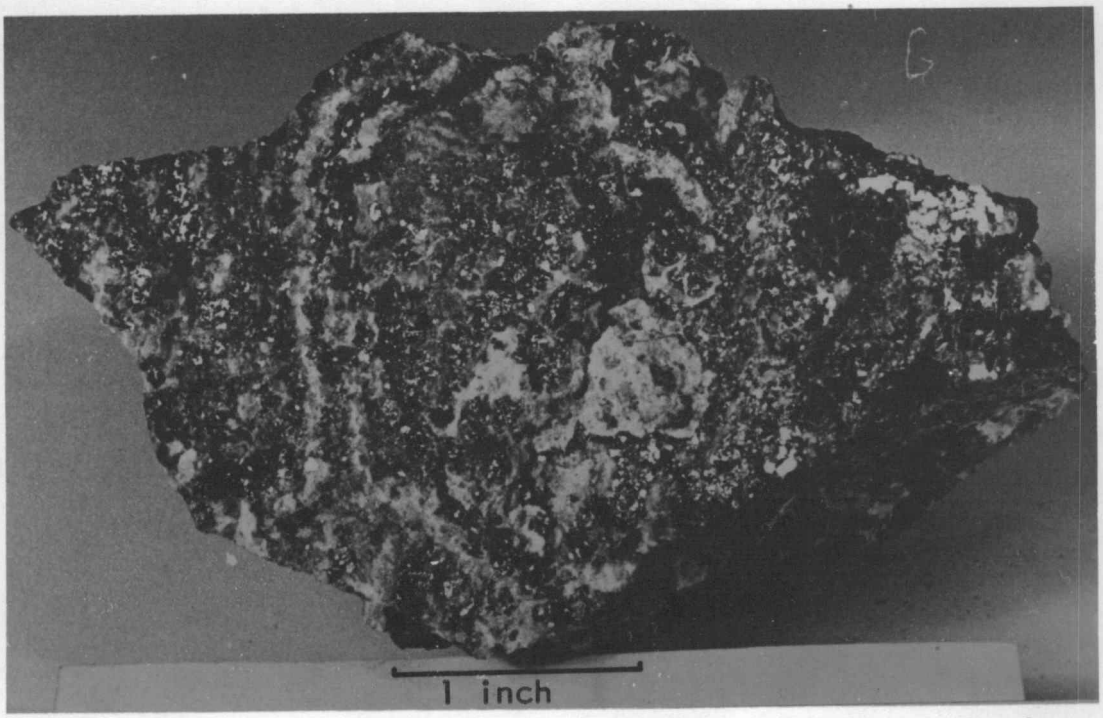

PLATE 5b. Complex open space filling, characteristic of the Musick ore.

CHEMICAL COMPOSITION OF THE ORE

A representative sample of the ore was prepared for analysis by standard laboratory crushing and grinding procedures. The results of the analysis are shown below in Table 1.

TABLE 1. Chemical Analysis

Pb	Cu	Zn	Au	Ag	Insoluble residue
4.5%	1.0%	6.3%	0.05oz.	1.3oz.	62%

A spectrographic analysis of material from the same stope, run by the U. S. Bureau of Mines in 1951, is shown below in Table 2.

TABLE 2. Spectrographic Analysis

Al	Cd	Ca	Cu	Pb	Mg	Ag	Zn	Mn	Mo	Ni	Si	Ti	Fe	V
C	E	C	D	C	D	F	C	E	F	F	A	E	B	E

Legend:
- A - More than 10%
- B - 5% to 10%
- C - 1% to 5%
- D - 0.1% to 1%
- E - 0.01% to 0.1%
- F - 0.001% to 0.01%
- G - Less than 0.001%

PREPARATION OF SAMPLES FOR SPIRAL SEPARATION

On inspection of the reflecting briquettes, it was apparent that the liberation grind for this ore was 65 mesh. Accordingly, the samples for spiral separation were prepared as stated below.

A 1000-pound sample of ore from the Musick Mine was crushed to pass through a 3/4 inch screen. This material was fed to a Hardinge Dry Ball Mill equipped with an air classifier. The air classifier was adjusted for a product size of 96 percent -65 mesh. The resulting -65 mesh product was further split into two fractions, 65 to 100 mesh and -100 mesh. This sizing was accomplished by dry screening with a Sweco Separator. A size analysis of the ground product is shown below in Table 3.

TABLE 3. Size Analysis

size, mesh	+65	-65 +100	-100 +140	-140 +200	-200 +325	-325
percentage	4	10	10	21	21	33

As may be seen, more than 50 percent of the ground material is finer than 200 mesh. The overgrind was due to the characteristics of the air classification system and ball charge used with the Hardinge Mill. Since the Humphreys Spiral does not work well with a feed which contains particles finer than 200 mesh, it is obvious

that this ball mill product was far from being an ideal feed for the Spiral. However, since no other grinding facilities were available, the tests were conducted on this product. It must be noted that the results of this investigation were seriously biased by the use of this grinding equipment. This grinding bias could probably be markedly reduced by the use of a closed circuit, wet ball mill, operating under conditions of high recirculating load with constant classification to remove the 65 to 100 mesh product as soon as it is formed. This technique should reduce the retention of 65-100 mesh material in the ball mill and lessen grinding to smaller sizes.

TECHNIQUE OF SPIRAL OPERATION

At the beginning of each test, the sump tank was partially filled with approximately 10 gallons of water. The pump was then started and the water circulated through the Spiral. A charge of ground ore was then slowly added to the sump tank. The weight of the charge necessary to attain a particular pulp density was estimated from a graph prepared from experimental data. (See Figure 4, page 24). The pulp density was then adjusted to the desired value by the addition of water or ground ore.

Flow rate was then set at the desired value by adjustment of the bypass valve. (See flow diagram Figure 2, page 10). The actual flow rate was determined initially by a series of timed tests of a measured volume as a function of the pump pressure. A graph of the relation of pump pressure to flow rate was prepared, and all subsequent flow-rate adjustments were made in terms of pump pressure. The pump pressure gauge may be seen in Plate 3, page 9, and the flow rate/pump pressure graph is presented in Figure 5, page 25.

The number of splitters used ranged from 5 to 15. Five splitters were used with 10 ports closed, when operating with course material at high flow rate, and 15 splitters with special extensions were used when operating with fine material at low flow rates.

The splitters were adjusted for visually determined optimum cut. The area of optimum cut is established from observation of the stream flow. An idealized illustration of the stream flow is shown in Figure 6, page 30.

The concentrates from all splitters were mixed in a delivery pipe and sampled as a bulk concentrate.

ANALYTIC METHODS

Analyses of the samples for copper, lead, and zinc, were done by conventional wet quantitive methods.

Analyses of the samples for gold and silver were done by conventional fire assaying.

More than 100 wet analysis of copper, lead, and zinc and 12 fire assays of gold and silver were completed. Analytical work consumed more than 70% of the total time spent on the project.

