PROBLEMS IN CYANIDING A COMPLEX BASE ORE WITH LOG OF TESTS MADE, FLOW SHEET, AND EQUIPMENT DECIDED UPON AS THE BEST ADAPTED TO TREAT THE ORE.

A THESIS

SUBMITTED TO THE FACULTY OF THE COLLEGE OF ENGINEERING

IN CANDIDACY FOR THE DEGREE OF ENGINEER

OF MINES.

By

FRANK SILVA,

SANTA PAULA, CALIFORNIA.

1923.

CONTENTS.

PREFACE.

The South American Development Company has at their mines in Zaruma, Ecuador, a large body of low grade ore known as the Contabria Vein. This ore has continually been a source of trouble when treating it in the mill with their general run of quartzose ore.

They also have a large slime pile of accumulated slimes which accumulated from treating Contabria ore by amalgamation in their old amalgamation plant. The ore was stamped, amalgamated and then classified. The slimes were collected in the slime pile and the sands leached in their leaching plant with cyanide solution. It was calculated that there are sufficient slimes to treat 50 tons per day for five years.

The writer spent several months in South America and with the General Engineering Company, Salt Lake City, Utah; making tests and working out a flow sheet with a view of treating this ore and the accumulated slimes, by means of an addition to their present mill.

The following is a report of the work done.

I. CHARACTER AND ANALYSES OF ORE.

The Contabria Ore is a complex base ore containing sulphides of copper, iron, zinc and lead in the ratio as shown by analyses which is as follows:

% Insolubles	58.45
% Copper	.87
% Iron	11.45
% Zinc	7.77
% Lead	6.12
% Sulphur	11.45
% Undetermined	3.59

Value Au @ 20.67 U.S. Currency	= 7.82
Oz. Ag.	= 4.56

A small part of the sulphides mentioned above have been oxidised forming lead and zinc carbonates, and also oxides and sulphates of iron.

The gangue material is quartz.

The silver is in sulphide form and very refractory. Only about 50% of the silver can be saved by cyanidization. Abput 40% of the gold is free, and the remainder of the gold is locked up in the sulphides. For many years this ore was treated by amalgamation with an extraction of 45 to 50% of the gold and 18 to 20% of the silver. When the new cyanide plant was designed, amalgamation was not considered, due to loss in theft by the natives. Also when the present tests were made, the management was very much opposed to amalgamation because of the above reason, so no amalgamation tests were run.

Only the gold and silver will be considered as of value in the extraction of the metals.

II. PRESENT METHOD OF MILLING.

Their present method of milling is as follows:

Wet crushing in stamps to four mesh, classifying in Dorr classifiers in closed circuit with the tube mills, which grind to 85% -200 mesh. The pulp, which is 12½% solids, is thickened in #1 Dorr Thickener to 40% solids, then agitated in Dorr Agitators for 36 hours. The pulp is then de-watered in a series of Dorr Thickeners arranged for counter-current decantation. The overflow from #1 thickener is clarified, precipitated in Merril presses, and is then used as the wash solution in the counter-current decantation system.

III. REASONS WHY ORE CANNOT BE TREATED SUCCESSFULLY BY THE PRESENT METHOD OF MILLING.

This base ore from Contabria Vein cannot be treated successfully by the present methods of milling for the following reasons:

1. It is more refractory than the general run-of-mine ore requiring a longer period of agitation to obtain the same extraction.

2. It is partly oxidised, at least it has some oxidised copper minerals and some sulphates of iron, which consume cyanide rapidly.

3. These oxidised minerals not only consume cyanide, but foul the solutions on accumulating, resulting not only in poor extraction of values from the base ore, but the extraction from the remainder of the ore is also reduced.

4. There is considerable silver in the base ore; and in order to extract the silver, a higher strength cyanide solution must be used as well as a longer period of agitation.

5. To consume the least amount of cyanide, the Contabria ore requires a high protective alkalinity, while the general run-of-mine ore requires a low alkalinity.

From the above it is seen that if the Contabria ore were treated at all it should be treated separately.

IV. OBJECT OF FLOTATION TESTS.

A series of flotation tests were made with the idea that the process would act as a wash. In the concentrating of the ore, the oxidised material and the sulphates, which are the large consumers of cyanide, would be reduced to a negligible amount. Also the preliminary alkaline treatment which would be necessary to break up the froth, would neutralize all acid substances and reduce the cyanide consumption.

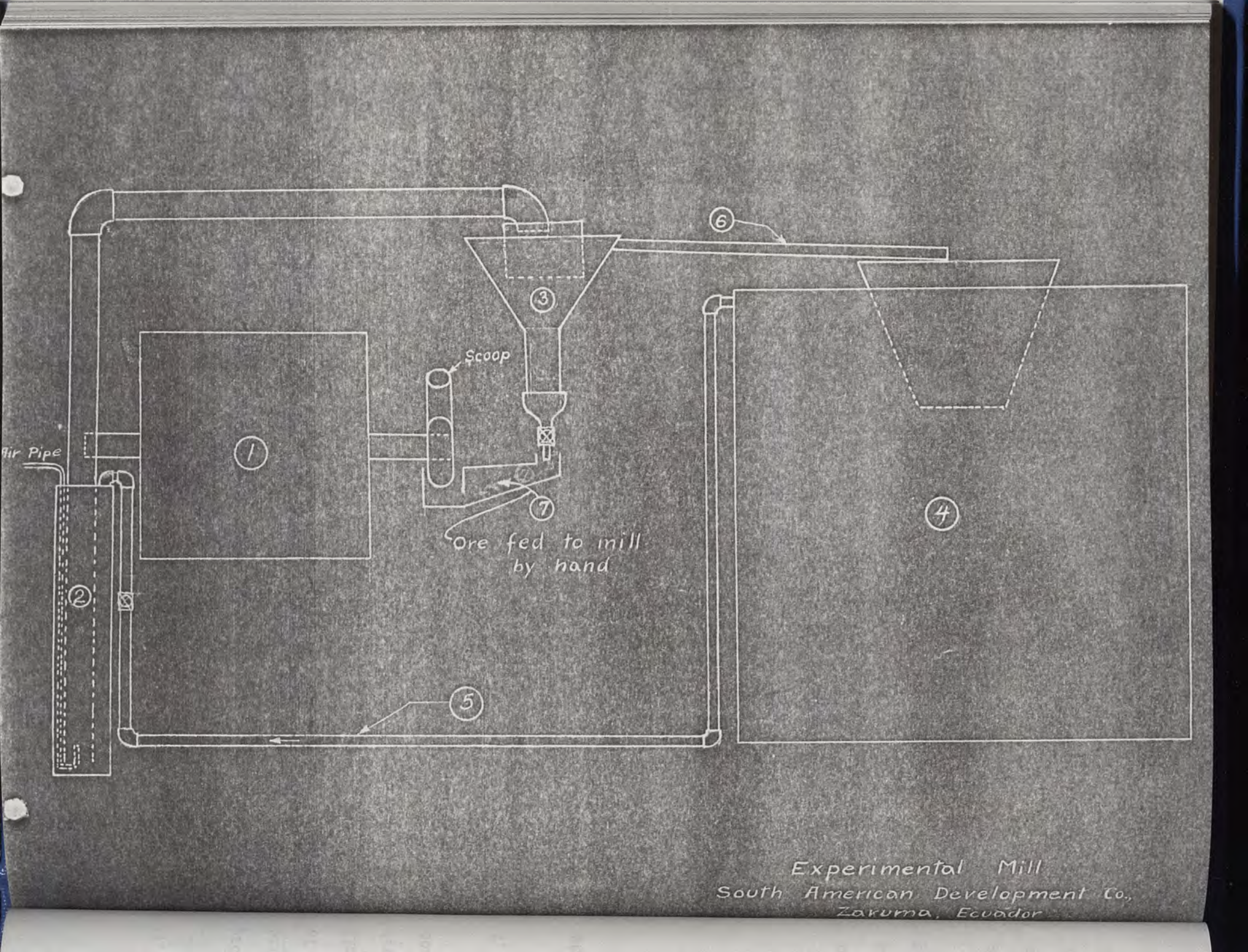
V. EQUIPMENT AND FLOW-SHEET USED IN TESTS.

The equipment and flow-sheet used for the experimental mill was as follows: (The numbers noted correspond to those shown in sketch).

1. 2' x 2' Pebble Mill 38 R.P.M.
2. Air lift to raise pulp to cone classifier.
3. Diaphragm, cone classifier.
4. Settling tank 4' diameter by 4' high.
5. Over-flow solution from settling tank, returned to dilute pulp for cone classifier.
6. Over-flow from cone classifier, 79%-200 mesh.
7. Under-flow sands from cone classifier, returned to mill for regrinding.

The sample of ore was prepared by crushing in a #1 Dodge Crusher to ½" and then through a disc pulverizer to 12 mesh. The sample taken, usually about 300 pounds, was then ready for the fine grinding in the pebble mill.

In order to start the mill, the settling tank was first filled with water, and water was also allowed to circulate in the tube mill circuit. The tube-mill was then started and ore fed by hand into the scoop box. Five hours were required to grind a 300 pound sample to 79%-200 mesh. The size of mesh to which ore was crushed for tests was



Experimental Mill
South American Development Co.,
Zaruma, Ecuador

regulated by the mesh of the present mill feed, which is 75 to 80%-200 mesh. As the settling tank began to fill up, the water which was being replaced by the pulp was allowed to over flow and run to waste.

After grinding the sample, the pulp was left to settle for about twelve hours, and then the extra water decanted. The sample was then ready for the flotation tests.

The flotation machine used was a Jones- Belmont experimental machine. The most satisfactory speed for the machine was found to be 1340 R.P.M. 2000 Grams of ore was used and about 16,500 grams of water for each test. This gave the pulp of 12% dry ore. To calculate the oil, the grams used per test corresponded to pounds of oil required per ton of ore.

The most satisfactory method found for adding the oil was by means of a pipette. For each oil used, the number of drops required for one gram of the oil were determined. The oil was then added accordingly by counting the drops.

There were no flotation oils obtainable at the mine with the exception of Pine Oil. However, a number of oils were used for various purposes about the mine and mill, were gotten together and used as follows:

Stockholm oil of tar, a heavy black oil, used chiefly for oiling wire cables in the mine; black car oil; mineral castor oil; and crude oil. These four oils were used as collectors.
Creoline was used as a dilutant.
Pine oil, turpentine and kerosene were used as frothers.

Of the collectors tried, the Stockholm oil of tar appeared to be the best. Creoline was found to be a very good dilutant and probably answered the purposes or creosote. Pine oil was found to be by farthe

best frother. Kerosene was found to be a very poor frother, though when mixed with pine oil, appeared to improve the grade of concentrates; and, in several instances, the extraction.

Finally, after experimenting with about 27 different mixtures, using the above oils in various ratios and combinations, the following mixture was decided on as being the best mixture to use on the ore. It was called mixture #17 in the reports and cinsisted of the following:

Stockholm oil of tar	50% (by weight)
Creoline	25%
Kerosene	12½%
Pine Oil	12½%

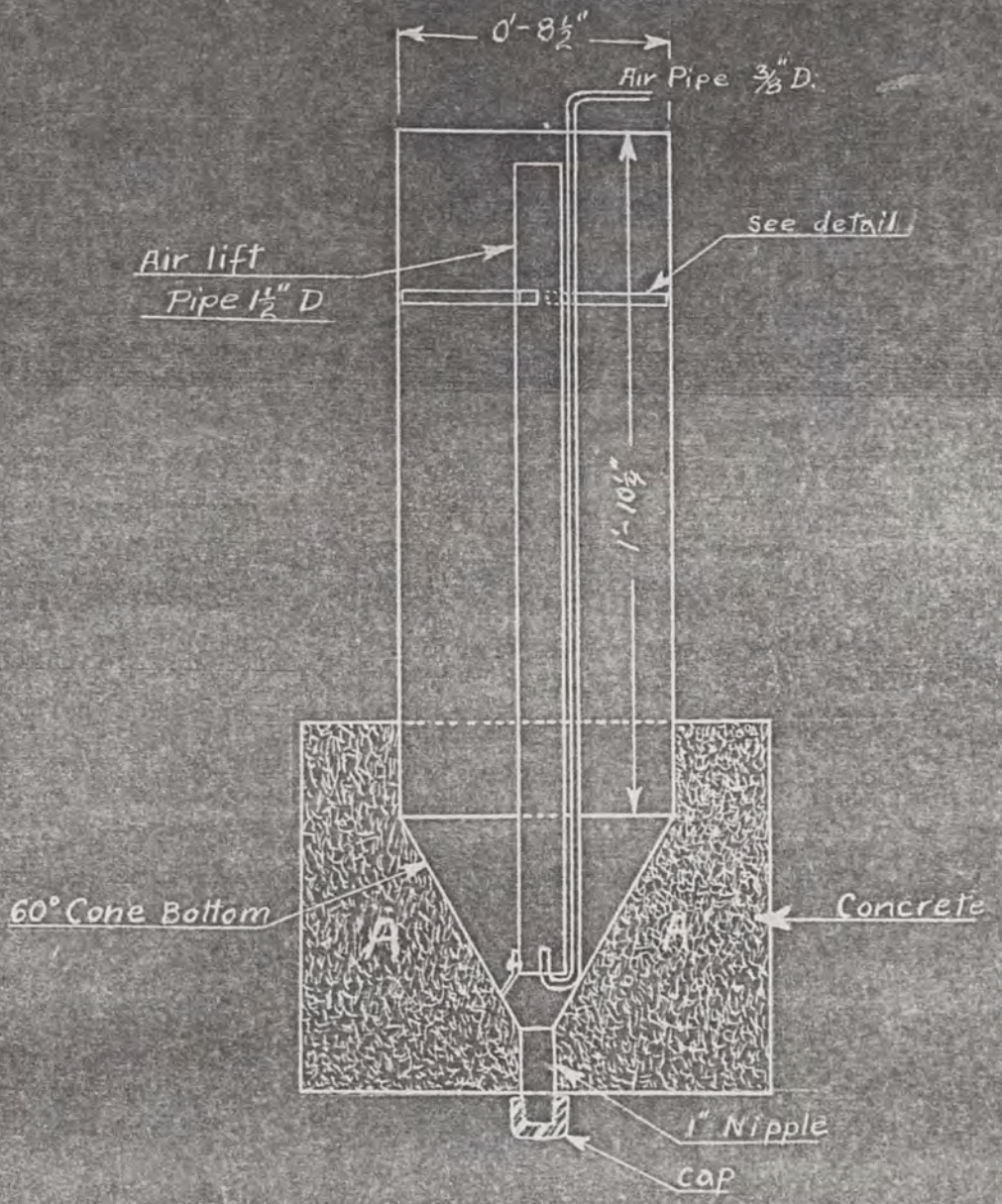
For the cyaniding of the concentrates three small Pachucha agitators were made (See sketch). They were made of an 8½" pipe cut in 22" lengths, fitted into the cement form (A) which had a 60 degree cone bottom. This made an agitator of sufficient size so that small samples could be taken for titrating and assaying without materially changing the results.

