United States Patent [19]

Weston

[11] **3,919,079**

[45] ***Nov. 11, 1975**

[54] **FLOTATION OF SULPHIDE MINERALS FROM SULPHIDE BEARING ORE**

[76] Inventor: **David Weston,** 34 Parkwood Ave., Toronto, Canada

[*] Notice: The portion of the term of this patent subsequent to May 29, 1990, has been disclaimed.

[22] Filed: **Mar. 7, 1973**

[21] Appl. No.: **339,384**

Related U.S. Application Data

[63] Continuation-in-part of Ser. No. 267,205, June 28, 1972, abandoned, Continuation-in-part of Ser. No. 270,961, July 19, 1972, Pat. No. 3,735,931, Continuation-in-part of Ser. No. 304,540, Nov. 7, 1972, abandoned.

[52] **U.S. Cl.** **209/166;** 209/3

[51] **Int. Cl.**[2] **B03D 1/02**

[58] **Field of Search** 209/166, 167, 3, 4, 9

[56] **References Cited**

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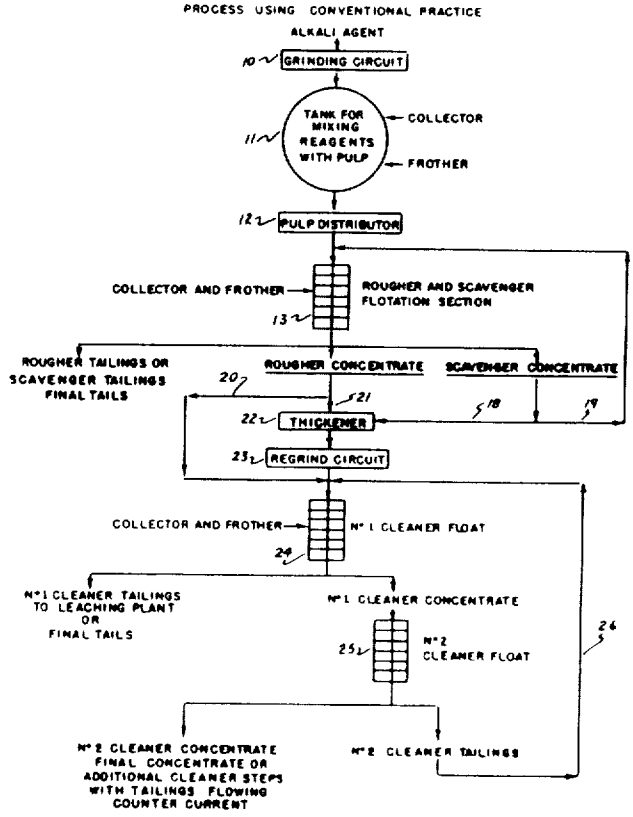
Primary Examiner—Robert Halper

Attorney, Agent, or Firm—Depaoli & O'Brien

[57] **ABSTRACT**

A process for the recovery by flotation of sulphide minerals from ores whereby a pulp of the ground ore is conditioned in at least one agitation conditioning stage in the pH range of about 8.0 to 12.0 in the presence of a sulfhydryl anionic collector, after which the conditioned pulp is subjected to flotation to produce a concentrate high in the desired sulphide minerals recovery and a tailings impoverished in the desired sulphide minerals.