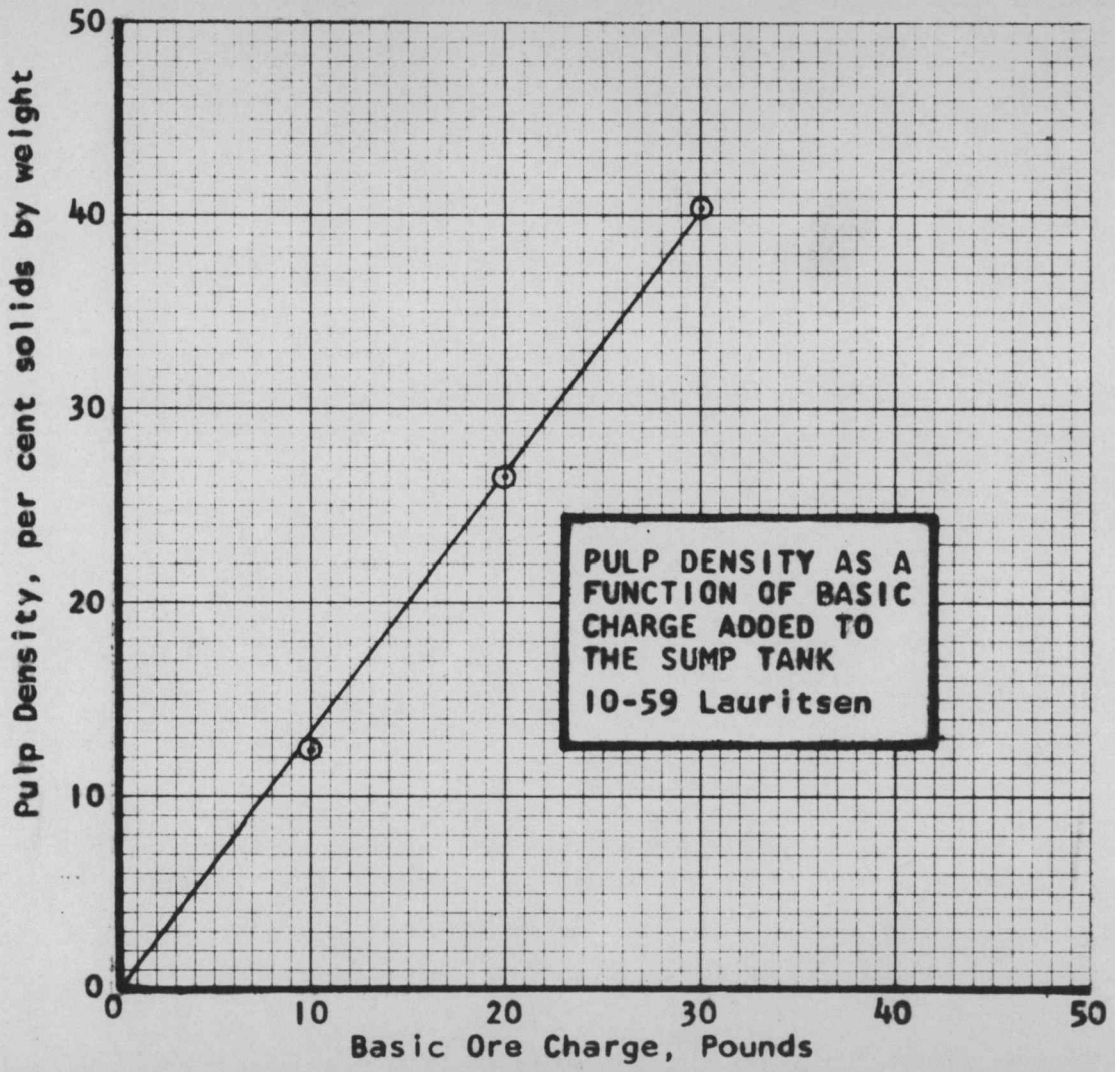
PULP DENSITY AS A FUNCTION OF BASIC CHARGE ADDED TO THE SUMP TANK
10-59 Lauritsen
Pulp Density, per cent solids by weight
Basic Ore Charge, Pounds
0
10
20
30
40
50

FIGURE 4

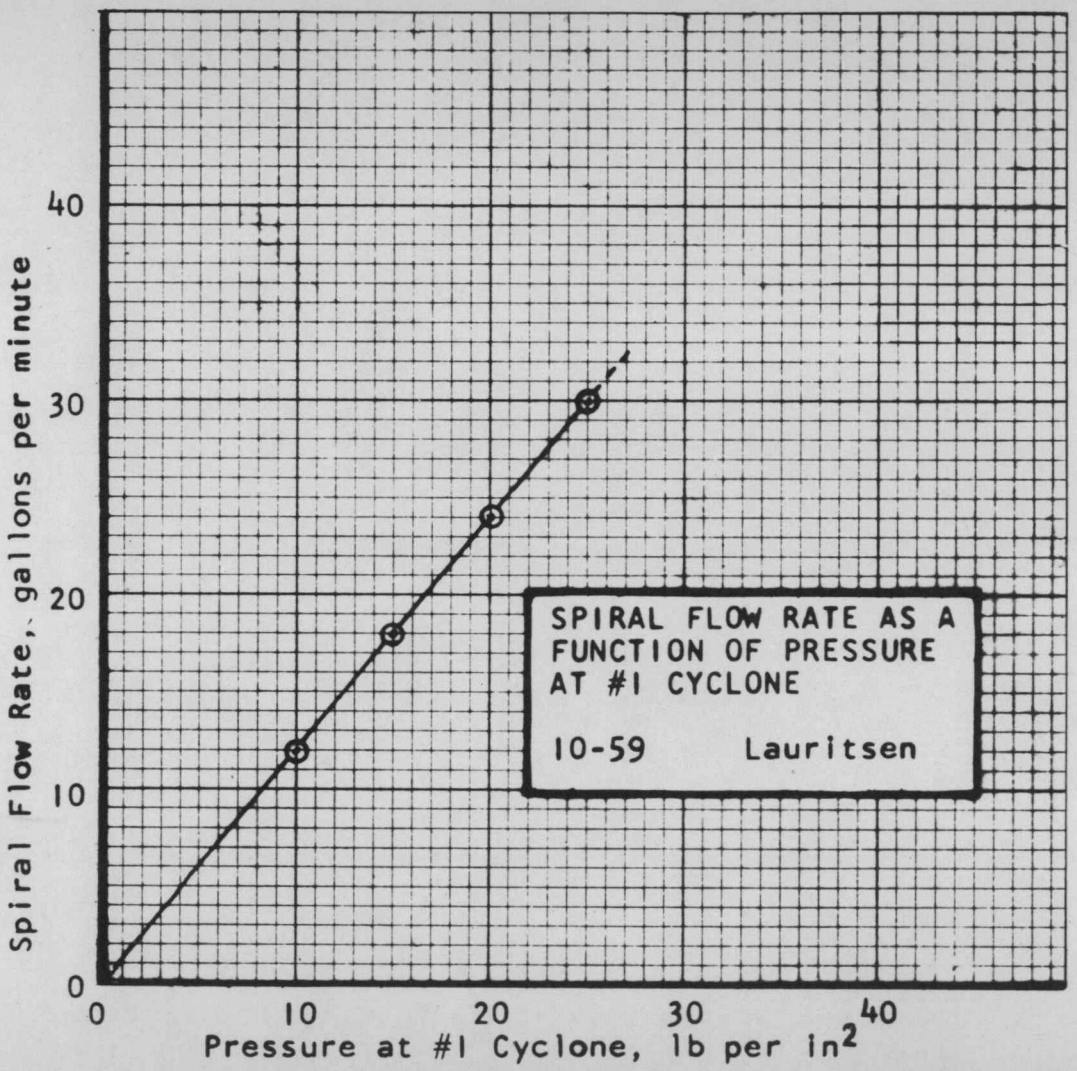
SPIRAL FLOW RATE AS A FUNCTION OF PRESSURE AT #1 CYCLONE
10-59 Lauritsen
Spiral Flow Rate, gallons per minute
0
10
20
30
40
Pressure at #1 Cyclone, lb per in^2
0
10
20
30
40

FIGURE 5

TEST PROGRAM

A two-phase test program was planned. Phase one was planned as a series of preliminary tests for the investigators to become familiar with the operational characteristics of the Humphreys Spiral and to evaluate the practical range of the variables. Since feed size was controlled, the variables to be evaluated in phase one, were pulp density, flow rate, and concentration ratio. A second-phase was planned as a series of follow up tests for optimum utilization of the data from phase one. Thus the testing done in phase two would be conducted in the optimum ranges of the variables as were determined in phase one.