VI. CHARACTER AND ANALYSES OF CONCENTRATES.

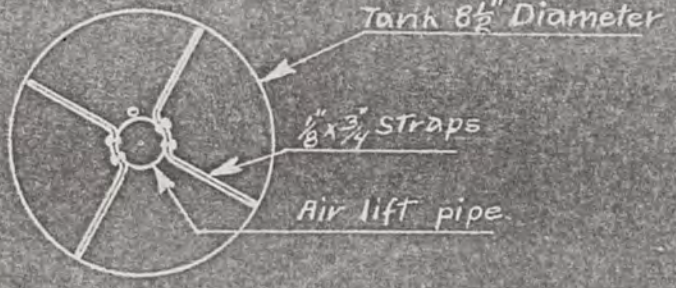
Three samples of Contabria ore were taken:

Sample #1	Value Heads*		Value Concentrates	
	Au.	Ag.	Au.	Ag.
1	9.10	2.42	39.28	9.31
2	7.86	2.28	22.33	5.82
3	4.65	2.09	14.47	7.60

* All values as given in thesis are in U.S. Currency except as where stated.



Vertical Section through center of tank.



Detail showing method of holding air lift in center of tank.

Working Drawing of
Experimental Agitator
South American Dev. Co
Zaruma, Ecuador
Scale 2" = 1'

Analyses, as follows, were made on the heads, concentrates, and tails of sample #2 :

	Concentrates	Heads	Tails
Value Au.	18.61	7.82	.83
Oz. Ag.	11.28	4.56	.36
% Insolubles	19.18	58.45	91.33
% Copper	2.59	.87	Trace
% Iron	17.83	11.45	3.84
% Zinc	16.08	7.77	1.34
% Lead	13.11	6.42	.90
% Sulphur	24.51	11.45	1.18
% Undetermined	6.70	3.59	1.41

To obtain the above grade of concentrates, a third of the concentrates were collected as a finished product, and the remaining two-thirds were collected and re-treated. The most satisfactory method found for collecting the concentrates in this manner was to first find out the least amount of oil which could be used to obtain the proper extraction and the time required for treatment. Then give the ore three treatments, adding each time, a third of the oil which was found to be necessary in the preliminary tests and at the same time dividing the time for treatment into three equal intervals. In this manner more uniform results could be obtained; and in most instances, a better extraction and a better grade of concentrates. Also it was possible to closely duplicate a certain flow sheet by this method. This is desirable because some concentrates require more re-cleaning than others.

From the above analyses it appears quite possible that the concentrates could be separated into a lead concentrate and a zinc concentrate; and possible an iron concentrate, containing a greater part of gold which could be cyanided. The greater part of the silver of which only 50% is saved by cyaniding would probably be saved in the lead concentrate.

Owing to the isolated locality of the mine, this latter method is

out of the question, and no tests were made to separate out the different metals in the concentrate. However, if the mine were in the States, it would probably be treated so.

VII. CYANIDING OF CONCENTRATES.

In cyaniding the concentrates, the most satisfactory method found was to agitate the concentrates in an alkaline solution of caustic soda for a period of five hours to break up all the froth, neutralize all acid substances and a greater part of the latent acidity. Cyanide to be added after five hours agitation in the alkaline solution. Lead acetate was found to be unnecessary because the solution gave a strong test for lead.

The Sp. Gr. of pulp in agitators is to be kept at 1.425.

The caustic soda solution in the preliminary agitation should be of such strength that the concentrates leaving the second agitator should titrate not over .5 pounds CaO per ton, as the alkali leaving this agitator would be wasted. All the caustic soda should be added to the first agitator.

From the preliminary agitator, the concentrates are to pass to the second thickener where they will receive a thorough washing by adding extra water before passing to the Oliver filter.

The cyanide solution will be added by means of a mixer attached to the Oliver filter.

The concentrates will then receive a primary agitation of twelve hours in cyanide solution before flowing into # 1 thickener of the present mill. Lime will be added to the primary agitators instead of caustic soda and the alkalinity kept up to.5 pounds CaO per ton of

solution.

The cyanide strength of solution in the primary agitators is to titrate 3 pounds NaCN per ton in the first agitator which strength decreases to 1.8 pounds in the second agitator, thus flowing into #1 thickener of the present mill at only a slightly higher strength than the regular stock solution.

The cyanide solution in the present mill titrates as follows:

# 1 Thickener.
NaCN = 1.2 pounds per ton.
CaO = .5 " " "

#1 Agitator.
NaCN = 1.7 " " "
CaO = .5 " " "

#4 Agitator.
NaCN = 1.2 " " "
CaO = .5 " " "

#6 Thickener.
NaCN = .5 " " "
CaO = .2 " " "

No cyanide tests were made on Sample #1. From samples # 2 and 3 cyanide tests were run for both the heads and the concentrates.

Sample #	Value Heads Au.	Ag.	Total	Value Cyanide tails Au.	Ag.	Total	Value Conc. Au.	Ag.	Total	Value Cyanide tails from conc. Au.	Ag.	Total
2	7.86	2.28	10.14	2.07	1.04	3.14	22.33	5.82	28.15	2.48	2.91	5.39
3	4.65	2.07	6.72	.41	1.02	1.43	14.47	7.60	22.07	1.24	4.48	5.72

Using Hamilton's* formula for ratio of concentration, we have $R = \frac{C-T}{H-t}$

When R= Ratio of Concentration
C= Concentrates
T= Tailing from cyaniding of concentrates
H= Heads, or original ore
t= Tails from cyaniding of heads.

*Taken from Hamilton's "Manual of Cyanidization."

Sample #2

$$R = \frac{28.15 - 5.39}{10.14 - 3.14} = \frac{22.76}{7.00} = 3.25$$

Sample #3

$$R = \frac{22.07 - 5.72}{6.72 - 1.43} = \frac{16.35}{5.29} = 3.09$$

Considerable tests were run on the direct cyaniding of the ore to determine the cyanide consumption. It was found that the cyanide consumption varied considerably and increased in proportion to the oxidised material. The consumption varied from two pounds to as high as seven pounds per ton of ore with an average of between four and five pounds.

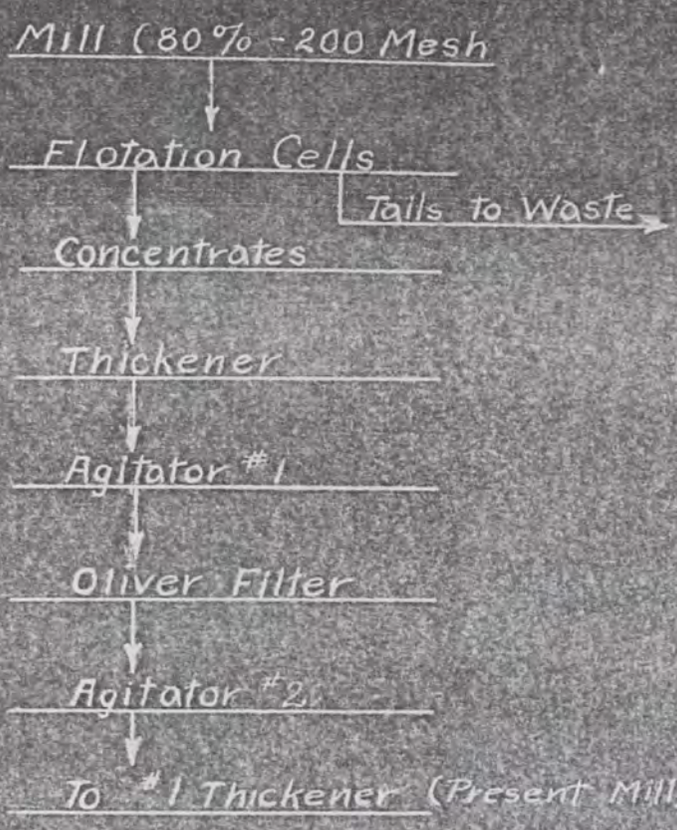
The concentrates contain mostly the clean sulphides and the free gold, and is a more uniform product. For this reason the cyanide consumption does not vary within such wide limits and was found to be about four pounds per ton of concentrates. This shows a saving in cyanide of over 200%. However it is offset some what by the loss of values in concentrating which amount to 8% of the total value of the ore. (See Test #3 pages 25, 26 and 27).

VIII. FLOW SHEET.

The following is the flow-sheet finally decided on, showing possibly the best method to treat ore. (See blue print)

The flow-sheet was designed for an addition to their present mill to treat 50 tons of Contabria ore and 50 tons of accumulated slime by flotation and cyaniding the concentrates.

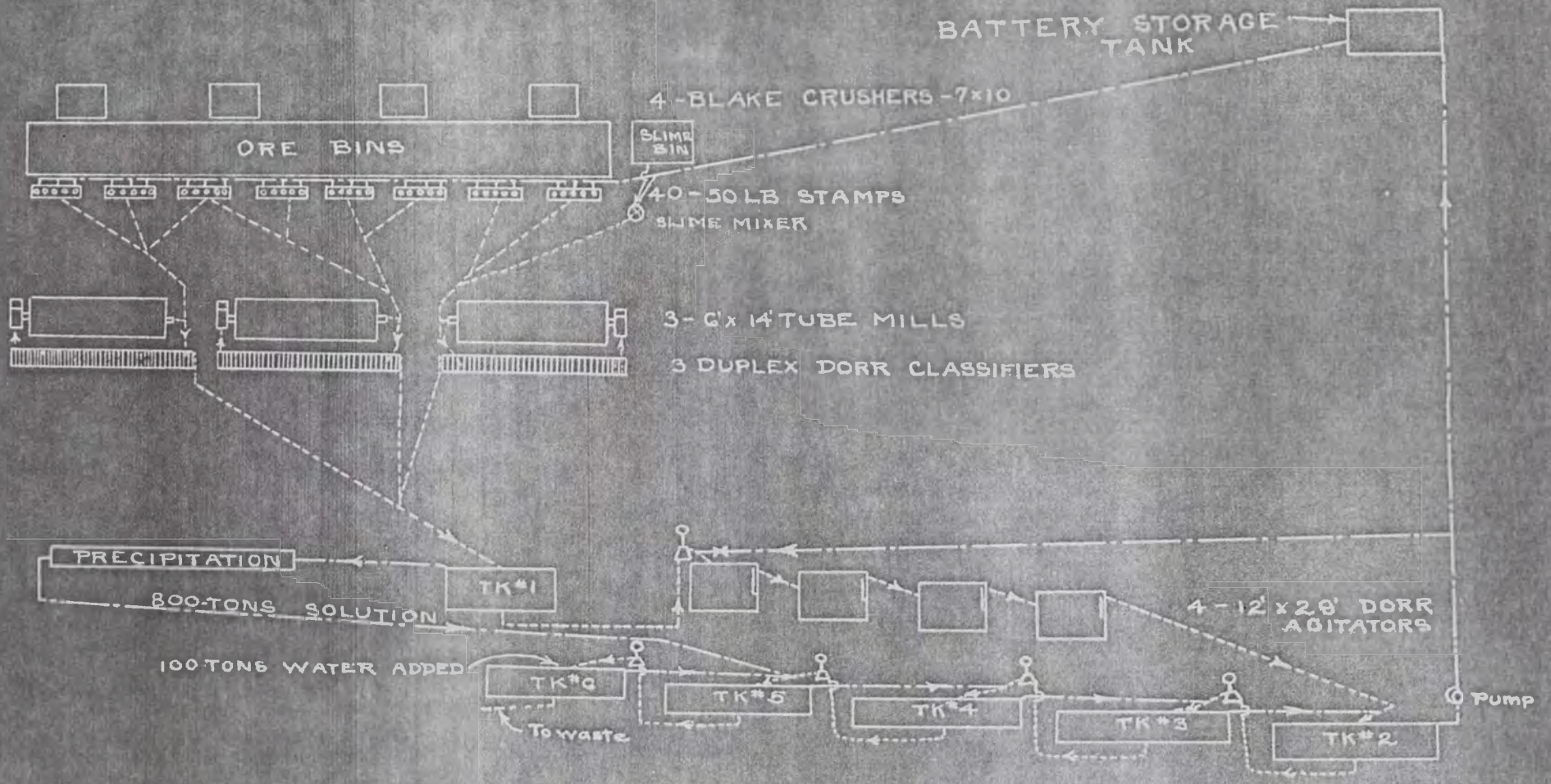
Flow sheet showing proposed addition to present mill of the South American Development Co., Zaruma, Ecuador.



Concentrates are to be given a preliminary agitation of five hours in #1 Agitator to break up the froth and neutralize the latent acidity. Agitaion to be made in an alkaline solution of caustic soda.

#2 Agitator is to give concentrates a cyanide treatment of 12 hours before turning them into the present mill circuit.

Ore is to be milled in water.



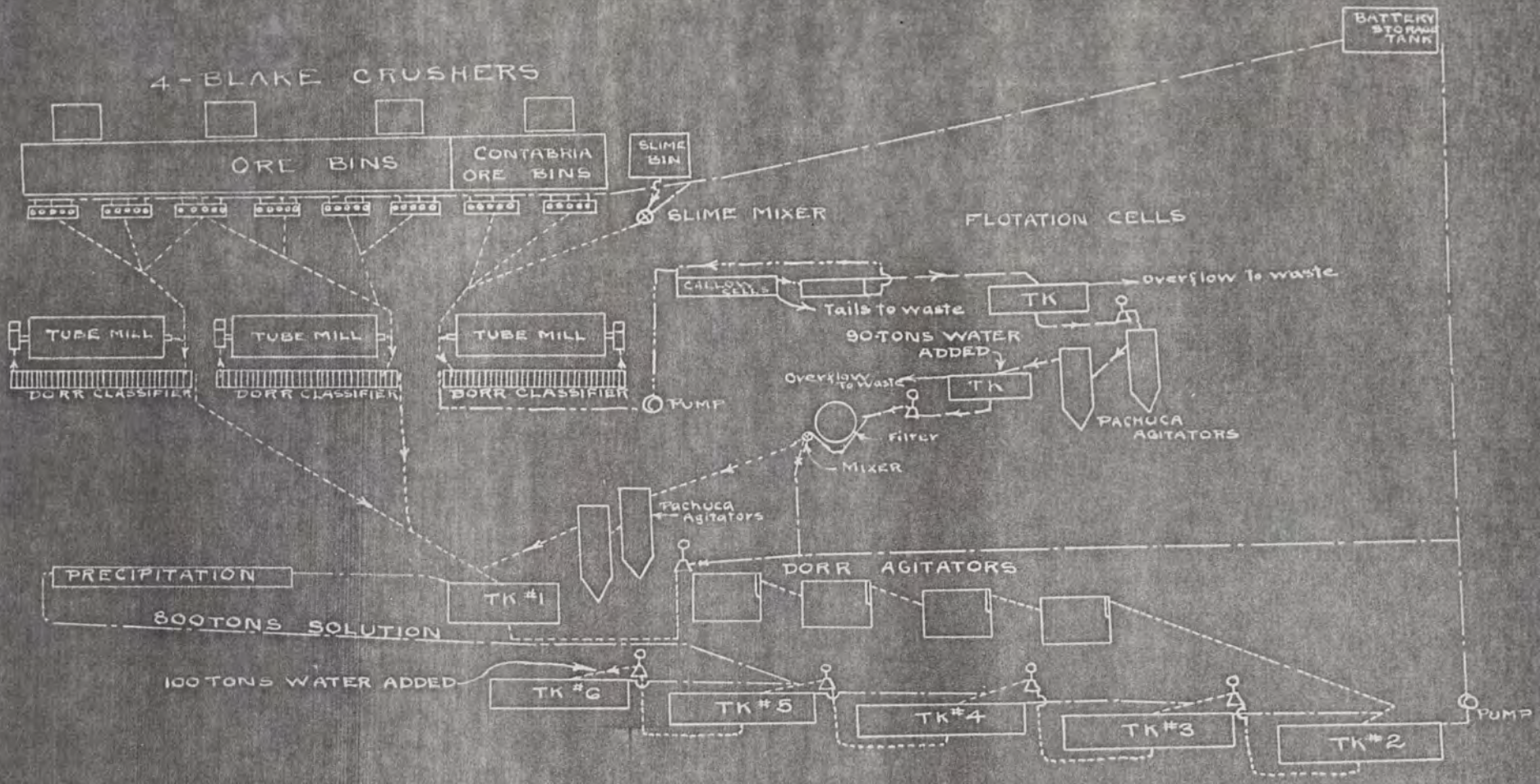
FLOW SHEET OF 280 TON CYANIDE PLANT
SOUTH AMERICAN DEVELOPMENT CO.,
ZARUMA, ECUADOR.