91 Claims, 19 Drawing Figures



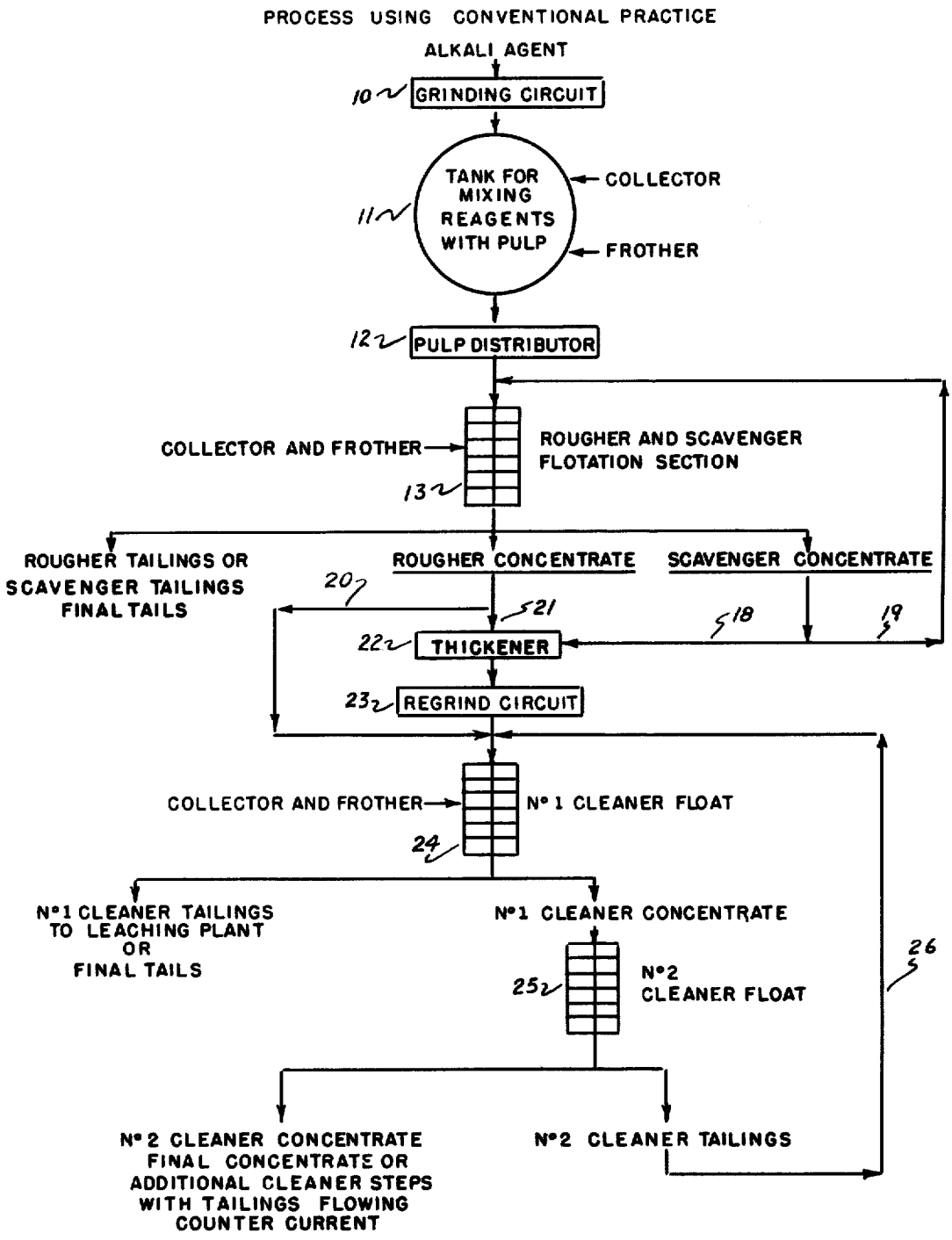

PROCESS USING CONVENTIONAL PRACTICE
ALKALI AGENT
10
GRINDING CIRCUIT
TANK FOR MIXING REAGENTS WITH PULP
COLLECTOR
FROTHER
11
12
PULP DISTRIBUTOR
COLLECTOR AND FROTHER
ROUGHER AND SCAVENGER FLOTATION SECTION
13
ROUGHER TAILINGS OR SCAVENGER TAILINGS FINAL TAILS
ROUGHER CONCENTRATE
SCAVENGER CONCENTRATE
20
21
18
19
22
THICKENER
23
REGRIND CIRCUIT
COLLECTOR AND FROTHER
N° 1 CLEANER FLOAT
24
N°1 CLEANER TAILINGS TO LEACHING PLANT OR FINAL TAILS
N°1 CLEANER CONCENTRATE
25
N°2 CLEANER FLOAT
26
N° 2 CLEANER CONCENTRATE FINAL CONCENTRATE OR ADDITIONAL CLEANER STEPS WITH TAILINGS FLOWING COUNTER CURRENT
N°2 CLEANER TAILINGS