PRELIMINARY TESTS TO EVALUATE THE EFFECT OF VARIOUS PULP DENSITYS, FLOW RATES, AND CONCENTRATION RATIOS

It was proposed to evaluate the effects of the following variables on spiral efficiency: pulp density, flow rate, and concentration ratio. This evaluation was accomplished as described below.

Flow rate in the spiral was initially established with a pulp density of 35% solids by weight. The flow rate was then set at the maximum rate attainable with the pump used. This flow rate was 30 gallons per minute. The splitters were adjusted for optimum cut, and timed samples were taken of the concentrate and the tailing. Pulverized ore and water equal to the amount removed in sampling were then added to the sump tank. This test was repeated at flow rates of 24, 18, and 12 gallons per minute.

The entire series of tests was conducted twice, first on the 65 to 100 mesh fraction of the ore, and second on the -100 mesh fraction of the ore.

At this point in the test program, analyses were made of the samples taken, and the results were evaluated.

Three very significant results were noted in this primary evaluation. They are summarized as follows:

1. Pulp density is critical at 20% solids.

2. On the 65 to 100 mesh fraction, recovery increases with flow rate up to the limit of the pump capacity.
3. On the -100 mesh fraction, recovery is maximum at the minimum flow that can be maintained in the spiral while maintaining enough pressure to operate the wash water cyclone.

Figure 7, page 31, shows the typical relation of recovery of ore to the pulp density of the spiral feed. It may be seen in this graph that only a pulp density in the immediate vicinity of 20% solids will yield acceptable recovery.

Figures 8 and 9, page 32, clearly show that the optimum operating conditions with regard to flow rate are beyond the operating range of the test spiral system used.

The implications of these factors are profound. Since the recovery is so severely affected by the pulp density, the optimum pulp density may be considered relatively constant at 20% solids. Also, since insofar as flow rate is concerned, the operating limits of the test machinery preclude optimum recovery, the optimum flow rate for recovery of the 65 to 100 mesh fraction of the ore may be considered a constant at 30 gallons per minute, and the optimum flow rate for the -100 mesh fraction of the ore may be considered a constant at 12 gallons per minute.

It has been observed then, that for all practical purposes three of the four variables have become constant. The remaining variable is concentration ratio.

Based on the above observations from phase one (the preliminary tests), the phase two or final tests, were planned to evaluate the effect of concentration ratio on recovery of values.

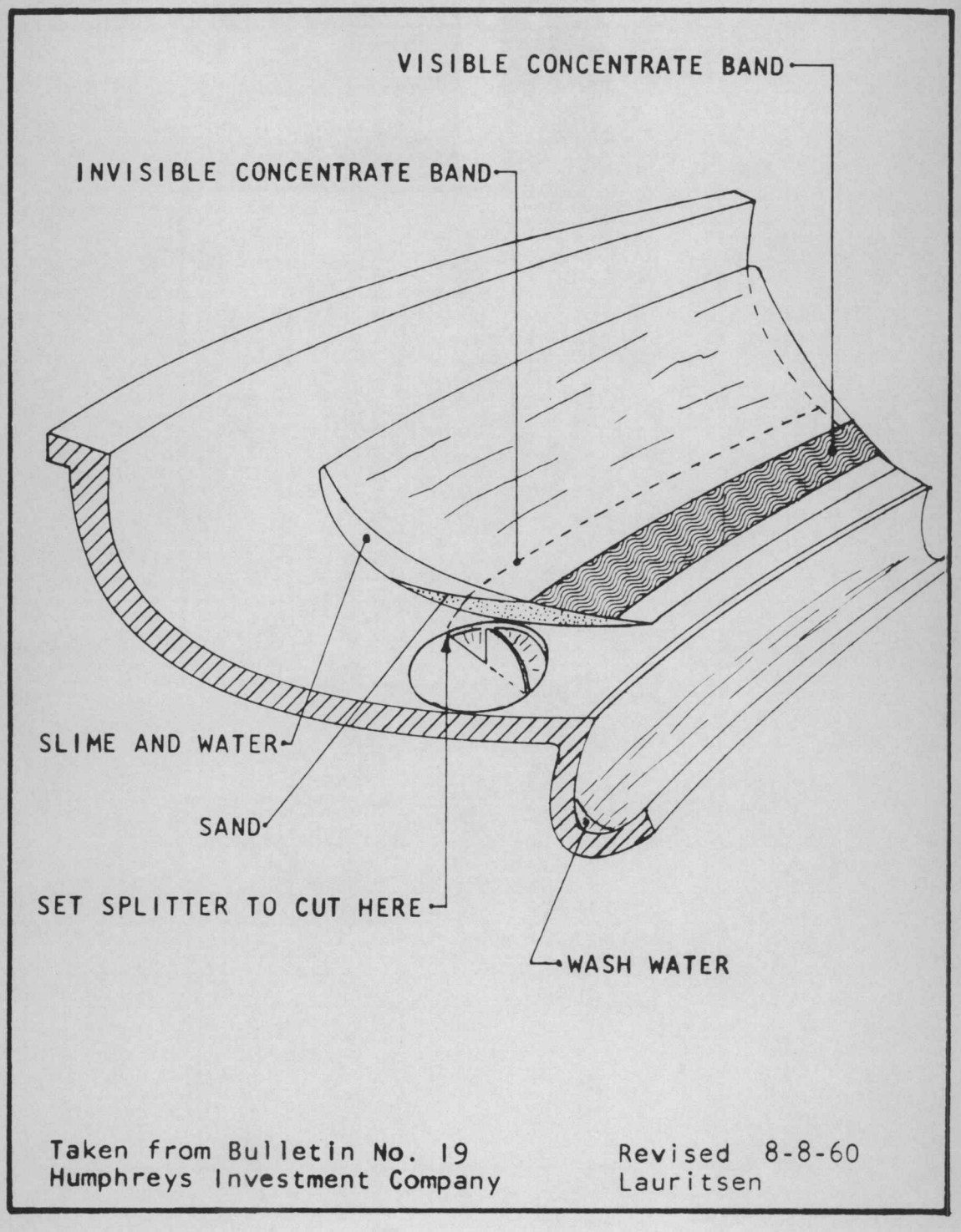

FIGURE 6: Spiral flow cross section.