LEGEND:

——·—— CYANIDE SOLUTION

---------- TAILS IN CYANIDE SOL.

TK - THICKENER



FLOW SHEET OF SOUTH AMERICAN DEV. CO MILL
WITH ADDITION OF PROPOSED FLOTATION PLANT.
ZARUMA, ECUADOR.

LEGEND:

——·——·—— CYANIDE SOLUTION

---------- TAILS IN CYANIDE SOL.

—— —— —— CONCENTRATES IN WATER SOL.

——·——·—— ORE PULP " " "

TK = THICKENER

Owing to the fact that all equipment delivered at the mines must be sectionalized for mule-back haul, the present mill has four 7 x 10 Blake crushers, or one for every two batteries of five stamps each. Also there are three pebble mills. In remodeling the mill so that the extra ore may be treated by flotation, it is planned to use one of the Blake crushers, two batteries and one tube mill for the Contabria ore, leaving the other three crushers and six batteries for the general run of mine ore. The ore-bin can be partioned off so that the ore will not be mixed. By the above, it is seen that no extra crushing equipment will be needed.

The tube mills are run in closed circuit with Dorr classifiers, and the ground ore overflows the classifiers at 85% - 200 mesh. The accumulated slimes which will pass through a mixer will flow into the same classifier with the Contabria ore. The overflow from this classifier will form the flotation feed and will be pumped to the Callow Flotation cells.

The tails from the rougher cell will be run to waste and the tails from the cleaner cell will be elevated back to the head of the rougher cell. The concentrates from the cleaner will flow to the first thickener 17' diameter by 5' high equipped with Dorr Mechanism.

The concentrates are to be thickened to a Sp. Gr. of 1.60 in the first thickener and pumped to the first of two preliminary Pachuca Agitators 5' diameter by 14' high. The overflow from the thickener to be run to waste. While the concentrates are being pumped to the agitator, the pulp is to be diluted to a Sp. Gr. of 1.45 by means of a water pipe connected up to the pump suction and regulated with a gate valve.

The agitation is to be in an alkali solution containing 1.5 lbs. NaOH per ton in the first agitator which will decrease to .5 lbs. NaOH per ton in the second agitator. There will be two small preliminary agitators instead of one because it was thought that the concentrates would receive a more uniform agitation and have less chance to pass through the agitator without receiving the required agitation; and, also because the use of two agitators would make a saving in caustic soda.

From the second preliminary agitator the pulp will pass to the second thickener where the pulp will be thickened to a Sp. Gr. of 1.60. Considerable water will be added to this thickener so as to give the concentrates a thorough washing before going to the filter. This thickener will be similar to the first thickener previously described. The overflow from the second thickener will also be run to waste.

The thickened product from thickener #2 will be pumped to the Oilver filter.

There will be a small mixer attached to the filter where the filtered pulp will be mixed with the cyanide solution from the present mill and pumped to the second series of Pachuca agitators, 6' diameter by 20' high, where the concentrates will receive a primary agitation of twelve hours in cyanide solution containing three pounds of NaCN per ton. The concentrates will then be ready to be mixed with the present mill feed and will flow into #1 thickener of the present mill.

IV. SPECIFICATIONS AND COST OF EQUIPMENT.

(a) Flotation Equipment.

1, 7 - compartment, single 3' x 3', Fu. Callow Pneumatic Flotation cell complete with all iron work, wood work, blankets, main air header pipe with valves and fittings attached.

1, No.461, 4" Gate Valve, F. & D. with 5/8" extension stem, 5' 5" long, valve complete with 2- 1/16" CC ring gaskets and necessary machine bolts and hexagon nuts.

1, 2- compartment single, 3' x 3' Fu. Callow Pneumatic Flotation cell, complete with all iron work, wood work, blankets, main air header pipe with valves and fittings attached.

1, No. 461, 3" Gate Valve, F. &.D. with 5/8" extension stem, 5' x 5" long, valve complete with 2- 1/16" CC ring gaskets and necessary machine bolts and hexagon nuts.

Total weight for foreign shipment .....12,273 lbs

Price $4,640.00

1, No. 3 Flotation type blower with heavy duty half-box bearing 2- 1/2" relief valve, good for a minimum of 900 cu.ft. air per minute at 4 pounds pressure at 275 R.P.M.

Total weight for foreign shipment ..... 5,490 lbs

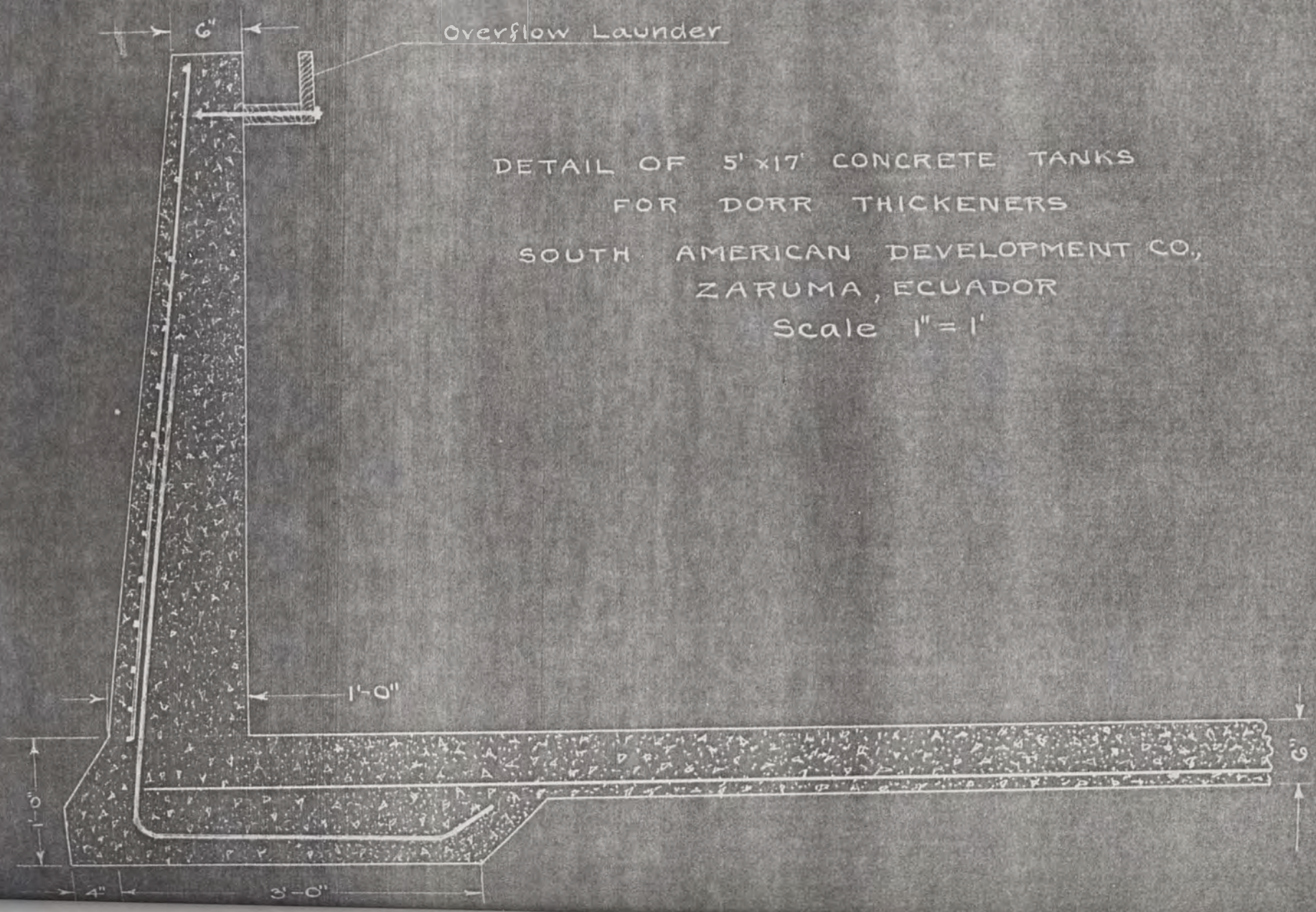
Price ..... $ 782.10

Flotation cells and valves, f.o.b. Salt Lake City, Utah.

Blower, f.o.b. Connersville, Indiana.

(b) Thickeners.

The thickener tanks are to be made of reinforced concrete, 1:2:4: mixture. The reinforcement to consist of ½" twisted steel rods located 2" inside the exterior face and spaced apart from 3½" to 10" at the top. Vertical rods spaced 9' apart around the wall to be used to hold the rods in place. To tie the wall to the bottom, ⌊ shaped ½" rods, 7' long are to be used. The bottoms are to be reinforced with #42



DETAIL OF 5' x 17' CONCRETE TANKS
FOR DORR THICKENERS
SOUTH AMERICAN DEVELOPMENT CO.,
ZARUMA, ECUADOR
Scale 1" = 1'

Triangular Mesh wire.

The following formulae were used in determining the dimensions of the steel bars and the thickness of the concrete:

A= Area of bars in square inches per square foot of height of wall.

D= Vertical distance in inches between two layers of bars at the point selected.

P= Water Pressure in pounds per square foot at the point selected.

T= Thickness of concrete wall in inches at the point selected.

r= Radius of the tank in feet.

S= Stress in steel in pounds per square inch allowed under the assumption that the concrete carries no tension.

c= Stress in concrete in pounds per square inch allowed.

a= Area in square inches of steel in each layer.

The area of bars required per foot of height of wall is: $A = \frac{P\ r}{S}$

The vertical distance between the layers of bars is: $D = \frac{12a}{A}$

The thickness of concrete wall at any point is: $T = \frac{Pr - 9Ac}{12c}$

The cost of each tank is as follows:

Dorr Mechanism 735.00

Concrete Tank 1350.00

Attached is a detail cross-section of the concrete tank.

(c) Pachucha Agitators.

The Pachucha tanks are to have conical bottoms having a slope of 60 degrees, the cylindrical shell to extend to the floor, and stiffened by vertical angle irons. The lower end of the cone to be closed by a cast iron plate with a 4" discharge

hole with quick opening gate valve. Each tank to have one manhole in the cone to allow inspection of the inside. Each tank to have a vertical cylindrical tube (air lift) open at both ends and 1/10 the diameter of the tank. The air lifts to extend from 12" of the bottom to 18" of the top. A 1" air pipe to pass downward outside the central tube about 4" from the bottom. A second 1" air pipe to pass downward outside the central tube within a few inches of the bottom of the tank, which is to serve to keep the pulp in suspension while the tank is being filled or emptied.

Calculations for Pachuca Agitators:

Preliminary Agitators:

To agitate 30 tons of concentrates a period of five hours:

30 tons per day equals 1.25 tons per hour or 6.25 tons per five hours. Specific Gravity of charge = 1.427.

% Dry concentrates = 40.59

% Solution = 59.41

Tons solution in 6.25 tons concentrates = 8.70

Total tons in charge = 14.95

14.95 tons = 29,900 pounds.

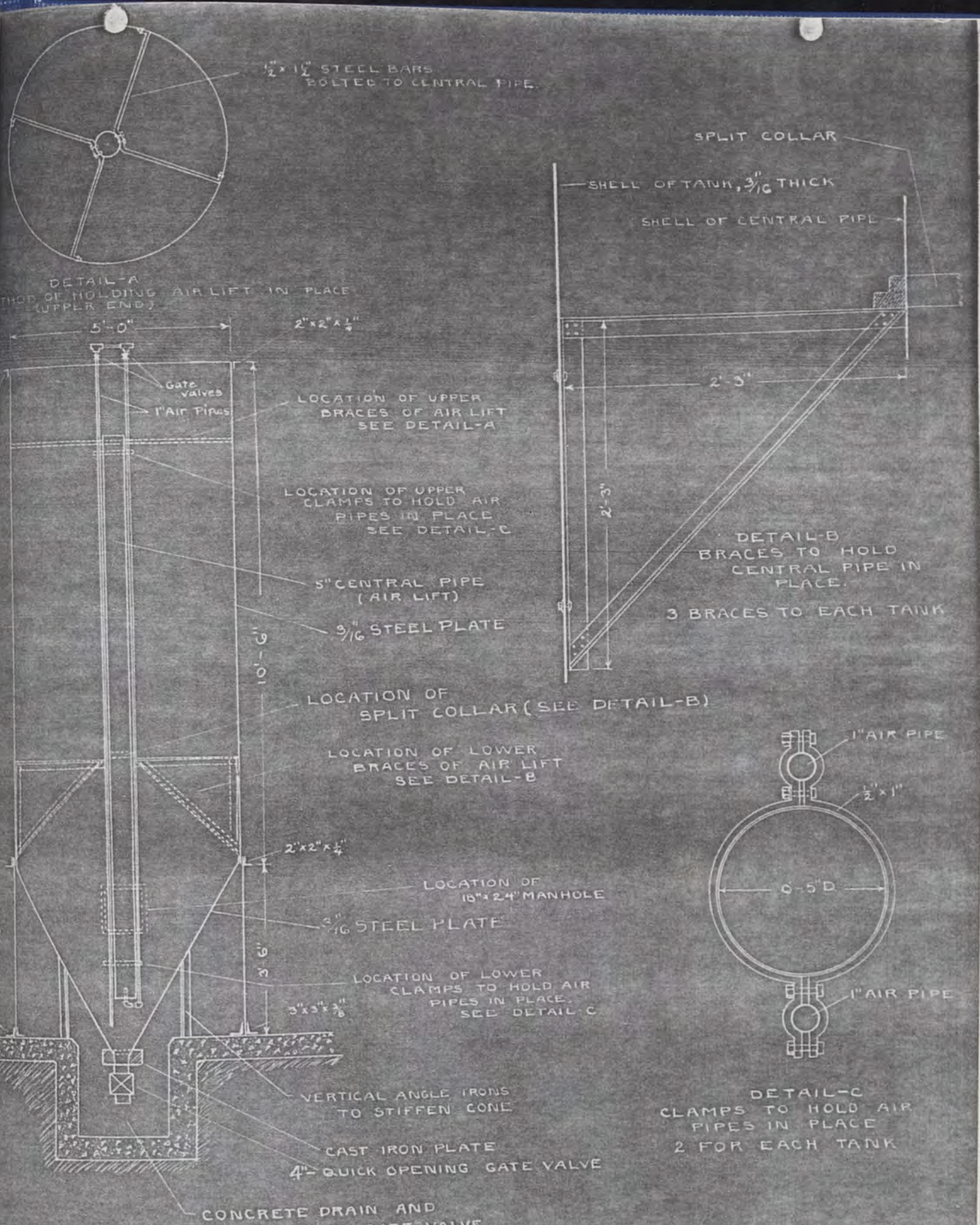
1 cu.ft. at 1.427 Sp.G. = 62.5 x 1.427 = 89.187 lbs.