FIG 1

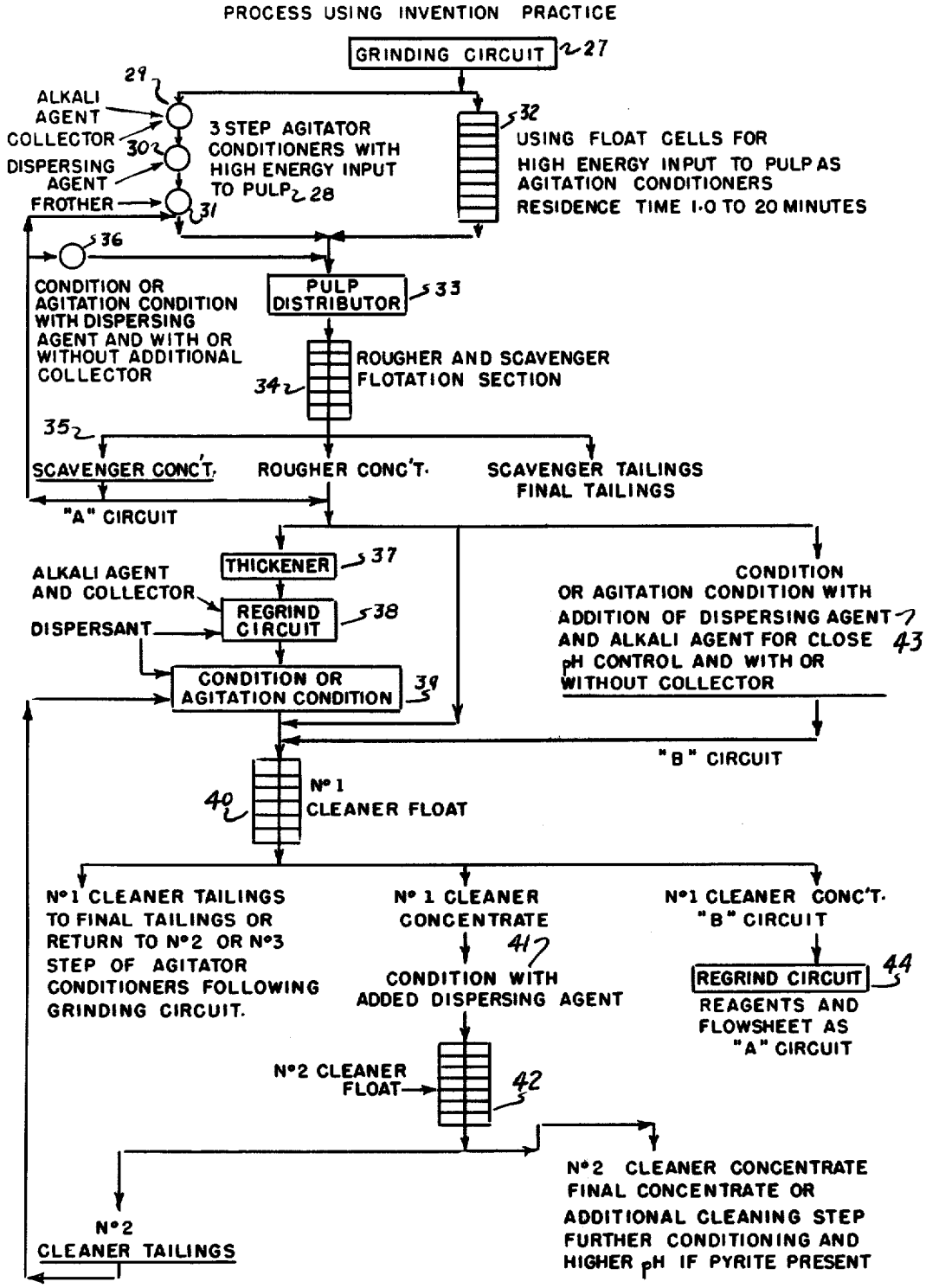

PROCESS USING INVENTION PRACTICE
GRINDING CIRCUIT
27
29
ALKALI AGENT
COLLECTOR
DISPERSING AGENT
FROTHER
30
31
3 STEP AGITATOR CONDITIONERS WITH HIGH ENERGY INPUT TO PULP
28
32
USING FLOAT CELLS FOR HIGH ENERGY INPUT TO PULP AS AGITATION CONDITIONERS RESIDENCE TIME 1.0 TO 20 MINUTES
36
CONDITION OR AGITATION CONDITION WITH DISPERSING AGENT AND WITH OR WITHOUT ADDITIONAL COLLECTOR
PULP DISTRIBUTOR
33
34
ROUGHER AND SCAVENGER FLOTATION SECTION
35
SCAVENGER CONC'T.
ROUGHER CONC'T.
SCAVENGER TAILINGS FINAL TAILINGS
"A" CIRCUIT
THICKENER
37
ALKALI AGENT AND COLLECTOR
REGRIND CIRCUIT
38
DISPERSANT
CONDITION OR AGITATION CONDITION
39
CONDITION OR AGITATION CONDITION WITH ADDITION OF DISPERSING AGENT AND ALKALI AGENT FOR CLOSE pH CONTROL AND WITH OR WITHOUT COLLECTOR
43
"B" CIRCUIT
40
N° 1 CLEANER FLOAT
N°1 CLEANER TAILINGS TO FINAL TAILINGS OR RETURN TO N°2 OR N°3 STEP OF AGITATOR CONDITIONERS FOLLOWING GRINDING CIRCUIT.
N° 1 CLEANER CONCENTRATE
41
CONDITION WITH ADDED DISPERSING AGENT
N°1 CLEANER CONC'T. "B" CIRCUIT
REGRIND CIRCUIT
44
REAGENTS AND FLOWSHEET AS "A" CIRCUIT
N°2 CLEANER FLOAT
42
N°2 CLEANER CONCENTRATE FINAL CONCENTRATE OR ADDITIONAL CLEANING STEP FURTHER CONDITIONING AND HIGHER pH IF PYRITE PRESENT
N°2 CLEANER TAILINGS