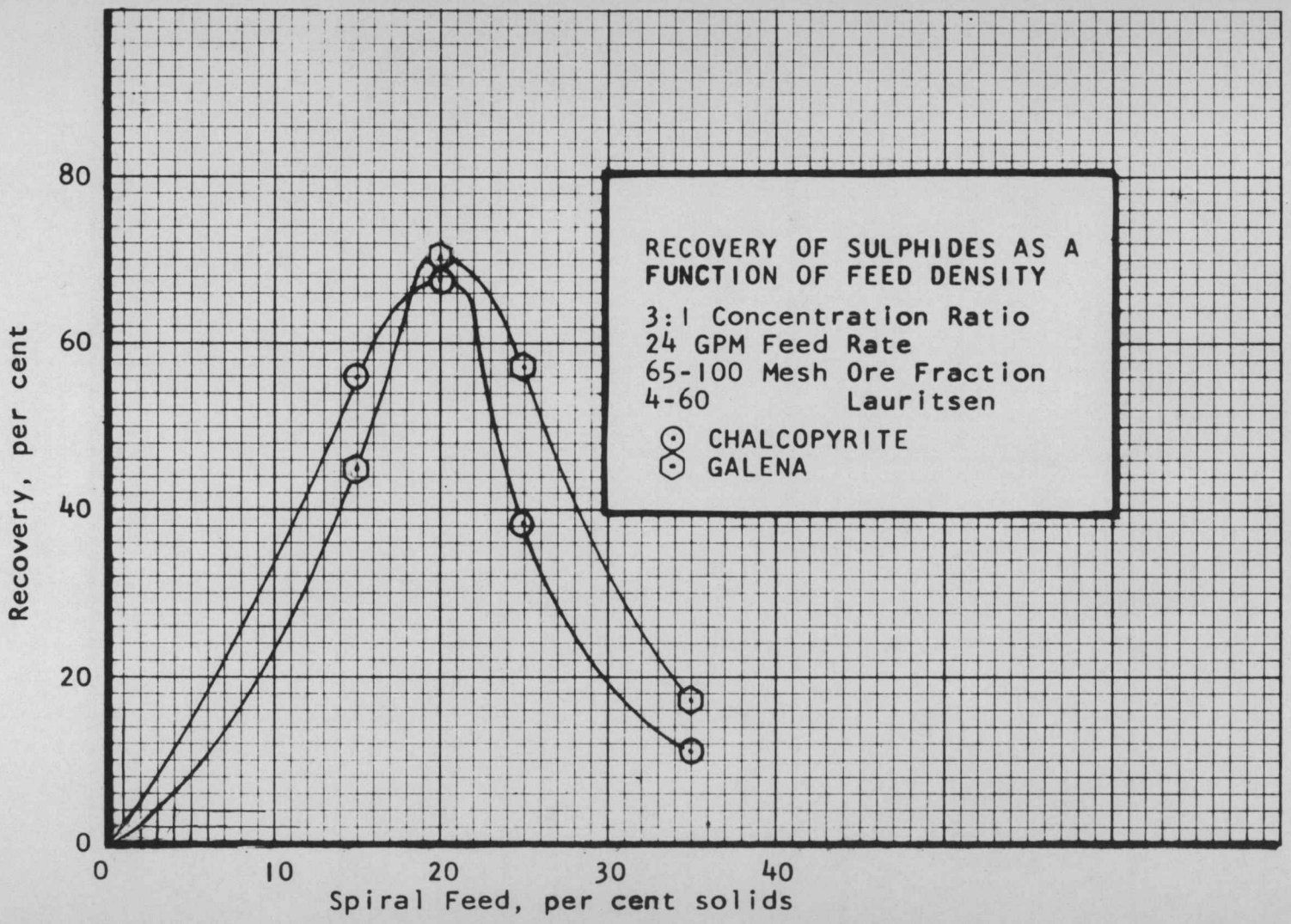
RECOVERY OF SULPHIDES AS A FUNCTION OF FEED DENSITY
3:1 Concentration Ratio
24 GPM Feed Rate
65-100 Mesh Ore Fraction
4-60 Lauritsen
CHALCOPYRITE
GALENA
Recovery, per cent
0
20
40
60
80
0
10
20
30
40
Spiral Feed, per cent solids