$\frac{29900}{89.187}$ = 335 cu.ft. or 167.5 cu.ft. per agitator.

Primary Agitators:

To agitate 30 tons per day a period of 12 hrs. = 1.25 x 12 = 15 tons per 12 hours.

6:25:15 = 335:X

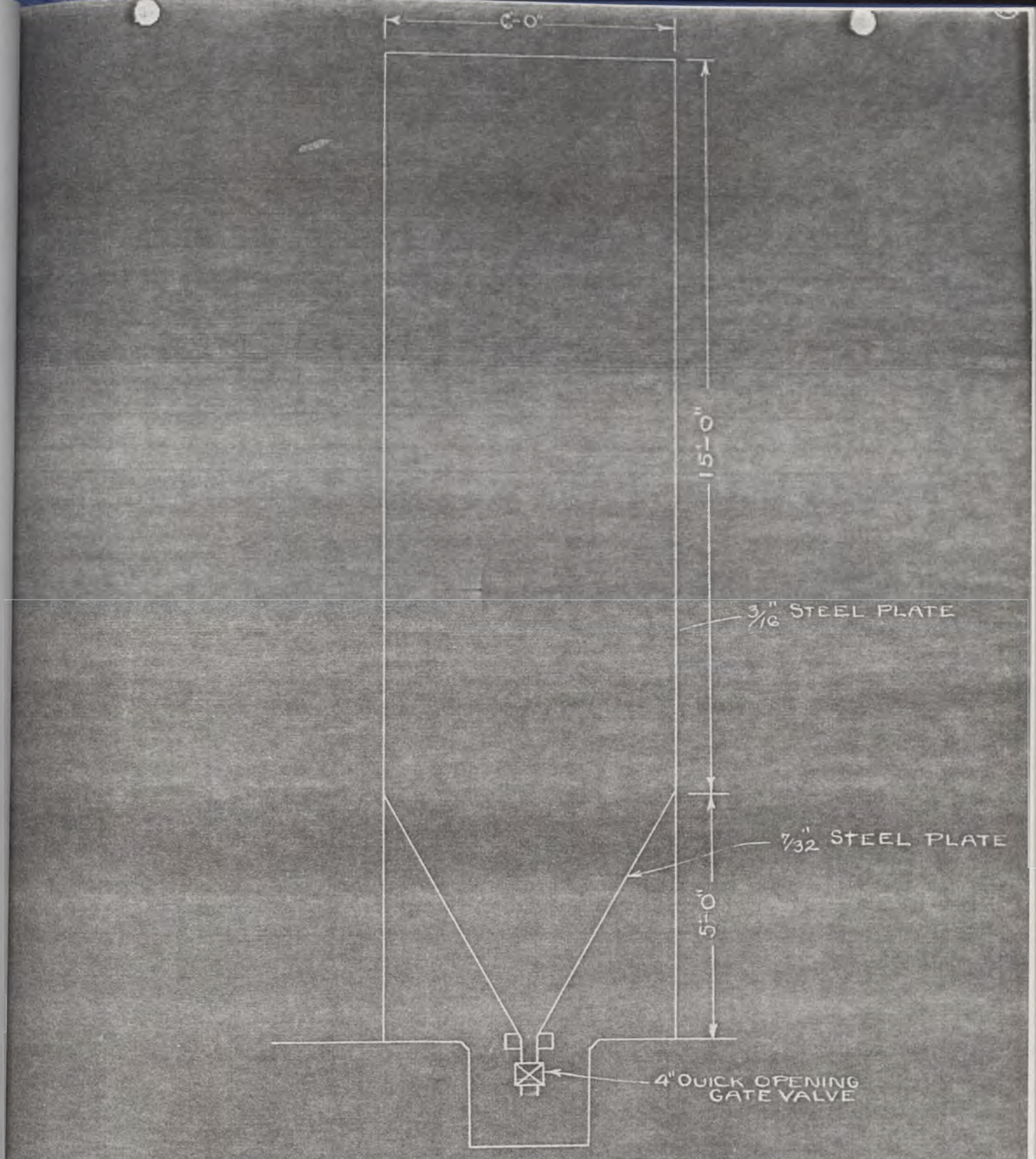


PACHUCA AGITATING TANK

5' DIA x 14' HIGH

Scale 3/8" = 1'

DRAWN BY FRANK J. SILVA



DIMENSIONS OF

PACHUCA AGITATING TANK

6' DIA. x 20' HIGH

Scale-3/8"=1'

The Agitator is to conform to drawing #1 as to details, with the ception of the size. The central air lift to be 6" in Dia and to tend from 12" of the bottom to 18" of the top. The air pipes to be in Dia. The Angle irons to be of same dimensions. Tank to be made steel plate with the exception of the cone 7/32" steel plate

625X = 5025
X = 804 cu.ft. or 402 cu.ft per agitator.

Cost of small tanks $500.00

Cost of large tanks 700.00

Attached are detail drawings of the tanks.

(d) Filtering Equipment:

1, Oliver Continuous Filter, 8' diameter x 8' face, belt d driven, complete with steel tank, oscillating agitator, steel arms and rims, redwood staves, steel pipe, steel screen, canvas cover, steel wite winding, automatic wiring device and brake wheel and all necessary parts and fittings; shipped knocked down; requires a maximum of 2 H.P.; gross weight 12,000 pounds net weight 11,000 pounds; measurements approximately 350 cu.ft. Price $4110.00

1, Oliver Dry Vacuum Pump, 9½" x 8" JO type, with enclosed crank case, splash oiling system, light steel leaf valves interchangeable for inlet of discharge; requires 6 H.P. under normal filter operating conditions of 22" vacuum; displacement 200 cu.ft. at 300 R.P.M.; weight 1700 lbs.Gross; 1550 lbs. Net., measurements 41 cu.ft. Price 518.00

1, Centrifugal Pump, 1-1/2"Oliver, cast iron,belt-driven with special suction,ball bearings and single pulley. Requires 2 H.P. under filter operating conditions; weight 125 lbs.gross;110 lbs. net; measurements 6 cu.ft. Price 77.00

1, Vacuum Receiver, 16" x 60",steel, complete with automatic safety vacuum release valve, side connections and drain connections;weight 240 lbs.net and gross; measurements 12 cu.ft. Price ..... $ 96.00

1, Moisture Trap, 16" x 60",steel, complete with inlet and outlet connections,drain connections and necessary internal baffles;weight 170 lbs.net and gross; measurements 12 cu.ft. Price 64.00

The total of the above apparatus is ..... $ 4865.00

All of the above prices are FOB cars or FAS vessel at San Francisco, California, and include export packing and sectionalizing.

X. SUMMARY OF COSTS.

Flotation Equipment	$5422.10
Filtering Equipment	4865.00
2 - Dorr Thickener Mechanisms @ $735.00	1470.00
2 - Reinforced Concrete Tanks for Thickeners	2700.00
2 - Pachucha Agitators 5' dia. x 14' high	1000.00
2 - " " 6' " x 20' "	1400.00
3000 sq.ft. of concrete floorings, drains, etc.....	1272.60
Wooden frame structure with tile roofing (Lumber locally made in Company's saw mill)	5700.00
Pipes, pumps, accessories, etc.....	3500.00
Transportation Costs	3528.40
Installation Costs	4500.00
Motors, wiring, line shaft, pulleys	700.00
Consular Vise , 1½% of invoice value	360.44
	$ 36418.54

XI. TREATMENT COSTS.

Costs for mining ore were taken from Semi-annual Report June 30, 1921, and include total costs for mining, development and tramming to mill.

Averages for the first eleven months of 1921 were taken in figuring costs for slime transportation and also all mill operating costs.

% Extraction and consumption of chemicals were taken from results obtained in tests.

For value of slimes, the average value for the months, September to November, inclusive, were taken. These, only, were taken, because previous to September, slimes were taken from upper cut of the slime pit, which was of different material and higher grade.

Value of Contabria ore was taken from report on ore broken in stopes and represents an average of 35,000 tons.

Treatment costs calculated on a 100 ton plant, treating 50 tons Contabria ore and 50 tons of slimes.

	Sucres. Per ton	-- 50 tons.
Mining Costs	10.19	509.50
Crushing, grinding and classification	.794	39.70
Slime Transportation		
Supplies	.054	2.70
Labor	.205	10.25
Operation of slime mixer and preparing slime for flotation feed	.204	10.20
Flotation Costs (which includes flotation oils, thickening and filtering of concentrates by Oliver Filter)	.718	(100 tons) 71.80
Milling (exclusive of crushing, grinding and classification)		(30 tons)
Mill maintenance and repair	.666	20.04
Labor	.174	5.22
Agitation of concentrates in Pachucha agitators	.123	3.69
Chemicals used for treating 30 tons concentrates. NaCN 4 lbs. P/T @ .80	3.20	96.00
Mechanical loss NaCN in tails 5 lbs. @ .80	.40	12.00
NaOH - 5 lbs. @ .24	1.20	36.00
CaO - 2 lbs. @ .025	.05	1.50
Total costs		818.60

2.05 sucres = $1.00 " " ..... $ 399.27

XII. VALUES RECOVERED

	Per ton	50 tons.
Value Broken Ore	Au -10.75	537.50
	Ag = 3.21	160.50

Value lost in tails in concentrating

50 tons Contabria Ore Au = 37.24
Ag = 13.77

Value of concentrates- Au = 537.50 - 37.24 ..... 500.26

" " " - Ag = 160.50 - 13.77 ..... 146.73

% Extraction by cyaniding concentrates(taken from tests)

	Au	Ag
Sample # 2-	77.27	43.77
" #3 -	91.43	38.63
Average-	84.33	41.20

Value recovered in cyaniding concentrates from Contabria ore:

Au = 500.26 x.8433 .....421.87

Ag = 146.74 x.412 ..... 60.45

Total 482.32

Value recovered (Accumulated Slimes)

		50 tons.
Value of accumulated slimes -	Au = 2.71	135.50
	Ag = 1.23	61.50

Value lost in tails in concentrating the accumulated slimes - Au= 21.43
Ag= 9.22

Value of concentrates Au = 135.50 - 21.43 ..... 114.07
Ag = 61.50 - 9.22 ..... 52.28

% Extraction obtained by cyaniding concentrates from accumulated slimes-

Au = 90%
Ag = 40%

Value recovered in cyaniding concentrates from the accumulated slimes.

Au = 114.07 x.90 ..... 102.66
Ag = 52.28 x.40 ..... 21.31

Total 123.97

Total value extracted from Contabria Ore and slimes:

482.32 plus 123.97 ..... 606.29

Total profit per day: 606.29 - 399.27 ..... 207.01

From the above, it appears that the installation of the plant would be advisable.

XIII. TRANSPORTATION COSTS.

In order to obtain an idea on the transportation costs of equipment, the transportation costs of the flotation equipment, blower and filtering equipment, are itemized as an example:

Flotation Equipment, Salt Lake to San Francisco	$153.66
" " , San Francisco to Puerto Bolivar 12,275 lbs. at $18.00 P/T	110.39
Blower, Connersville, Ind. to New York City	27.45
Blower, New York City to Puerto Bolivar, 5490 lbs. at $12 P/T	32.94
Oliver Filter & Accessories, San Francisco to Puerto Bolivar, 13,070 lbs. at $18.00 P/T	117.63
Puerto Bolivar to Santa Rosa on River Steamer 30,835 lbs. at .256 per 100 lbs.....	78.94
Mule-back Transportation Santa Rosa to Mines 30,835 lbs. at $1.28 per 100 lbs.	497.08

It is seen that it costs almost as much to transport the equipment the last 45 miles on mule-back as it does for the entire distance from the States.

Owing to the high cost of transportation, bulky material should be avoided. For example, concrete is very expensive, and should be avoided as much as possible.

The following is the cost of cement:

Value of 1 bbl. cement in Los Angeles,Calif.....	$4.04
Cost of freight,Los Angeles to Puerto Bolivar $10.50 P/T....	$1.995
" " " Puerto Bolivar to Santa Rosa, per bbl.....	.973
Mule-back haul, Santa Rosa to mines	4.864
	$7.832

Total cost of cement is $11.872, or almost three times the original cost of the cement.

XIV. LABOR.

It is thought by many that owing to the fact that the labor is much cheaper in Ecuador than in the States, that the labor costs would be much less. This is not found to be true. The labor per man is cheaper; but, due to the fact that the natives do so much less work per day than an American, the labor costs figure out about the same. It is only after eight or ten years of training that they become efficient workers, and even then, they cannot be compared with American labor.

XV. APPENDIX. FLOTATION TESTS:

The following is a complete log of all tests previously made by the writer with a view of finding a successful method to treat the Contabria ore and base slimes:

TEST NO. 1.

OBJECT: To determine the oil or mixture of oils most suited for flotation tests on base slimes taken from old tailings pile.

Screen analysis of heads (+ 200 -- 21%
(- 200 -- 79%

Value Heads (Au. 4.03 Au. at 20.67
(Ag. 2.28 Ag. at .50

METHOD: The charge was put into the flotation machine, while running up to speed, and with air already turned on. Extra water was added to make up for water overflowing with froth. Each test was run under the same conditions as nearly as possible, with regard to speed of machine, time, etc.

The following oils or mixtures were used:

Pine Oil

Mixture #1	Crude Oil	40%
	Creoline	40%
	Pine Oil	20%
Mixture #2	Black Car Oil	40%
	Creoline	40%
	Pine Oil	20%
Mixture #3	Black Car Oil	40%
	Turpentine	40%
	Pine Oil	20%

Ore Grams	1950	Oil	Derived Data: Assay of Tails Au.	Assay of Tails Ag.	% Extraction Au.	% Extraction Ag.	Total Ex.
Water cc	16500	Pine Oil	1.65	1.04	59.0	54.4	57.3
Oil	3cc	Mix.#2	1.45	.37	64.0	83.7	71.1
Speed RPM	990	"	1.24	.40	69.2	82.4	74.0
Time Minutes	7	Mix.#1	1.03	.60	74.2	73.7	74.2
		"	1.24	.35	69.2	84.6	74.8
		Mix.#3	2.48	.51	58.5	77.6	53.2
		"	1.86	.44	55.8	80.7	63.5

CONCLUSIONS: 7 minutes was found to be more time than necessary for test as froth was coming over clean before the end of period. By reducing the amount of air, it might have been found that a longer period of agitation would be needed, giving cleaner concentrates as not so much gangue would be brought up with the concentrates due to the more violent agitation.

According to data Mixture # 1 gave the best results with an average total extraction of 74.5% for the two tests.

TEST No. 2.

OBJECT: To determine the oil or mixture of oils most suited for flotation tests on Cantabria Ore.

METHOD: The ore was crushed in "Dodge" crusher, at assay office, to 1/2" then through the disc-crusher to 12 mesh, then through experimental tube-mill to 79% -200 mesh. The experimental mill was connected for continuous grinding. Discharge from tube-mill was diluted to about 12% by weight of dry slimes then air-lifted to cone-classifier. Overflow from classifier flowed into settlingtank, and underflow returned to mill. Material flowing into settling-tank settled sufficiently so that clear overflow was returned to mill for re-use.

Assay of heads (Au. $5.59 Au. at 20.67
(Ag. 1.57 Ag. " .50

Oils used:- Mix. #1 of Ex. #1

Mix #1 followed by turpentine.

			Derived Data.				
			Assay of Tails		% Extraction		Total Ex.
Ore Grams	1900	Mix. #1	Au.	Ag.	Au.	Ag.	
Water	16500						
Oil	Variable	1cc	3.10	.82	44.5	47.8	45.2
Speed	1540	1 1/2cc	1.65	.25	70.5	84.1	73.4
Time Min.	7	2cc	1.05	.17	81.6	89.2	83.2

On examination 1cc of oil was seen to be not sufficient for good results, so tails from previous test using 1cc were reagitated with 3/4cc of turpentine and results recorded.

Value tails (Au. 1.65 % Ex. Au. 70.5
(Ag. .26 % Ex. Ag. 83.4 Total Ex. 73.3

Mixture #4 composed of the following oils was tried as a comparison with Mixture #1.

Mixture #4 (Stockholm oil of tar 40%
(Creoline 40%
(Pine Oil 20%

			Derived Data				
			Assay of Tails		% Extraction		Total
			Au.	Ag.	Au.	Ag.	Ex.
Ore Grams	1900	Oil					
Water	16500						
Oil	3 cc + 50 drops H_2SO_4	Mix.#1	.83	.15	85.2	90.4	87.6
Speed	3 cc + 50 drops H_2SO_4	Mix.#4	.62	.095	88.9	94.0	90.0

CONCLUSIONS: In first tests, at least two more tests should be run increasing the amount of oil (Mix. #1) as it appears that the maximum extraction had not been reached. Also comparative tests to show whether the H_2SO_4 added in last tests was an improvement or merely due to the extra amount of oil. The H_2SO_4 was added to neutralize the small amount of lime used for settling purposes. The solution in pulp titrated neutral before adding the acid.

Mixture #4 appears to be the best mixture of oils so far used.

TEST No. 3.

OBJECT: To decide whether Sulphuric Acid was an aid to good extraction as was left undecided from Ex. #2 and at the same time try out some new mixtures of oils.

The following mixtures were tried:

Mix. #6	(Stockholm oil of tar	20 parts
	(Turpentine	10 "
	(Creoline	10 "
	(Pine Oil	3 "
Mix. #7	(Castor Oil (Mineral)	40%
	(Creoline	40%
	(Pine Oil	20%
Mix. #9	(Castor Oil (Mineral)	2 parts
	(Creoline	1 "
	(Pine Oil	1 "

METHOD: Same as for Ex. #1 except that the duplicate test in each case was run with acid, (50 drops $H_2 SO_4$)

Ore Grams	1950	Oil Used	Derived Data				
			Value Tails		% Extraction		Total Ex.
Water	16500		Au.	Ag.	Au.	Ag.	
Oil	4 cc	Mix.#6	1.03	.245	86.9	89.2	87.4
Speed	1340+H_2SO_4		.62	.235	92.1	89.8	91.6
Air Pressure	1/2#	Mix.#7	.83	.180	89.4	92.1	90.0
Heads Au.	-7.86	"					
Ag.	-2.28 + H_2SO_4		1.03	.175	86.9	92.4	88.1
Au. at	20.67	Mix.#9	1.24	.380	84.2	83.3	84.0
Ag. at	.50	"					
Feed	84%-200 Mesh + H_2SO_4		.83	.330	89.4	85.5	88.6

CONCLUSIONS: The acid seemed to improve the extraction in the case of mixtures # 6 and 9 but with Mixture # 7 the extraction was less in using the acid than it was without it. With the above results, and with the results of the last two tests of Ex. #3, it is safe to say that the improvement in extraction, if any, would not warrant the extra expense of using it; and the extra cost of lime for neutralizing concentrates before cyaniding.

The oils used in this experiment did not appear to be an improvement over some of the oils used before. Mix. #7 would be a cheap oil to use and with proper regulation could no doubt be made to give as good results.

Creosote could probably replace creoline in the mixtures of oils used so far.

CALCULATIONS ON FLOTATION CONCENTRATION CONTABRIA ORE, SAMPLE # 1:

Unfortunately the sample for these calculations was not of sufficient size for exhaustive tests such as were run on the more base sample #2, but sufficient were run to justify in saying that an equally good extraction can be gotten under the same conditions.

Concentration 4.5 into 1.
Treating 4.5 tons of ore -(1 Ton concentrates
(3.5 Tons tails

Original Value of ore Per Ton -(9.10 Au. (Au. at 20.67
(2.42 Ag. (Ag. at .50

Value of Tails, Per Ton -(.90 Au.
(.22 Ag.

Value 3.5 Tons Tails -(3.15 Au.
(.77 Ag.

Total Value Lost - 3.92

Value of 4.5 Tons ore -(40.95 Au.
(10.89 Ag.

Total 51.84

% in ore saved -- $\frac{51.84 - 3.92}{51.84} \times 100 = 92.2$

Value of concentrates -- 51.84 - 3.92 = 47.92

Results obtained from assay on concentrates

Value Au. -- 39.28, Ag. -- 9.31, Total $ 48.59
(Which checks with calculated value of $47.92

CALCULATIONS ON FLOTATION CONCENTRATION CONTABRIA ORE, SAMPLE # 2.

Concentration 3 into 1.
Treating 3 tons ore -(1 Ton concentrates
(2 Tons tails

Original Value of Ore -(7.86 Au. Au. at 20.67
(2.28 Ag. Ag. at .50

Value of Tails -(.83 Au.
(.18 Ag.
Total 1.01

Value 2 tons tails -(1.86 Au.
(.36 Ag.
Total Value Lost 2.22

Total Value 3 tons ore -(23.58 Au.
(6.84 Ag.
Total 30.42

% in ore saved -- $\frac{30.42 - 2.22}{30.42} \times 100 = 92.7\%$

Value of concentrates -- 30.42 - 2.22 = 28.20

Assays taken from concentrates:

Au.	Ag.	Total Value.
18.85	6.28	25.13
19.85	7.22	27.07
21.09	7.22	28.31
18.81	6.78	25.59

Which checks closely with calculated value of $28.20

NOTE: It will be noted that tail assays lower in value were reported in experiments made on Sample #2 but a value was taken such as was thought that could be duplicated in practice. Also a concentrate of total value of $65.06 was obtained from sample #1, but the value taken was also such as could be duplicated in practice and still retain the extraction given.

TEST No. 4.

OBJECT: To determine the possibility of concentrating Contabria ore and then cyaniding the concentrates.

METHOD: Ore from sample #2 was ground to 84% - 200 mesh in experimental tube-mill.

Assay value of heads - Au. 7.86 Ag. 2.28 Ag. at .50

The ore was then concentrated in the Jones Experimental Flotation Machine using Mix. # 10 for the Flotation oil.

Mix. # 10	(Stockholm Oil of Tar	1 part
	(Creoline	1 part
	(Pine Oil	1 part

The oil was added in three parts at the rate of 3 lbs. oil per ton of ore and 1 1/2 lbs. $H_2 SO_4$ per ton added before agitation. One lb. oil per ton of ore being added each treatment. After agitating for another three minutes the 2nd. lb. per ton was added and the pulp agitated for another three minutes. The concentrates from the first two treatments were saved as cleaned product. The 3rd lb. of oil per ton was then added

and the pulp agitated until clean froth came over which was about another three minutes. The concentrates from the 3rd treatment was saved as middlings and added to the following charge:

The charge consisted of about 3900 grams of ore and about 280 grams of middlings, diluted to about 4 parts by weight of water to one of dry slime.

A sample of tailings was taken of each charge, and this was dipped into a bucket, and used as a composite sample for the day's run.

Value of tails	1st day's run	Au.	1.03	Ag.	.14
" " "	2nd " "	"	.87	"	.13
	Average		.95		.135

Value of concentrates	1st day's run	Au.	15.30	Ag.	3.64
" " "	2nd " "	"	13.54	"	3.27
	Average		14.37		3.45

Preliminary Treatment of concentrates:

Concentrates from flotation machine were de-watered several times by mixing them thoroughly and allowing concentrates to settle, then decanting the water. This was done with the idea of getting rid of considerable of the oil. The concentrates were then agitated violently with air in alkaline solution titrating 2.2 lbs. CaO per ton. The froth formed was collected and rebroken up with water, allowed to settle and returned to original charge. This preliminary agitation was done so as to reduce the amount lost in froth during the cyanide treatment.

The concentrates were then put in Experimental Pachuca Agitator and commenced agitating with lime solution titrating 1.6 lbs. CaO per ton. Pachuca agitation started at 1 P.M. Oct. 28.

Oct. 29 Titrated at	7 A.M.	.7	CaO per Ton.
	1 P.M.	.6	" " "

CYANIDING: Started cyaniding at 2:30 P.M. Oct. 29 in NaCN solution titrating 4.0 KCN per ton. Alkalinity brought up to 1.2 CaO per ton.

Oct. 30- 9.A.M. Alkalinity - .6 CaO per ton.
NaCN -2.6 KCN " "
Alkalinity brought up to 2.8 lbs. CaO per ton.

Oct. 31- 8 A.M. (2.4 lbs. CaO P/T
(2.5 " KCN P/T

3 P.M. (2.4 " CaO P/T
(1.8 " KCN P/T

NaCN brought up to 2 lbs. KCN P/T

Nov. 1 - 8 A.M. (1.8 CaO P/T
(1.8 KCN P/T

Nov. 2 - 2:30 P.M. (1.7 CaO P/T
(1.4 KCN P/T

NaCN increased to 1.6 KCN per ton
Agitator stopped at 2:30 P.M. Nov. 3. Solution titrated

Nov. 3 - 3.P.M. (1.4 CaO P/T
(1.4 KCN P/T

A sample was taken each day at 2:30 P.M. after the 2nd days agitation and the same assayed for gold and silver.

The following results were obtained:

	Gold	Silver
Original Concentrates	14.57	3.45
2 days agitation	3.31	1.84
3 " "	2.48	1.77
4 " "	1.65	1.74
5 " "	1.65	1.74

From the above it appears that more than 4 days agitation is useless. Fifures on cyanide consumption, etc., will be based on 4 day agitation.

DERIVED DATA: Concentration 2 into 1

Value 2 tons ore --	15.72 Au.	4.56 ag.	Total 20.28
" 1 ton conc.--	14.57 "	3.45 "	" 18.02
" 1 " tails --	.95 "	.135"	" 1.085

Saved in Cyaniding- 14.57 - 1.65 = 12.92 Au.
3.45 - 1.74 = 1.71 "

Total value Au. lost in 2 tons ore = 1.65 + .95 = 2.60
= 1.30 per ton.

Total value Ag. lost in 2 tons ore = 1.71 + .135 = 1.845
= $\frac{1.845}{2}$ = .92 per ton.

% Extraction of Au. in concentrating and cyaniding = $\frac{7.86-1.30}{7.86} \times 100$
= 83.4%

% Extraction of Ag. = $\frac{2.28 - .92}{2.28} \times 100 = 59.6\%$

Total extraction = $\frac{10.14 - 2.22}{10.14} \times 100 = 78.1\%$

Specific Gravity of concentrates calculated as follows:

(1)	Weight of bottle	39.4 Grams
(2)	" " " + H_2O	169.2 "
(3)	" " " + Conc	72.7 "
(4)	" " " + " + H_2O	192.9 "

Subtract (1) from (3) to get weight of conc. 72.7 - 39.4 = 33.3 Grams

" (4) from this to get wt. of water displaced 202.5-192.9 = 9.6 "

Divide wt. of conc. by wt. of water displaced to get Sp. Gr.
33.3 ÷ 9.6 = 3.45 Sp. Gr.

Let p equal percentage of dry material
D " density of pulp = 1.38
S " specific gravity of conc. = 3.45

$p = \frac{100\ S\ (D-1)}{D\ (S-1)} = \frac{100\ (138 - 1)\ 3.45}{138\ (3.45 - 1)} = 39.1$

Let q equal percentage of solution in charge of pulp.
q = 100 - 39.1 = 60.9

To calculate total cyanide consumed:

Oct. 29- 2:30 P.M. to Oct. 31- 3.P.M. = 4.0 - 1.8 =2.2 by titration.

Oct. 31- 3.P.M. to Nov. 2-2:30 P.M. = 2 - 1.4 =.6 by titration.

2.2 + .6 = 2.8 -- 2.8 x $\frac{60.9}{39.1}$ =4.36 lbs.HCN per T concentrates.

$\frac{4.36}{2}$ = 2.18 lbs. HCN per Ton of ore treated.

The lime required cannot be calculated accurately due to the method followed, but will approximate 2.025 lbs. CaO per Ton. On the basis of lime used 36% CaO = $\frac{2.025}{.36}$ = 5.63 lbs. lime per ton.
Much CaO was saved due to the water washes given the concentrates.

CONCLUSIONS: No re-agents were added in Cyanidation except lime and sodium cyanide. The solution was not changed during treatment. It is probable that if lead acetate had been added during treatment a higher extraction of the silver would have resulted. It is also probable that if the solution had been built up to a little higher strength, say 15% KCN, at the end of the 2nd days agitation and kept up to this strength, a higher extraction in gold and silver would have resulted. The protective alkali was increased too much at the end of the first days agitation with cyanide. A low alkalinity such as will test from .2 to .5 lbs. CaO per Ton in sulphide ores, results in less sulphur in solution, and a more rapid solution of the gold.

If the concentrates from the first treatment had been saved instead of from the first two, a much cleaner concentrate would have resulted. The concentrates from each agitation with the flotation machine were saved of one of the charges. They were filtered, dried and weighed separately and the following results noted:

	Weight	Value Au.	Value Ag.
1st treatment	333 grams	24.81	8.73
2nd- "	310 "	7.44	3.40
3rd- "	282 "	4.13	1.50

From the above it is seen that the concentrates from the 2nd treatment should have gone as middlings for re-treatment as they were of no better grade than the heads.