FIGURE 7

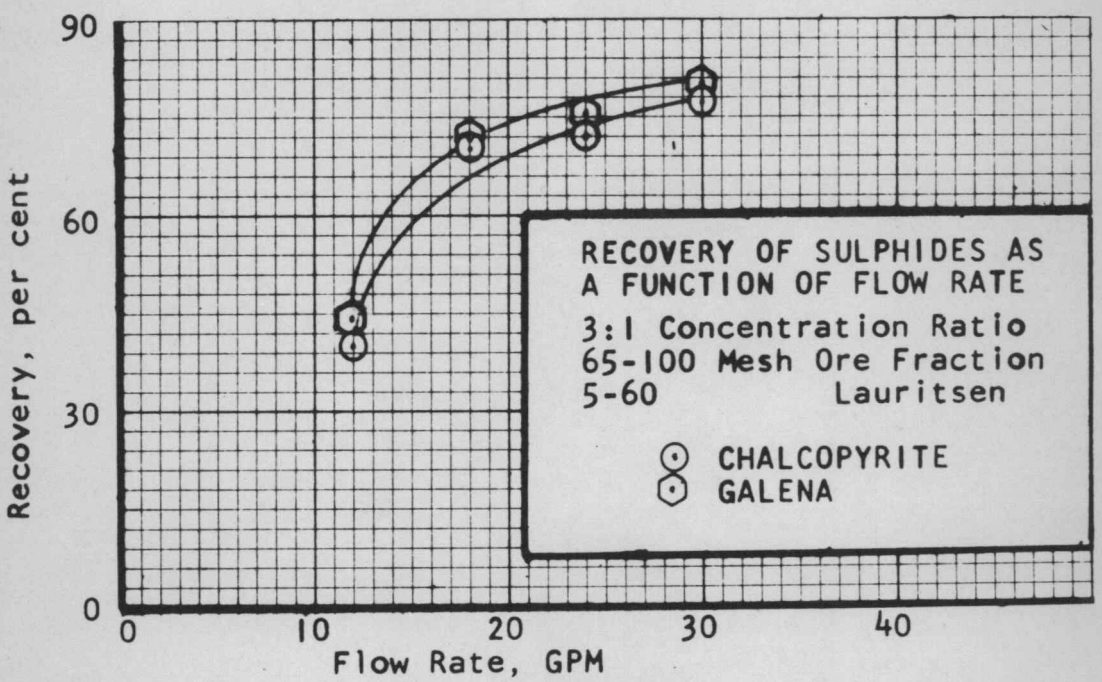

FIGURE 8

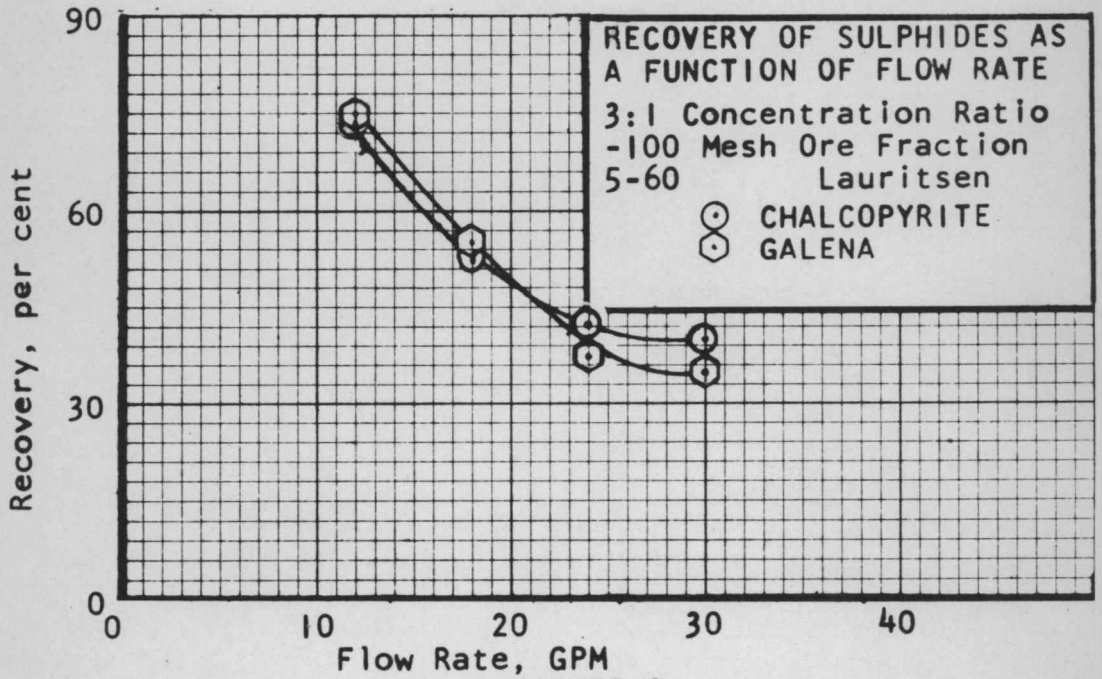

FIGURE 9

FINAL TESTS TO DETERMINE THE EFFECT OF CONCENTRATION RATIO ON RECOVERY

The final tests were set up in the following manner.

Flow rate in the Spiral was established with a pulp density of 20% solids by weight, composed of water and an ore charge of the 65 to 100 mesh fraction. The flow rate was then set at 30 gallons per minute. Three tests were made: first, with a tailing reject of 25% by volume of the total spiral feed; second, with a tailing reject of 50% by volume of the total spiral feed; and third, with a tailing reject of 75% by volume of the total spiral feed.

A similar series of three tests was run with the -100 mesh ore fraction. In this case the pulp density was 20% solids by weight and the flow rate was 12 gallons per minute. The tailing reject percentages were again 25%, 50%, and 75% by volume.

Concentration ratio is defined as the relation of total feed to amount retained. However, since the amount of the feed which is discarded (the tailing) is not subject to further transportation costs, it becomes economically significant and a very useful mode of evaluating the upgrading process. Therefore, the results of these tests are presented as a function of the percent of total spiral feed rejected as tailing.

Figures 10, 11, and 12, pages 35, 36, and 37, show the recovery of the base metals present in the spiral feed as a function of the rejected material (tailing). As only trace amounts of gold and silver were found in the tailing, the recovery is assumed to exceed 90%.

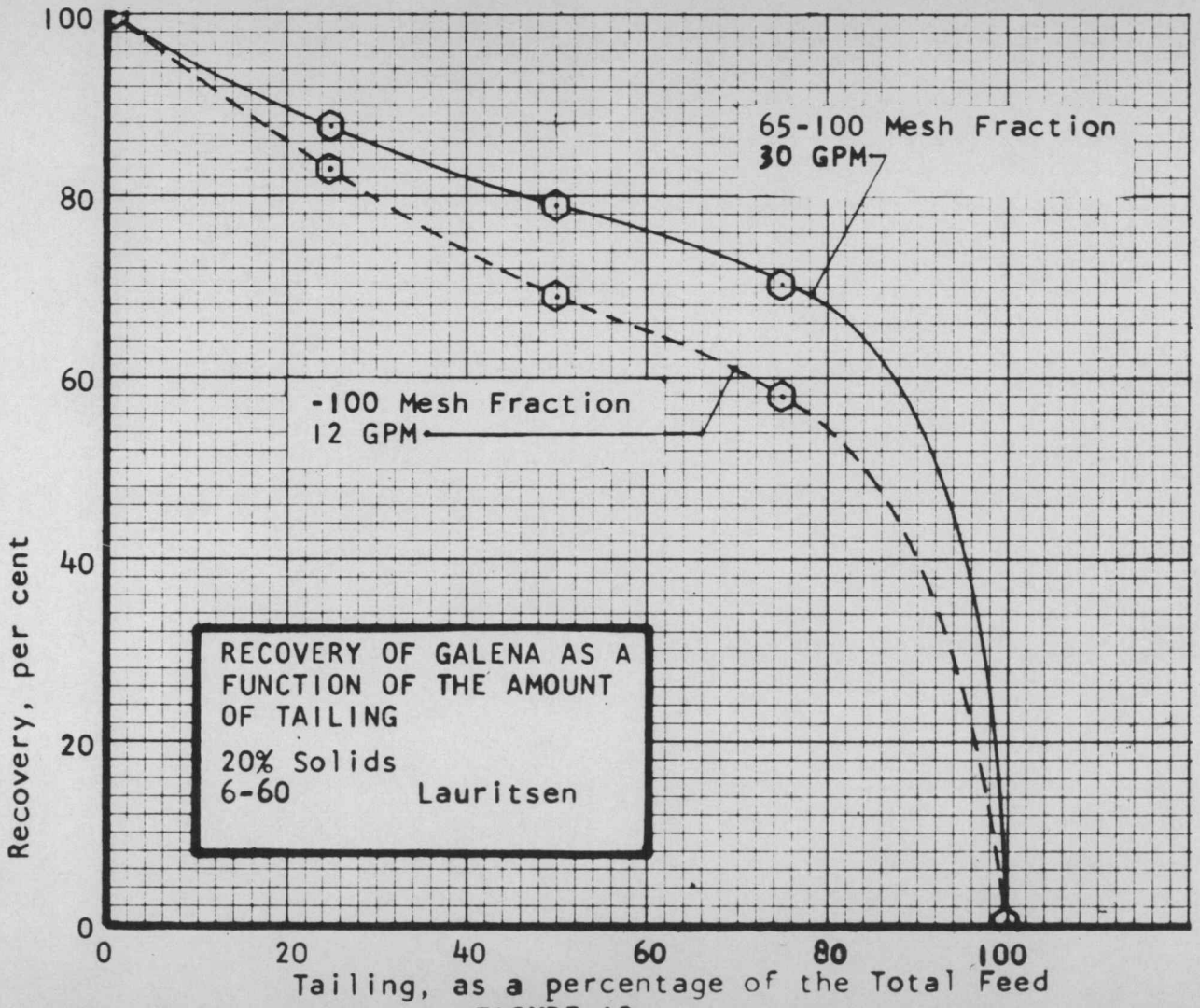
100
80
60
40
20
0
Recovery, per cent
0
20
40
60
80
100
Tailing, as a percentage of the Total Feed
65-100 Mesh Fraction
30 GPM
-100 Mesh Fraction
12 GPM
RECOVERY OF GALENA AS A
FUNCTION OF THE AMOUNT
OF TAILING
20% Solids
6-60 Lauritsen