TEST. No. 5.

OBJECT: To determine whether a greater concentration than three into one of Contabria ore, sample #2 could be effected, or whether the ore is too base for a greater concentration.

METHOD: The concentrates, heads and tails from a previous experiment were tested for copper, iron, zinc, lead and sulphur to determine the % Sulphides. Derived Data:

	Concentrates	Heads	Tails
Value Au.	19.61	7.82	.83
Oz. Ag.	11.28	4.56	.36
% Insolubles	19.18	58.45	91.33
" Copper	2.59	.87	Trace
" Iron	17.83	11.45	3.84
" Zinc	16.08	7.77	1.34
" Lead	13.11	6.42	.90
" Sulphur	24.51	11.45	1.18
Undetermined %	6.70	3.59	1.41

From the above it is seen that a concentration greater than three into one would not be possible without a selective action for gold and silver. The above figures show a concentration of approximately 2.6 into 1 and to increase the concentration much more would reduce the insolubles to a prohibitive amount, and increase the values lost in the tails. The heads from sample #2 are at least 30% sulphides.

The concentration of three into one cannot be accomplished without retreating of a part of the concentrates as middlings, as will be shown from the following results obtained from equal charges of ore. Each charge receiving three treatments.

Mix. #9. Conc. collected from first two treatments separately.

1st and 2nd treatment.

Mt. Conc.	Value Au.	Oz. Ag.
558	21.07	14.44
767	15.30	11.06

3rd treatment.

222	10.75	6.06
301	7.03	3.24

Mix. # 10 Conc. collected from each treatment separately

Taken from Text No. 4.	Wt. Conc.	Value Au.	Oz. Ag.
1st treatment	333	24.81	17.46
2nd "	310	7.44	6.80
3rd "	282	4.13	3.00

Middlings taken from a compiled sample were taken and run through flotation machine without the addition of oil, and the cleaned concentrates collected, and also a sample taken of the tails. The following results were obtained:

	Middlings	Cleaned Conc.	Tails
Value Au	9.27	16.95	.83
Oz. Ag.	6.24	10.60	.76

From the above it is seen that fully the last half of the concentrates could be returned for retreatment without materially lowering the grade of the concentrates of the first half, or increasing the value of the tails.

TEST No. 6.

OBJECT: To obtain relative results between cyaniding Contabria ore and cyaniding only the concentrates thereof.

METHOD: A charge of concentrates was made up from concentrates collected from previous tests. The concentrates were first washed with a strong soultion of caustic soda to remove the oil. They were then allowed to settle and the solution drained to remove the surplus alkalinity. Water was then added to bring the specific gravity of pulp to 1.418, and charge put into agitator. Cyanide was then added.

For the Contabria ore a charge was made up from Sample #2, Sp.Gr.-1.332. This was agitated in alkaline solution for five days without cyanide to remove the latent acidity. At the end of the fifth day there were still substances in the ore, possibly sulphates of iron, which decomposed the lime. Cyanide was then added.

In the process of washing and decanting the concentrates were in alkaline solution for fully as long as the Contabria ore though not agitated.

Both concentrates and ore were treated in cyanide solution for a period of five days.

Dervied Data:

Columns (A)- Average strength of cyanide solution for each day of agitation in terms of KCN per ton.

Columns (B)- Average strength of alkalinity for each day of agitation in terms of CaO per ton.

Columns (C)- Consumption of cyanide for each day of agitation in terms of KCN per ton.

Columns (D)- Consumption of alkali for each day of agitation in terms of CaO per ton.

	A		B		C		D	
Day	Conc.	Ore	Conc.	Ore.	Conc.	Ore	Conc.	Ore.
1st	2.65	3.35	2.562	2.805	1.40	2.70	1.507	.330
2nd	3.60	4.75	.603	1.815	1.10	2.90	.000	.330
3rd	3.25	5.80	.452	1.320	1.10	2.75	.000	.165
4th	2.95	3.10	.452	.990	1.05	2.65	.151	.165
5th-	2.65	2.55	.452	.825	.90	2.60	.302	.000
Ave.	3.02	3.51	.904	1.551	1.09	2.72	.392	.198
Total			4.521	7.755			1.960	.990

Days Agitation	Value Conc.		Value Ore		%Extraction Conc.		%Extraction Ore		Total Extraction	
	Au.	Ag.	Au.	Ag.	Au.	Ag.	Au.	Ag.	Conc.	Ore
Original Value	22.53	5.78	7.86	2.28						
2	5.38		3.72		75.9		52.6			
3	2.89	3.59	no assay		87.1	37.9			76.9	
4	2.48	2.91	2.07	1.04	88.8	49.6	76.1	54.0	80.8	69.3
5	2.48	2.70	2.07	.87	88.8	53.3	76.1	63.2	81.5	71.0

CONCLUSIONS: From the above it appears that there is no dissolving of gold after 4 day's agitation and only slightly in the silver which conforms with results in cyaniding of TEST No. 4.

On the basis of four days agitation we have a cyanide consumption of 4.069 for the concentrates and 6.930 for the ore.

The concentrates show a concentration of 3 into one. Cyanide consumed per ton of ore in cyaniding concentrates = $\frac{4.069}{3}$ = 1.356 lbs. per ton as against 6.930 for the cyaniding of the ore without concentration.

A greater consumption of cyanide for the treatment of the ore may have been partly due to the fact that the solution in the treatment of the ore was not changed after the five days agitation with the lime solution. To obtain more nearly relative results the agitator should have been stopped and the solution decanted before the cyanide was added. By so doing the agitator could have been started again with a solution titrating about the same in alkalinity as that of the concentrates.

The cyanide consumption for the 2nd day's agitation was probably higher than it should have been in the case of the ore, due to a much higher cyanide strength for that day, but it is plainly seen that the ore was consuming more cyanide than the concentrates because the strength in cyanide of the ore fell below that of the concentrates on the 5th day. No cyanide was added to either agitator after the 2nd day.

It is also seen that the consumption of cyanide for the concentrates was much greater for the 1st day and also that the alkali consumption was much the greatest. This shows that if the concentrates had been agitated instead of merely washed and decanted, the latent acidity would have been reduced more, and would probably have decreased the cyanide consumption for that day.

TEST No. 7.

OBJECT: To note result of cyaniding concentrates from Contabria ore using a more dilute pulp, a more violent agitation, and a low strength cyanide solution.

METHOD: A charge of concentrates collected from previous experiments were washed in alkaline solution containing caustic soda, to break up the froth, and to give the concentrates a preliminary wash, the same as for Test #6. The concentrates were then put into agitator and agitated for five hours previous to adding the cyanide. Cyanide strength was increased the 2nd day the same as in Test No. 6. At the end of the 4th day the agitator was emptied and tailings assayed.

Derived Data:

Value of Concentrates (Au. - 19.85
(Ag. - 4.92

Cyanide added Nov. 15, 3 P.M. Titration (CaO - 1.0
(KCN - 3.2

Nov. 16, 3 P.M. (CaO - 1.0 increased to (CaO - 1.0
(KCN - 2.0 (KCN - 2.3

Nov. 17, 3 P.M. (CaO - 1.0
(KCN - 2.0

Nov. 18, 3.P.M. (CaO - .9
(KCN - 1.7

Nov. 19, 3 P.M. (CaO - .9
(KCN - 1.6

Let Column (A) equal average strength of cyanide for each day.
" (B) " cyanide consumed per ton of sol. for each day
" (C) " " " " " " ore " " "

The alkali consumed for the four days was a negligible amount, being only .3 lbs. per ton calculated in CaO. This was due to the thorough preliminary treatment given the concentrates before cyaniding.

A	B	C	
2.60	1.2	5.21	Sp.Gr. conc. = 3.80
2.15	.3	.80	Density of pulp = 1.254
1.85	.3	.80	% ore = 27.2
1.65	.1	.27	% sol.= 72.8
Average 2.063	Total-1.9	Total 5.08	

Assay of heads (Au. = 19.85
(Ag. = 4.92

" " tails (Au. = 2.27
(Ag. = 2.96

% Extraction Au. = $\frac{19.85 - 2.27}{19.85} \times 100 = 88.56$

" " Ag. = $\frac{4.92 - 2.96}{4.92} \times 100 = 40.0\%$

Total " -= $\frac{24.77 - 5.23}{24.77} \times 100 = 80.1\%$

CONCLUSIONS: The extraction in gold checks very closely with that of Test No. 6, but that of the silver is lower. This shows that the cyanide strength was sufficiently strong for the extraction of the gold, but for the dissolving of the silver, a cyanide strength equal to that used for the cyaniding of the ore in Test No. 6 is necessary. Such a strong solution would prove uneconomical due to the high consumption of cyanide and a probable fouling of the solutions.

The higher consumption of cyanide noted in this experiment over that obtained for the concentrates of Test No. 6 was evidently due to the more violent agitation necessary to keep the more dilute pulp in suspension.

Tests with some of the concentrates showed that the settling capacity increased very rapidly on diluting the pulp below a Sp. Gr. of 1.40. The most satisfactory Sp. Gr. will probably prove to be between 1.40 and 1.45.

TEST NO. 8.

OBJECT: To determine the oils most suited for flotation concentration of accumulated slimes of lower pond.

METHOD: Same as for Test. No. 1 except that oil was added in three parts, at intervals of about 3 minutes each.

Ore grams per charge 3000
Water 16000
Speed, R.P.M. 1340
Time for each test, minutes 9
New mixtures of oils tried:

Mix. # 15	(Crude Oil	2	parts
	(Creoline	1	"
	(Pine Oil	1	"
Mix. # 16	(Stockholm oil of tar	4	parts
	(Creoline	2	"
	(Turpentine	1	"
	(Pine Oil	1	"
Mix. #17	(Stockholm oil of tar	4	"
	(Creoline	2	"
	(Kerosene	1	"
	(Pine Oil	1	"
Mix. # 14	(Black Car Oil	4	"
	(Creoline	4	"
	(Pine Oil	1	"
Mix. # 18	(Crude Oil	40%	
	(Kerosene	20%	
	(Creoline	20%	
	(Turpentine	20%	
Mix. # 8	(Stockholm oil of tar	60%	
	(Kerosene	20%	
	(Pine Oil	20%	

Test #	Oils Used	Amt. Oil Used	Value Heads Au.	Value Heads Ag.	Value Conc. Au.	Value Conc. Ag.	Value Tails Au.	Value Tails Ag.	% Extraction Au.	% Extraction Ag.	Total Extraction.
1	Mix.#1	5cc.	3.31	1.34	8.27	4.20	.62	.20	81.3	85.1	82.4
2	" "	4 1/2cc.	"	"	6.20	3.27	1.03	.17	69.0	87.3	74.2
3	" 4	"	"	"	5.58	2.69	.83	.18	74.9	86.5	78.3
4	" 9	"	"	"	13.83	5.81	.83	.20	74.9	85.1	77.8
5	" 14	3cc	3.51	1.23	7.03	4.96	.83	.30	76.3	75.6	76.2
6	" "	"	"	"	-	5.09	1.03	.33	70.7	73.2	71.3
7	" 15	"	"	"	16.12	12.58	1.03	.28	70.7	77.2	72.3
8	" "	"	"	"	17.78	13.59	1.45	.25	58.7	79.7	63.7
9	" "	"	"	1.43	16.95	7.98	1.03	.38	70.7	75.4	71.1
10	" "	4 1/2cc.	"	"	10.75	4.79	.83	.30	76.3	79.0	77.1
11	" 16	3cc.	"	"	8.68	4.68	.62	.17	82.6	88.1	84.0
12	" 17	4 1/2cc	"	"	24.81	13.31	.62	.19	82.6	86.7	83.6
13	" "	"	2.27	1.45	7.44	4.86	.41	.22	81.9	84.8	83.1
14	" 8	"	"	"	6.62	5.34	.62	.21	72.7	85.5	77.7
15	" "	"	"	"	7.44	5.43	.62	.28	72.7	80.7	75.8
16	" 18	"	"	"	11.58	8.09	.83	.22	67.8	81.9	71.5
17	" "	"	"	"	9.51	8.19	.62	.33	72.7	77.2	71.8

Concentrates were collected from first treatment.

Of the nine mixtures of oils tried #17 appears to give the best results as a good grade of concentrates were obtained with very low value in tails.

TEST No. 9.

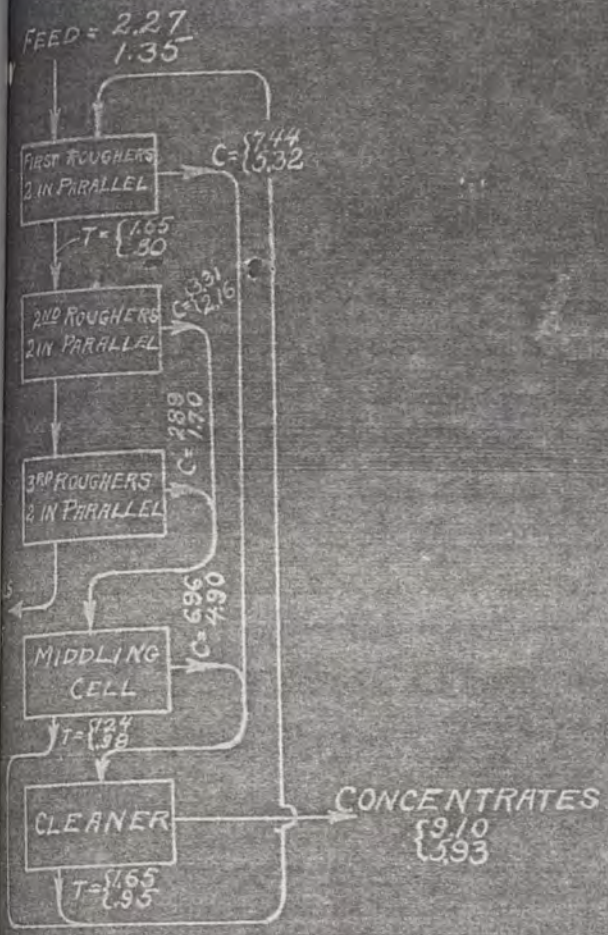
OBJECT: To work out a probable flow-sheet and to determine value of concentrates and tails in the different parts of circuit.

METHOD: The flow-sheets shown in the attached blue print were duplicated as nearly as possible by saving concentrates and tails from each test and retreating them according to the desired flow-sheet and at the same time saving samples of each test for assays. Samples were taken over a period of several days and averages taken to obtain the assay values shown on the blue print. The concentrates were collected and will be used for future experiments on cyaniding of concentrates.

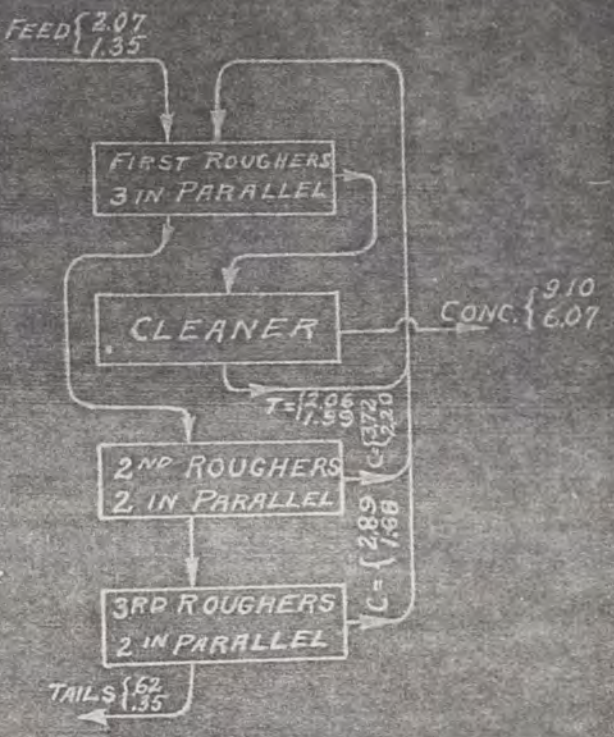
For example, to duplicate flowsheet A of blue print two charges of slimes were run through experimental flotation machine, giving the slimes three treatments of 3 minutes each. The concentrates from the first treatment and from the second and third treatments collected separately. The concentrates from the second and third treatments were retreated without the addition of extra oil, representing the middling cell and the concentrates collected mixed with the concentrates of the first treatment and the entire lot recleaned. The resulting concentrates would represent the final product. The tailings from the re-treating of the concentrates were mixed equally with the feed for the following two charges and the procedure repeated several times.

Calculations on the basis of the flow-sheet A to determine what the values of concentrates should be for slimes assaying $2.70, $3.00 and $3.30 per ton Au. respectively.

FLOW SHEETS FOR 8 CELLS



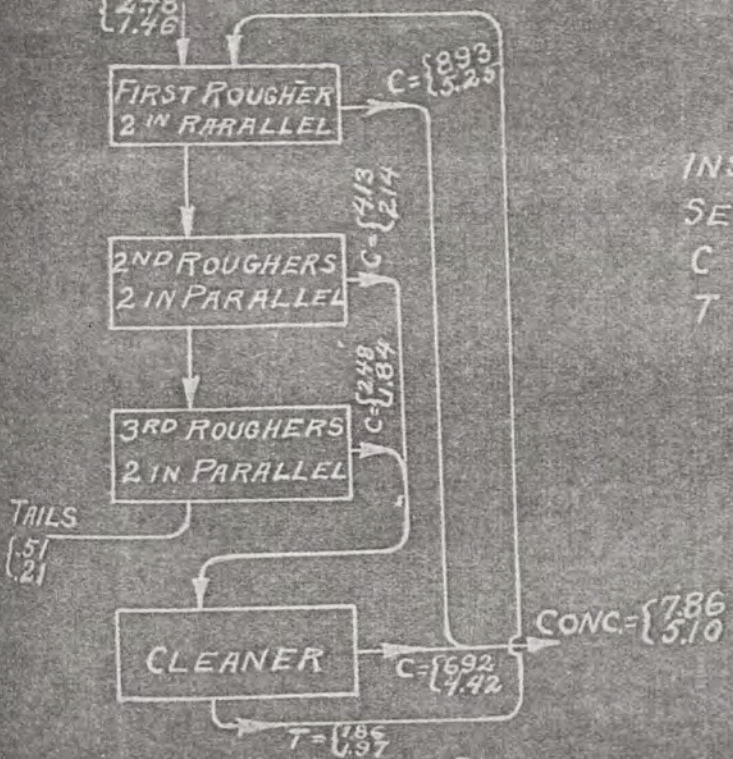
FLOW SHEET A



FLOW SHEET B.

FEED 2.48 / 1.46

FLOW SHEET FOR 7 CELLS



FLOW SHEET C.

NOTES:-

THE FIRST FIGURE IN EACH INSTANCE DENOTES VALUE OF Au. SECOND FIGURE VALUE Ag @ $.50

C DENOTES CONCENTRATES.,

T " TAILS.

Value of heads Au. = 2.57
" " tails " = .51
" saved per ton = 1.76

Tons slime required to produce 1 ton concentrates= $\frac{910}{1.76}$ = 5.17

% Extraction Au.= $\frac{2.27 - .51}{2.27} \times 100$ = 77.7%

(A) Value of Heads Au. = 2.70
" " tails " = 2.70 x (1 - .777) = .60
" saved per ton = 2.70 - .60 = 2.10

Calculated value of concentrates obtained)=2.10 x 5.17=$10.85 per T

(B) Value of Heads Au. = 3.00
" " tails " = 3.00 x (1 - .777) = .67
" saved per ton = 3.00 - .67 = 2.33

Calculated value of concentrates = 2.33 x 5.1 = $12.04

(C) Value of Heads Au. = 3.30
" " tails " = 3.30 x (1 - .777) = .74
" saved per ton = 3.30 - .74 = 2.56

Calculated value of concentrates = 2.56 x 5.17 = 13.24

TEST No. 10.

OBJECT: To determine the amount of cyanide and lime consumed, and extraction obtained in the direct cyaniding of the accumulated slime. (These data to be used for comparative results in later concentrating and cyaniding tests).

METHOD: Three samples of slime were taken; two from the lower, and one (the last) from the more base upper slime pile. The charges were made up; lime and cyanide added with no preliminary treatment given.

Derived data: Sample #1. Sp. Gr. of charge = 1.308 Sp. Gr. of slime = 2.60.

% Dry Slimes = 37.93
% Solution = 62.07

Titrations on starting agitator = (KCN = 3.4 #
(CaO = 1.1

Titrations after 48 hrs. agitation = (KCN = 1.4 #
(CaO = 0.5

KCN consumed per ton Sol.= 2.0 lbs.

" " " " dry slimes = $\frac{2.0 \times 62.07}{37.93}$ = 3.27 lbs.

CaO " " " sol. = .6 lbs.

CaO " " " dry slimes = $\frac{.6 \times 62.07}{37.93}$ = .98 lbs.

Value of heads (Au. = $ 2.79
(Ag. = 1.33

Value of tails after 48 hrs agitation (Au. = $ .41
(Ag. = .65

% Extraction Au. = $\frac{2.79 - .41}{2.79} \times 100$ = 86.30

% Extraction Ag. = $\frac{1.33 - .65}{1.33} \times 100$ = 51.12

Total Extraction = $\frac{4.12 - 1.06}{4.12} \times 100$ = 74.27 %

Sample # 2: Sp. Gr. of charge = 1.275

% Dry Slimes = 35.05
% Solution = 64.95

Titrations on starting agitator (KCN = 2.4#
(CaO = 1.5

" after 48 hrs. agitation (KCN = 1.0
(CaO = 1.1

KCN consumed per ton sol. =1.4#
" " " " dry slimes = $\frac{1.4 \times 64.95}{35.05}$ = 2.594 lbs.

CaO " " " Solution = .4
" " " " dry slimes = $\frac{.4 \times 64.95}{35.05}$ = .741 lbs.

Value of heads (Au. = $1.96
(Ag. = 1.20

Value of tails after 48 hrs. agitation (Au. = $ .10
(Ag. = .58

% Extraction Au. = $\frac{1.96 - .10}{1.96} \times 100$ = 94.82

% " Ag. = $\frac{1.20 - .58}{1.20} \times 100$ = 51.66

Total " = $\frac{3.06 - .68}{3.06} \times 100$ = 77.77%

Sample # 3: (Base Slimes) Sp. Gr. of Charge = 1.294

% Slimes = 35.96
% Solution = 64.04

Titrations on starting agitator (KCN = 3.5#
(CaO = 2.7

" after 48 hrs. agitation (KCN = 1.1
(CaO = 2.2

KCN consumed per ton sol. = 2.4 lbs.

" " " " dry slimes = $\frac{2.4 \times 64.04}{35.96}$ = 4.24 lbs.

CaO " " " sol. = .5 lbs.

" " " " dry slimes = $\frac{.5 \times 64.04}{35.96}$ = .89 lbs.

Value of heads (Au. = $ 2.07
(Ag. = 1.06

Value of tails after 48 hrs. agitation ($ .21
($ .51

% Extraction Au. = $\frac{2.07 - .21}{2.07} \times 100 = 89.85$

% Extraction Ag. = $\frac{1.06 - .51}{1.06} \times 100 = 51.88$

Total Extraction = $\frac{3.13 - .72}{3.13} \times 100 = 77.00\%$

CONCLUSION : The cyanide consumption for the treatment of the accumulated slime of lower pile (samples # 1& 2) check very closely with previous mill tests, which gave a cyanide consumption of approximately three lbs. per ton of dry slime. For sample # 3 of this test the cyanide consumption is much lower than in previous mill tests on the base slime, probably due to the varied amount of oxidised material on the different parts of the pile

TEST No. 11.

OBJECT: To determine the consumption of cyanide and lime and extraction obtained in cyaniding the concentrates from accumulated slime.

METHOD: Concentrates were agitated in an alkaline solution, titrating 3 lbs. CaO per ton for five hours, then KCN was added. A solution of high

cyanide strength was used at the beginning of agitation without building it up in strength during treatment and no change of solution was made during agitation.

Derived data:

% Dry Concentrate = 27.88
% Solution = 72.12

Value of heads (Au. = $ 6.62
(Ag. = 4.61

Date Dec.	Titrations KCN #	CaO #	KCN Consumed P/T Sol.	KCN Consumed P/T Conc.	Value Tails Au.	Ag.	% Extraction Au.	Ag.	Total
8	3.5	1.7			6.62	4.61			
9	2.1	1.6	1.4#	3.61#	.62	3.03	90.72	34.25	67.50
10	1.5	1.6	.6	1.55	.31	2.89	95.32	37.31	71.50
11	1.2	1.6	.3	.77	.21	2.67	96.83	42.08	74.35
Total			2.3	5.93					

(Ag. Calculated at .50)

Cyanide consumption for the first two days agitation was 5.16 lbs.per ton of dry concentrate. The percent extraction for the third days agitation was not sufficient to offset the extra consumption of cyanide.

Alkali consumption for the preliminary agitation was 3.36 lbs. Calculated in terms of CaO per ton and only .258 lbs. during the cyaniding, or a total of 3.618 lbs. for the entire treatment. The slime used to obtain the concentrate for this test corresponded very close to Sample # 2 of Test No. 10.

CONCLUSION: From this test it appears that a preliminary agitation of five hours is sufficient to reduce almost all the latent acidity when an alkaline solution titrating three pounds CaO per ton is used.

TEST No. 12.

OBJECT: Same as for Test No. 11.

METHOD: Concentrate agitated in alkaline solution titrating 3 lbs. CaO per ton for five hours then KCN added. A low strength cyanide solution was used for the first 24 hours, and then built up at the beginning of the 2nd day. No change of solution made during treatment.

Derived data: Value Head (Au. = $ 7.86
(Ag. = 5.36

Sp. Gr. of Charge = 1.250
% Dry Concentrates=32.0
% Solution =68.0

Date	Titrations KCN	Titrations CaO	KCN increased	KCN consumed P/T sol.	KCN consumed P/T dry conc.	Value Tails Au. $	Value Tails Ag. $	% Extraction. Au.	% Extraction. Ag.	% Extraction. Total
14	2.1	1.8	#	#	#	7.86	5.36			
15	.6	1.5	2.7	1.5	3.253	2.48	4.74	68.44	11.56	45.38
16	1.5	1.5		1.2	2.550	.51	3.81	93.51	28.91	67.32
17	.9	1.5		.6	1.775	.51	3.61	93.51	32.74	68.83

Cyanide consumption for the first two days agitation was 5.803 lbs per ton of dry concentrate.

Agitation for the third day was useless as there was no extraction in gold and only a small amount in silver.

Alkali consumption for the preliminary agitation was 2.55 lbs. calculated in terms of CaO per ton and .637 lbs. during the cyaniding or a total of 3.187 lbs. for the entire treatment.

CONCLUSION: The method used for this test did not appear to be an improvement over that of Test. No. 11 although a smaller cyanide consumption was noted the first day with the weaker solution, but it was offset by the cyanide consumption of the second day.

TEST No. 13.

OBJECT: To determine the ratio of concentration and per cent extraction of the base slime from the upper pile.

METHOD: Cut samples were taken of the slime as shown in attached sketch. The samples were run through flotation machine, and the value of heads, concantrates and tails recorded. Flow sheet C of Test No. 9 was used for this test except that tails from cleaner were discarded.

Derived Data:

Sample	Value Heads Au.	Ag.	Value Conc. Au.	Ag.	Value Tails Au.	Ag.	% Extraction Au.	Ag.	Total
1	4.76	1.54	7.44	3.90	2.07	.73	56.51	52.59	55.55
2	4.86	1.38	12.82	5.03	2.07	.61	57.41	55.79	57.05
3	2.48	1.44	9.10	6.00	1.03	.40	58.47	72.22	63.54
4	2.48	1.15	6.62	4.50	1.03	.37	58.47	67.83	61.43
5	2.07	1.34	6.62	4.80	.83	.44	59.90	67.16	62.75

Three samples of sand were taken from stratified layers between the slime and the following assays were given:

Sample #	Value Au.	Oz. Ag.
1	6.20	1.32
2	5.79	1.32
3	6.20	1.50

Concentrates show a concentration of about 3 to 1.

CONCLUSION: Samples # 1 & 2 show a much greater value in tails and poorer extraction, probably due to a considerable amount of sand being mixed with the slime. This sand is of a good grade in value as shown above and could easily be classified out and treated with the accumulated sand.

Concentrate from this test plus an extra sample taken at a cut through Samples # 2 & 3 will be used for Test No. 14.

TEST No. 14.

OBJECT: To note the effect of decanting the alkaline solution used in the preliminary agitation before adding the cyanide.

METHOD: Concentrate agitated in alkaline solution of caustic soda, titrating 1.8 lbs. in terms of CaO per ton, for 6 hours. Agitator then emptied and charge diluted to 3 times its original volume and then allowed to settle for 17 hours. Surplus solution was then decanted and charge put back into agitator and NaCn added. Concentrate used in this test was obtained from concentration of the base slime (See Test No. 13).

Derived Data: Sp. Gr. of charge = 1.223

% Dry Concentrate = 29.15
" Solution = 70.85

Value Heads (Au. = 10.75
(Ag. = 6.36

Date Dec.	Titrations KCN	Titrations CaO	Increased to KCN	Increased to CaO	KCN consumed P/T sol.	KCN consumed P/T dry conc.	Value Tails Au.	Value Tails Ag.	% Extraction Au.	% Extraction Ag.	% Extraction Total
21	1.9	2.0					10.75	6.36			
22	.5	1.7	2.2	2.0	1.4	3.402	1.24	4.62	88.46	27.35	65.75
23	1.4	1.8			.8	1.944	1.03	3.97	90.42	37.58	70.78
24	1.1	1.5			.3	.729	.83	3.73	92.27	41.51	73.34

Cyanide consumed for 48 hours agitation = 5.346 lbs. P/T.

Alkaline consumed in preliminary agitation and decantation =3.645 lbs. in terms of CaO per ton, and 1.215 lbs. during the cyanide treatment, or a total of 4.860 lbs. for the entire treatment.