FIGURE 10

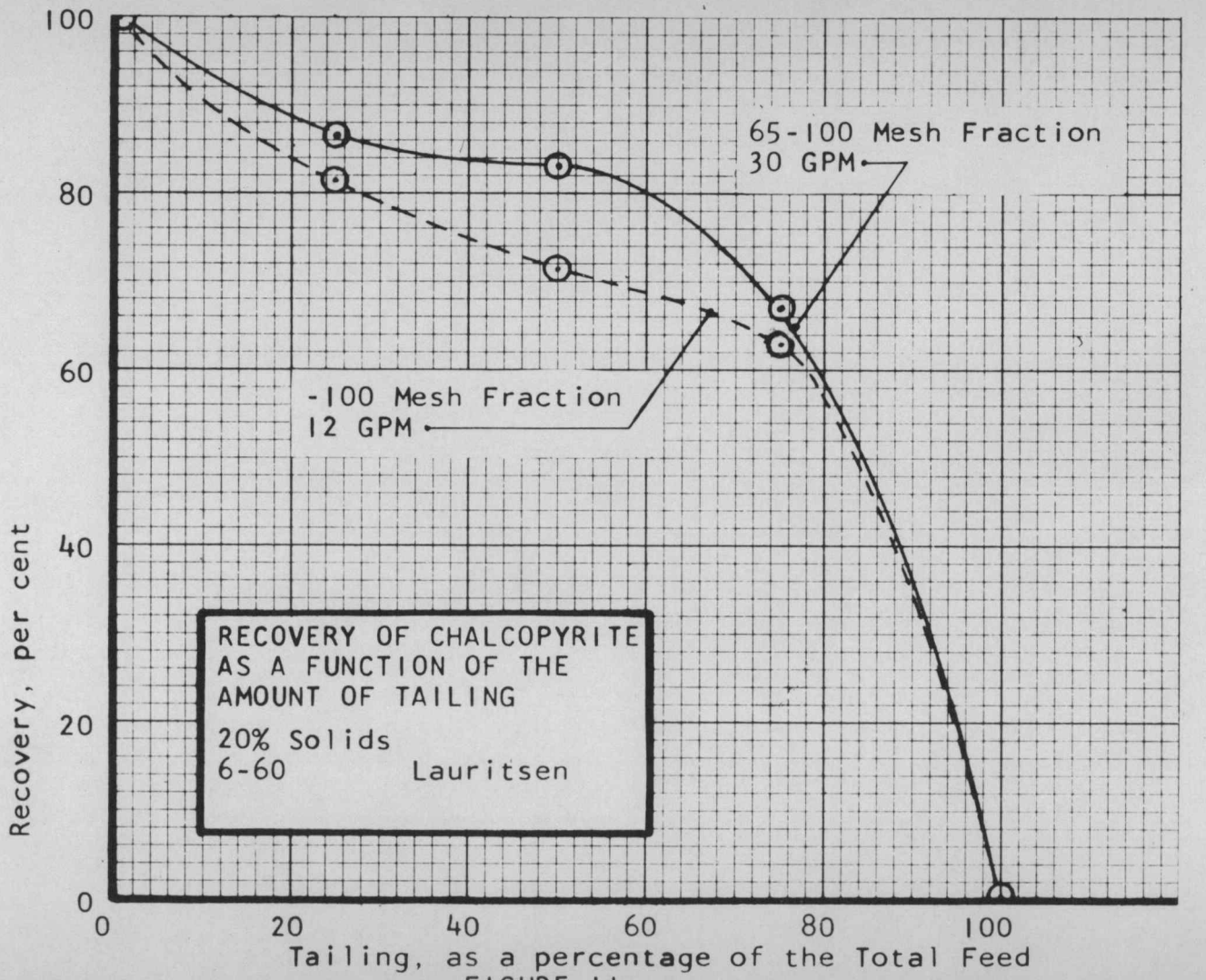
100
80
60
40
20
0
Recovery, per cent
0
20
40
60
80
100
Tailing, as a percentage of the Total Feed
65-100 Mesh Fraction
30 GPM
-100 Mesh Fraction
12 GPM
RECOVERY OF CHALCOPYRITE
AS A FUNCTION OF THE
AMOUNT OF TAILING
20% Solids
6-60 Lauritsen

FIGURE 11

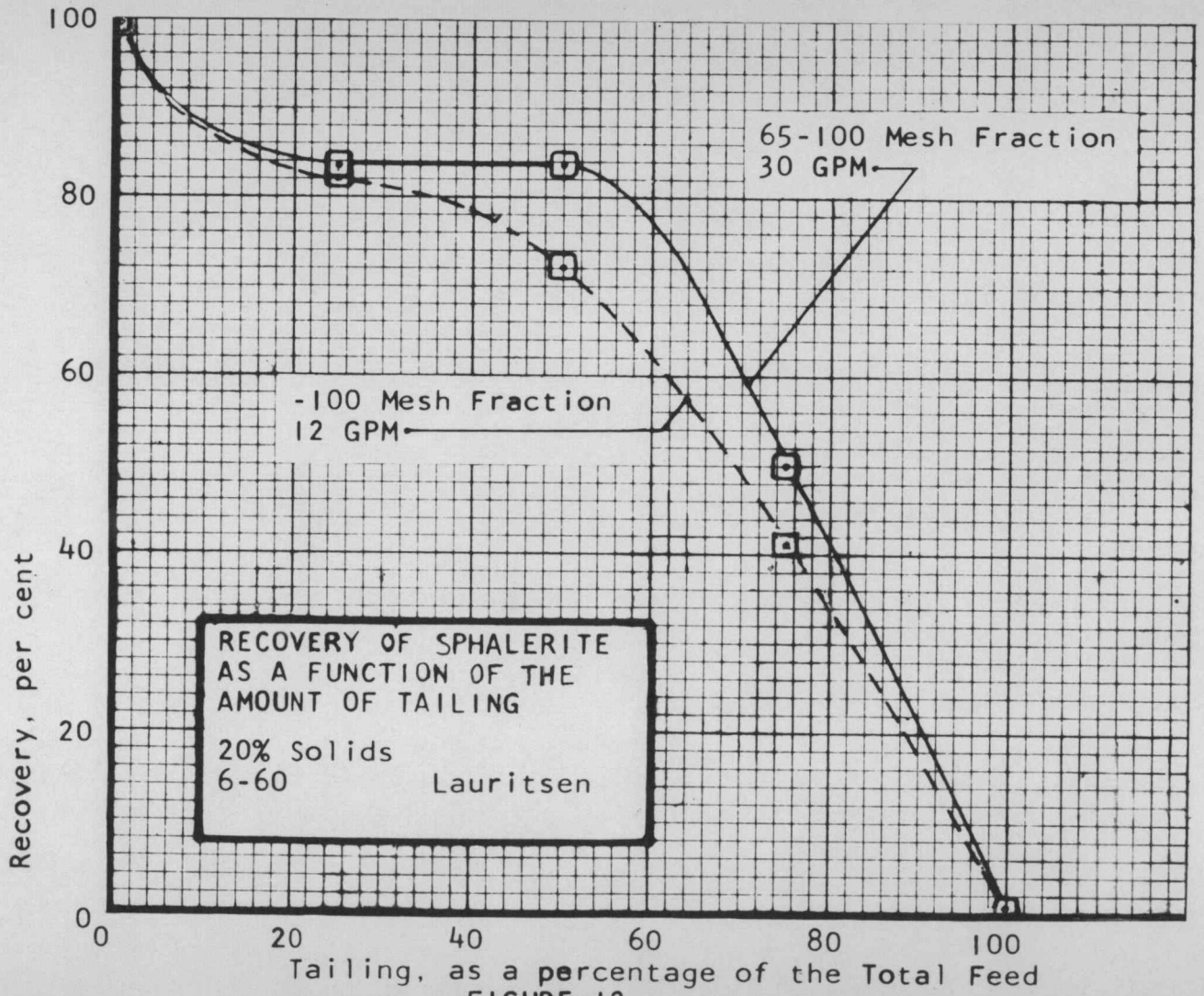
100
80
60
40
20
0
0
20
40
60
80
100
Recovery, per cent
Tailing, as a percentage of the Total Feed
65-100 Mesh Fraction
30 GPM
-100 Mesh Fraction
12 GPM
RECOVERY OF SPHALERITE
AS A FUNCTION OF THE
AMOUNT OF TAILING
20% Solids
6-60
Lauritsen