CONCLUSION: This test shows that the latent acidity in the concentrate was not entirely reduced in the preliminary treatment, because there was over a pound of alkali in terms of CaO consumed during the cyanide treatment. A stronger alkali solution in the preliminary treatment would probably have reduced the cyanide consumption. This test also shows that the concentrate from the base slime should not consume more cyanide than the concentrate from the slime of the lower pile.

TEST No. 15.

OBJECT: Same as for Test No. 13.

METHOD: Same as for Test No. 13 except that the preliminary agitation was for 24 hours instead of 6, and that a stronger alkaline solution was used both in the preliminary agitation and in the cyanide treatment. Concentrate used in this test was obtained from slime taken from slime bin Nov. 28 &29.

Approximate value (Au. = 2.48
(Ag. = 1.40

Derived Data: Sp. Gr. of charge = 1.210
% Dry Concentrate = 27.88
% Solution = 72.12
Value Heads (Au. = 8.27
(Ag. = 4.94

Date	KCN	CaO	KCN consumed P/T sol.	KCN consumed P/T Conc.	Value Tails Au.	Value Tails Ag.	% Extraction Au.	% Extraction Ag.	% Extraction Total
Nov. 30	3.5	2.6			8.27	4.94			
Dec. 1	2.6	2.5	.9	2.327					
2	2.1	2.5	.5	1.293	.62	3.55	93.71	28.12	68.43
3	1.85	2.5	.25	.647	.62	3.40	93.71	33.18	69.56

Cyanide consumption for two days agitation was only 3.630 pounds per ton of concentrate. The third days agitation was useless as there was no extraction in gold and only a small amount in silver.

Calculated on the assumption that the decanted solution from the preliminary agitation was rejected, the alkali consumption for the preliminary agitation and decantation was 9.312 pounds in terms of CaO per ton, and .258 pounds for the cyanide treatment or a total of 9.560 pounds for the entire treatment.

CONCLUSION: The method used in this test reduced the cyanide consumption by about two pounds over the method used in Tests No. 11 & 12, but the excessive alkali consumption would make the method prohibitive in practice. Also the slow settling qualities of the slime would require an excessive amount of settling area.

TEST No. 16.

OBJECT: To note the effect of using excessively strong alkali solution of caustic soda in the preliminary agitation and decantation.

METHOD: Charge was made up of concentrate from slime taken from slime bin. The concentrate was allowed to stand two days in strong alkaline solution titrating about 9 pounds CaO per ton. Surplus solution was then decanted and charge put into agitator, and agitated in the alkaline solution for 24 hours. Agitator was then emptied and pulp diluted with water to about three times its own volume. The concentrate was then allowed to settle

for 24 hours and solution decanted. The charge was then put back into agitator.

Derived Data:
% Solution in charge = 68.0
% Dry Concentrate = 32.0
Value Heads (Au. = 9.10
(Ag. = 6.12

Date	Titrations KCN	Titrations CaO	Increased to KCN	Increased to CaO	KCN consumed P/T sol.	KCN consumed P/T dry conc.	Value Tails Au.	Value Tails Ag.	% Extraction Au.	% Extraction Ag.	% Extraction Total
Dec.											
4	2.3	2.0					9.10	6.12			
5	.5	1.7	2.0	1.7	2.0	4.25	3.31	5.89			
6	.5	1.7	1.5	1.7	1.5	3.18	2.89	3.12	68.24	49.02	60.51
7	.5	1.6			1.2	2.55	1.03	3.06	88.89	50.00	73.12
			Total		4.7	9.98					

Cyanide consumed for 48 hrs. agitation = 7.43 pounds per ton.

CONCLUSION: This test showed that too strong an alkaline solution was injurious. Before the end of the first days agitation in the KCN solution the concentrate began to change from a black to an almost buff color which showed that they were undergoing decomposition, probably oxidising. The result was a high cyanide consumption and low extraction. The test showed conclusively that if an alkaline solution were used to break up the froth that the concentrates should be cyanided immediately and not allowed to stand in the alkaline solution for any considerable time.

NOTE: This test was repeated with the same results. A high cyanide consumption was noted and low extraction. Also the same change of color of the concentrates was noted.

TEST. No. 17.

OBJECT: To determine the amount of cyanide and lime consumed and per cent extraction obtained in the cyaniding of the concentrates from the accumulated slime, (upper pile). Part of the same sample of the heads used for obtaining the concentrate was saved for Test. No. 18. This was done so that relative results could be obtained between cyaniding the concentrate and cyaniding the slime directly.

METHOD: Concentrate was agitated in alkaline solution titrating 3.4 lbs. CaO per ton for four hours then KCN added. A low strength cyanide solution was used for the first 24 hours, then built up at the beginning of the second days agitation. No change of solution was made during treatment.

Derived data:
% Dry concentrate = 32.64
% Solution = 67.36
Value heads (Au. = 11.16
(Ag. = 5.82

Date	Titrations KCN	Titrations CaO	Increased to KCN	Increased to CaO	KCN consumed P/T sol.	KCN consumed P/T conc.	Value Tails Au.	Value Tails Ag.	% Extraction Au.	% Extraction Ag.	% Extraction Total
Jan.											
2	2.10	2.8					11.16	5.82			
3	.50	2.3	2.60	2.3	1.60	3.296					
4	1.50	2.0			1.10	2.266	.51	3.88	95.43	33.33	74.14
5	1.00	1.8			.50	1.030	.31	3.64	97.22	37.45	76.73

KCN consumed the first 48 hours agitation = 5.562 pounds.

Alkali consumed in terms of CaO per ton for the preliminary treatment was 1.296 pounds per ton. During cyanide treatment 2.060 pounds were consumed, or a total of 3.356 pounds per ton.

CONCLUSION: The method used in this test corresponds very closely with the methods used in Tests, Nos. 11 & 12. From the above tests it is shown that the cyanide consumption is about the same for the treatment of the concentrates obtained from the accumulated base slime (upper pile) as it is for the treatment of the concentrates obtained from the accumulated slime of the lower pile.

TEST No. 18.

OBJECT: To determine the amount of cyanide and lime consumed, and per cent extraction obtained in the direct cyaniding of accumulated base slime.

METHOD: The sample of base slime saved out from Test No. 18 was used. The charge was made alkaline, and KCN added at the beginning of agitation.

Derived data: Sp. Gr. of charge = 1.294
% Dry Slime = 36.65
% Solution = 63.35
Value Heads (Au. 3.31
(Ag. 1.23

Date	Titrations KCN	Titrations CaO	Increased to KCN	Increased to CaO	KCN consumed P/T sol.	KCN consumed P/T slime	Value Tails Au.	Value Tails Ag.	% Extraction. Au.	% Extraction. Ag.	% Extraction. Total
Jan.											
2	2.6	1.1					3.31	1.23			
3	0.8	0.4	2.2	1.1	1.4	3.110					
4	1.3	.7			.9	1.555	.51	.58	84.59	52.84	76.0

KCN consumed = 4.665 pounds per ton.

CaO consumed = 1.901 " " "

From data obtained from Test No. 17 and from this test, the ratio of concentration of the base slime is calculated as follows:

Taken from "Hamelton's" Cyanide Hand Book

Let R = Ratio of concentration
C = Concentrates
T = Tails from the cyaniding of concentrates
H = Heads
t = Tails from the cyaniding of the ore.

$$\text{then } R = \frac{C-T}{H-t} = \frac{16.98 - 4.89}{4.54 - .109} = \frac{12.59}{3.45} = 3.56$$

The cyanide saved by concentrating (See Test No. 17)

$$= 4.665 - \frac{5.562}{3.56} = 3.103 \text{ pounds per ton of dry slimes.}$$

TEST No. 19.

OBJECT: To note the effect of cyaniding the base slime with the oxidized material eliminated as much as possible.

REMARKS: There are considerable layers of sand mixed in with the base slime. In these layers it seems that the air has attacked the slime and oxidized them owing to the more porous media of the sand. In taking the sample for this test the oxidized layers were left out, and as much as possible of the oxidized material which remained in the sample was washed out by a preliminary wash. The sample was then cyanided.

Derived data: Sp. Gr. Charge = 1.322
% Dry Slime =37.05
% Solution =62.95

Date	Titrations KCN	Titrations CaO	Increased to KCN	Increased to CaO	KCN Consumed P/T sol.	KCN consumed P/T slime	Value Tails Au.	Value Tails Ag.	% Extraction Au.	% Extraction Ag.	% Extraction Total
Dec.											
26	2.2	1.8					3.82	1.58			
27	1.1	1.0	2.1	1.0	1.0	1.699	1.45	1.41	62.04	10.76	47.04
28	1.4	0.9			.7	1.189	.62	.64	83.77	59.49	76.29

KCN consumed P/T of dry slime = 2.888
CaO consumed = 1.529

CONCLUSION: This test shows conclusively that it is the oxidized material which consumes considerable of the cyanide, and as shown in Test No. 18, the cyanide consumption was reduced from 4.665 pounds to 2.888 pounds per ton of dry slime.

In previous tests on Contabria ore only two samples were used. A third sample was gotten to obtain relative results as to extraction, ratio of concentration, cyanide consumption, etc.

The third sample was obtained by dumping two car loads of Contabria ore on the ground outside of the tunnel, and a 300 pound grab sample taken.

The sample was crushed, ground to 71% - 200 mesh in the experimental mill and used for the following tests.

TEST No. 20.

OBJECT: To note the effect of treating Contabria ore (sample #3) with a high protective alkalinity.

METHOD: Two charges of the ore were made up, the first with a high alkalinity and the second with a low alkalinity.

Derived data: (A) Charge with high alkalinity.

Sp. Gr. of charge = 1.394
% Dry ore =42.83
% Solution =57.17

Value heads (Au.= 4.65
(Ag.= 2.09

Date	Titrations KCN	Titrations CaO	Increased to KCN	Increased to CaO	KCN consumed P/T sol.	KCN consumed P/T ore	Value Tails Au.	Value Tails Ag.	% Extraction Au.	% Extraction Ag.	% Extraction Total
Dec.											
26	2.9	3.3					4.65	2.09			
27	1.2	2.9	2.3	2.9	1.7	2.269	1.24	1.17	73.33	44.02	64.24
28	1.8	2.6			.5	.673	.41	1.02	91.18	51.20	78.78
29	1.6	2.5			.2	.266	.41	1.00	91.18	52.15	79.08

KCN consumed the first 48 hours agitation = 2.942
CaO " " " " " " = .950

(B) Charge with low alkalinity
Sp. Gr. of charge = 1.371
% Dry Ore =41.02
% Solution =58.98

Date	Titrations KCN	Titrations CaO	Increased to KCN	Increased to CaO	KCN consumed P/T sol.	KCN consumed P/T ore	Value Tails Au.	Value Tails Ag.	% Extraction Au.	% Extraction Ag.	% Extraction Total
Dec.											
26	3.5	1.3					4.65	2.09			
27	1.3	1.0	2.4	1.0	2.5	3.283	1.24	1.12	73.33	46.41	64.98
28	1.9	.9			.5	.719	.41	.99	91.18	52.20	79.23
29	1/5	.8			.4	.575	.41	.95	91.18	54.11	79.82

KCN consumed the first 48 hours agitation = 4.002 pounds per ton.

CaO " " " " " " = .515 " " "

CONCLUSION: The test shows that there is less cyanide consumed with a high protective alkalinity than there is with the low alkali strength solution. The latter test was made with about the same strength alkali solution as is used in the mill and shows that if the ore were direct treated in the mill it would consume at least 4 pounds of cyanide. The cyanide consumption would probably be more than four pounds because the sample taken was ground in water instead of cyanide solution and the excess water decanted before the cyanide was added. This gave the ore a preliminary wash.

TEST No. 21.

OBJECT: To determine the cyanide and lime consumed and the per cent extraction obtained from the cyaniding of the concentrates from sample #3 of Contabria ore.

METHOD: A charge of concentrates were made, put into agitator and agitated in alkaline solution of caustic soda, titrating 3.9 pounds CaO per ton for five hours. Cyanide was then added, and charge agitated in cyanide solution for three days.

Derived data: Sp. Gr. of charge = 1.427
% Dry concentrates = 40.59
% Solution = 59.41 Value Heads (Au.=14.47
(Ag.= 7.30

Date	Titrations KCN	CaO	Increased To KCN	CaO	KCN consumed P/T sol.	KCN consumed P/T conc.	Value Tails Au.	Ag.	% Extraction Au.	Ag.	Total
Dec.											
29	2.4	3.4					14.47	7.50			
30	.7	3.3	2.05	3.3	1.7	2.463	2.88	5.75	80.10	21.23	60.36
31	1.35	3.3	2.05	3.3	.7	1.024	1.24	4.48	91.43	38.63	73.72
Jan.											
1	1.50	3.2			.55	.805	1.24	4.32	91.43	40.00	74.46

KCN consumed the 1st 48 hrs. agitation=3.487 lbs. per ton of dry conc.

Alkali consumed in terms of CaO for the preliminary treatment =.744 pounds per ton of dry concentrates. Alkali consumed during the 48 hours agitation in cyanide solution = .148 pounds or a total of .892 pounds per ton of dry concentrates.

Using "Hamilton's" formula for ratio of concentration (See Test No.18) the following is obtained.

$$R = \frac{C-T}{W-t} = \frac{21.77 - 5.72}{6.72 - 1.40} = \frac{16.05}{5.32} = 3.01$$

CONCLUSION: From results obtained in this test and in Test No. 20 it appears that Sample #3 of Contabria ore was not as refractory as Sample #2 because a good extraction was obtained from the gold with comparatively low cyanide strength in 48 hours agitation.

CALCULATIONS ON CONCENTRATING & CYANIDING OF ACCUMULATED SLIME.

According to Flow Sheet A of Test No. 9 the slime of lower pile will concentrate in the ratio of 4 into 1.

As noted in Tests. Nos. 11 & 12, by giving concentrates a preliminary

agitation of 5 hours to break up the froth and to neutralize acid substances before adding cyanide, the cyanide consumption will be about 5 1/2 pounds per ton of concentrates or about 1.375 pounds per ton of slime.

To treat the slime direct, the cyanide consumption is about 3 lbs. per ton of slime.

The saving in cyanide is then about 1.6 lbs. per ton by concentrating before cyaniding.

The alkali consumption is about the same in either method but to treat the slime direct lime is used, and to treat the concentrates caustic soda would undoubtably have to be used to break up the froth which would probably be more expensive.

According to Test. No. 13, the base slime will concentrate in the ratio of 3 into 1.

The cyanide consumption in cyaniding the concentrates would be $\frac{5.5}{3}$ = 1.833 per ton of slime.

To treat the base slime direct will consume 4 1/4 pounds cyanide per ton (See Test No. 10).

The saving in cyanide is then 2.4 pounds per ton by concentrating before cyaniding.

What appears to be the chief advantage of concentrating before cyaniding would be the saving in space in the mill.

To treat 220 tons of mine ore and sands, and 20 tons of concentrates, or a total of 240 tons, would allow for a 36 hour agitation.

To treat the same tonnage of mine ore and sands and the same value in slime would mean a total of 300 tons and reduce the period of agitation in the agitators by seven hours.

Frank Silva