FIGURE 12

ECONOMIC EVALUATION

The prices of metals (except gold), fluctuate from time to time as do the purchasing policies of the smelters. For purposes of evaluation, the following arbitrary values (based on average prices from 1955 to 1960) were assigned to the metals involved: Copper, 30¢ per pound; Lead, 15¢ per pound; Zinc, 10¢ per pound; Gold, $35.00 per ounce; and Silver, 90¢ per ounce. The actual value of the material at any specific time must be computed in accordance with prices and smelter purchasing policies as of that date.

Figure 13, page 40, shows the relationships between the amount of concentrate, the value of the concentrate, and the recovery efficiency as a function of the amount of total material rejected as a tailing.

As may be observed, the amount of concentrate decreases and the concentrate value increases when more material is rejected as tailing. The recovery efficiency decreases very rapidly above 75% reject of the total feed. Final choice of operating conditions is contingent on an economic balance between mining costs, transportation costs, and recovery efficiency.

Typical operating conditions which might be experienced at the Musick Mine are at 50% reject of tailing

material. Under these conditions, transportation costs are reduced by 50% and value of the material is increased from $35.02 per ton to $51.44 per ton. This is accomplished at a loss of 27% of the mined ore value. Another way of expressing this is that one ton of concentrate transported to the smelter will bring the same pay as two tons of the ore would, less of course, the 27% concentrating loss. Since optimum recovery was not possible with equipment available, losses should be lower in actual plant operation.

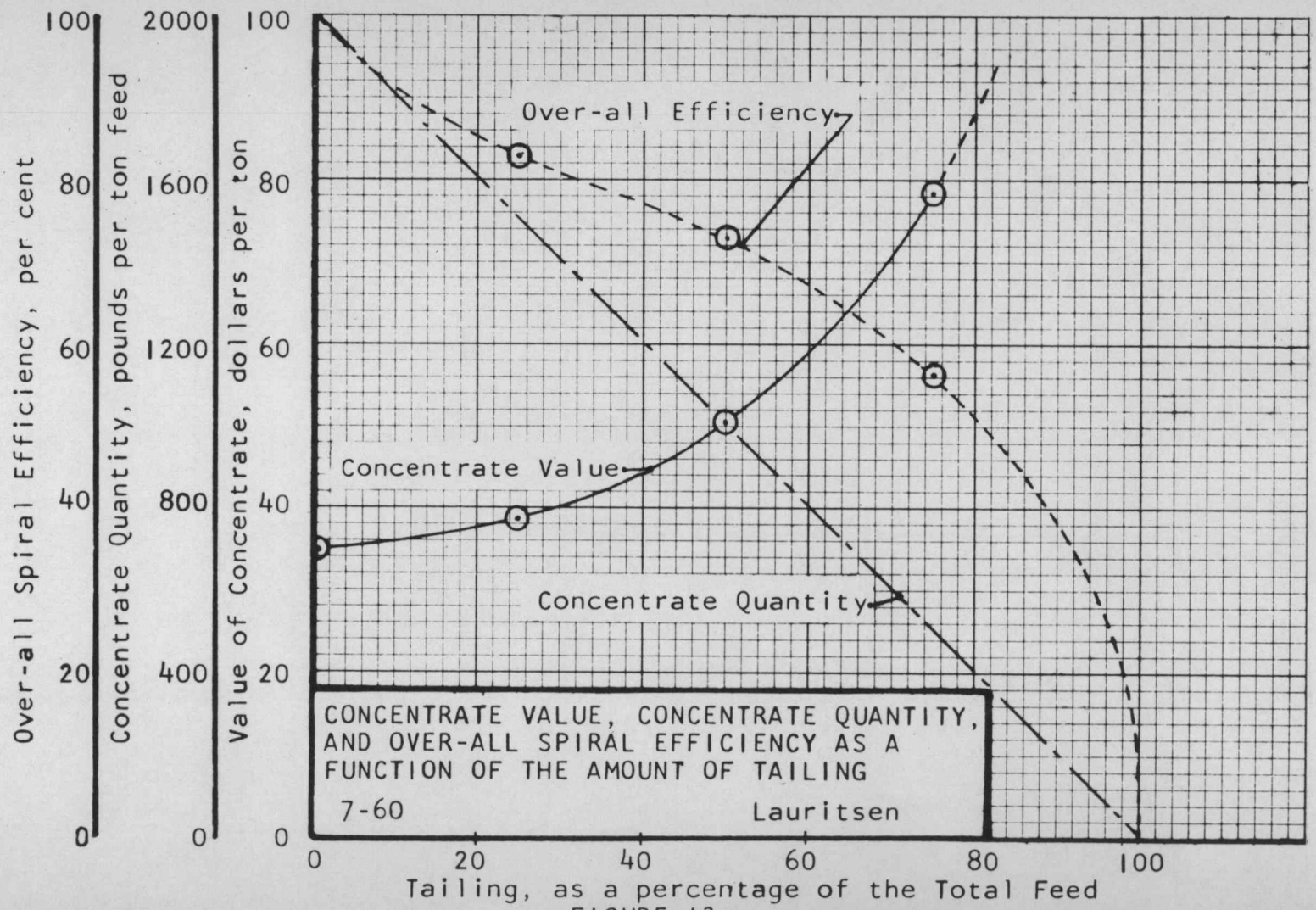
Over-all Spiral Efficiency, per cent
Concentrate Quantity, pounds per ton feed
Value of Concentrate, dollars per ton
100
80
60
40
20
0
2000
1600
1200
800
400
0
100
80
60
40
20
0
Over-all Efficiency
Concentrate Value
Concentrate Quantity
CONCENTRATE VALUE, CONCENTRATE QUANTITY, AND OVER-ALL SPIRAL EFFICIENCY AS A FUNCTION OF THE AMOUNT OF TAILING
7-60
Lauritsen
0
20
40
60
80
100
Tailing, as a percentage of the Total Feed

FIGURE 13

CONCLUSIONS AND RECOMMENDATIONS

The results of the investigation of the application of the Humphreys Spiral to the sulphide ore of the Musick Mine lead to the following conclusions:

I. The Spiral can be used to upgrade the sulphide ore of the Musick Mine.

   A. As the concentrate value increases, cost of transportation of the concentrate decrease and the Spiral recovery (efficiency) decreases.

   B. The economic feasibility may be determined by establishing the optimum balance between mining costs, Spiral efficiency, and transportation costs.

II. Operating techniques for this type of ore have herein been presented.

   A. The optimum number of splitters, their settings, and their type vary with the feed size and flow rate.

   B. A slurry density of 20 percent solids by weight yields optimum recovery.

   C. Optimum flow rate varies with the particle size of the feed and could not be attained with the system used.

   D. Beneficiation of slimes, (material of the 100 to 400 mesh size) is possible at low flow rates

with modified splitters.

1. Recovery of the total feed is lower on the 100 to 400 mesh slime fraction than on the 65 to 100 mesh fraction.
2. Nearly nine-tenths of the ore prepared was in the 100 to 400 mesh size range due to the grinding technique used.

E. Splitters must be set to take a wider cut than appears necessary by examination of the stream, in order to attain a reasonable recovery efficiency.

F. Conversely, while reducing the width of the cut increased the concentrate value and reduced the amount of concentrate which would have to be transported, it also reduced the efficiency of the system.

G. The grinding method used in preparation of the ore is a critical factor. The grinding method used was deleterious to attaining feasible results.

H. The pump used in the system was not adequate for this investigation.

1. Maximum flow rate was not sufficient to attain optimum flow rate for the 65 to 100 mesh fraction.

2. The minimum flow rate was limited by the necessity of retaining pressure to operate the wash water cyclone.

III. On the basis of the results of this investigation, it is recommended that:

A. An investigation be conducted to determine optimum grinding methods for producing a Spiral feed in the range of 65 to 100 mesh.

1. Utilizing a hammermill instead of a jaw crusher.

2. Utilizing a wet ball mill with cyclone classification and a high rate of load recirculation.

B. That the pump circuit be modified to permit flow rates up to 50 gallons per minute.

C. That recovery efficiency be re-evaluated when conditions A and B above have been satisfied.

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APPENDIX I

Tabulation of Data on the Relation of Sulphide Recovery To Pulp Density

Sample Number	Flow Rate GPM	Feed Density % Solids	Recovery of Galena, %	Recovery of Chalcopy-rite, %
3	24	15	44.1	55.7
27	24	20	70.1	67.5
6	24	25	56.8	38.1
10	24	35	17.2	10.5

APPENDIX II

Tabulation of Data On The Recovery of Sulphides As A Function Of Flow Rate

Flow Rate GPM	Recovery of 65-100 Mesh Fraction, %		Recovery of Minus 100 Mesh Fraction, %	
	Chalcopyrite	Galena	Chalcopyrite	Galena
30	86.1	87.8	44.2	39.8
24	80.0	84.1	46.8	41.2
18	77.3	78.4	58.3	61.0
12	44.2	48.1	81.5	83.0

APPENDIX III

Tabulation Of Data On The Relation Of Sulphide Recovery As A Function Of Amount Of Tailing

DATA FOR GALENA, CHALCOPYRITE, AND SPHALERITE

Recovery at 20% Solids

Sample No. *	Flow Rate GPM	% Feed in Tailing	% Feed in Concen-trate	% Wt. Pb	% Wt. Cu	% Wt. Zn	Recovery Pb, % **	Recovery Cu, % **	Recovery Zn, % **
25A	30	25	75	15.20	2.58	17.70	87.8	86.1	83.8
B	30	25	75	2.10	0.42	3.42			
26A	30	50	50	10.30	1.42	12.30	78.8	83.1	83.7
B	30	50	50	2.78	0.29	2.38			
27A	30	75	25	11.40	1.10	8.84	70.1	67.5	50.0
B	30	75	25	4.85	0.53	8.83			
28A	12	25	75	5.30	0.45	3.46	83.0	81.5	82.2
B	12	25	75	0.72	0.10	0.75			

Sample No. *	Flow Rate GPM	% Feed in Tailing	% Feed in Concen-trate	% Wt. Pb	% Wt. Cu	% Wt. Zn	Recovery Pb, % **	Recovery Cu, % **	Recovery Zn, % **
29A	12	50	50	2.20	0.35	2.69	69.2	71.4	72.1
B	12	50	50	0.94	0.14	1.04			
30A	12	75	25	2.70	0.25	1.45	58.0	63.0	40.8
B	12	75	25	1.96	0.15	2.10			
31A	30	0	100	0	0	0	0	0	0
B	30	0	100	18.00	3.10	24.30			
32A	30	100	0	21.00	3.60	28.40	100.0	100.0	100.0
B	30	100	0	0	0	0			

Sample No. *	Flow Rate GPM	% Feed in Tailing	% Feed in Concen-trate	% Wt. Pb	% Wt. Cu	% Wt. Zn	Recovery Pb, % **	Recovery Cu, % **	Recovery Zn, % **
33A	12	0	100	0	0	0	0	0	0
B	12	0	100	6.10	0.90	7.20			
34A	12	100	0	6.40	0.80	6.90	100.0	100.0	100.0
B	12	100	0	0	0	0			

* Where "A" and "B" represent concentrate and tailing respectively.

** Where % recovery = $\frac{\%\ Wt.\ A}{(\%\ Wt.\ A + \%\ Wt.\ B)} \times 100$

All samples with a flow rate of 30 GPM are the 65 to 100 mesh fraction and all samples with a flow rate of 12 GPM are the -100 mesh fraction.

APPENDIX IV

Tabulation Of Data For Economic Evaluation Of The Concentrate Produced From One Ton Of Ore

Condition 1. 25% tailing (75% of feed retained)

Fraction	Pb		Cu		Zn	
	Recovery %	Value $	Recovery %	Value $	Recovery %	Value $
65-100 mesh	87.8	1.18	86.1	0.52	83.8	1.06
-100 mesh	83.0	10.10	81.5	4.40	82.2	9.32
Sub Total		11.28		4.92		10.38

Total * recovery $29.21

Value of concentrate per ton of concentrate $\frac{(\$29.21)}{(0.75)} = \$38.94$

Condition 2. 50% tailing (50% of feed retained)

Fraction	Pb		Cu		Zn	
	Recovery %	Value $	Recovery %	Value $	Recovery %	Value $
65-100 mesh	78.8	1.06	83.1	0.50	83.7	1.06
-100 mesh	69.2	8.42	71.4	3.86	72.1	8.18
Sub Total		9.48		4.36		9.24

Total * recovery $25.71

Value of concentrate per ton of concentrate $\frac{(\$25.71)}{(0.50)} = \$51.42$

* Includes $1.05 for Ag, $1.58 for Au.

Condition 3. 75% tailing (25% of feed retained)

Fraction	Pb		Cu		Zn	
	Recovery %	Value $	Recovery %	Value $	Recovery %	Value $
65-100 mesh	70.1	0.94	67.5	0.40	50.0	0.64
-100 mesh	58.0	7.06	63.0	3.40	40.8	4.62
Sub Total		8.00		3.80		5.26

Total * recovery $19.69

Value of concentrate per ton of concentrate $\frac{(\$19.69)}{(0.25)} = \$78.76$

* Includes $1.05 for Ag, $1.58 for Au.