FIG 2

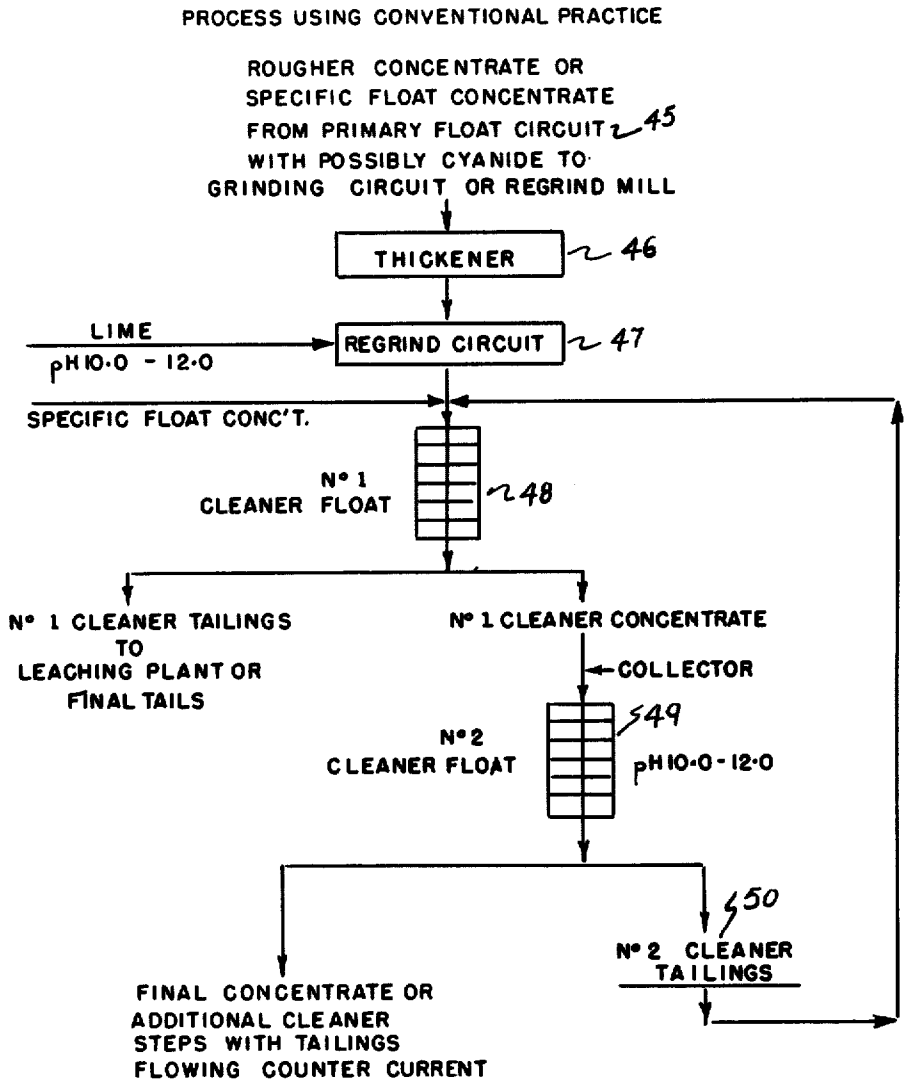

PROCESS USING CONVENTIONAL PRACTICE
ROUGHER CONCENTRATE OR
SPECIFIC FLOAT CONCENTRATE
FROM PRIMARY FLOAT CIRCUIT
WITH POSSIBLY CYANIDE TO
GRINDING CIRCUIT OR REGRIND MILL
45
THICKENER
46
LIME
pH 10.0 - 12.0
REGRIND CIRCUIT
47
SPECIFIC FLOAT CONC'T.
N° 1
CLEANER FLOAT
48
N° 1 CLEANER TAILINGS
TO
LEACHING PLANT OR
FINAL TAILS
N° 1 CLEANER CONCENTRATE
COLLECTOR
49
N° 2
CLEANER FLOAT
pH 10.0 - 12.0
50
N° 2 CLEANER
TAILINGS
FINAL CONCENTRATE OR
ADDITIONAL CLEANER
STEPS WITH TAILINGS
FLOWING COUNTER CURRENT

FIG 3

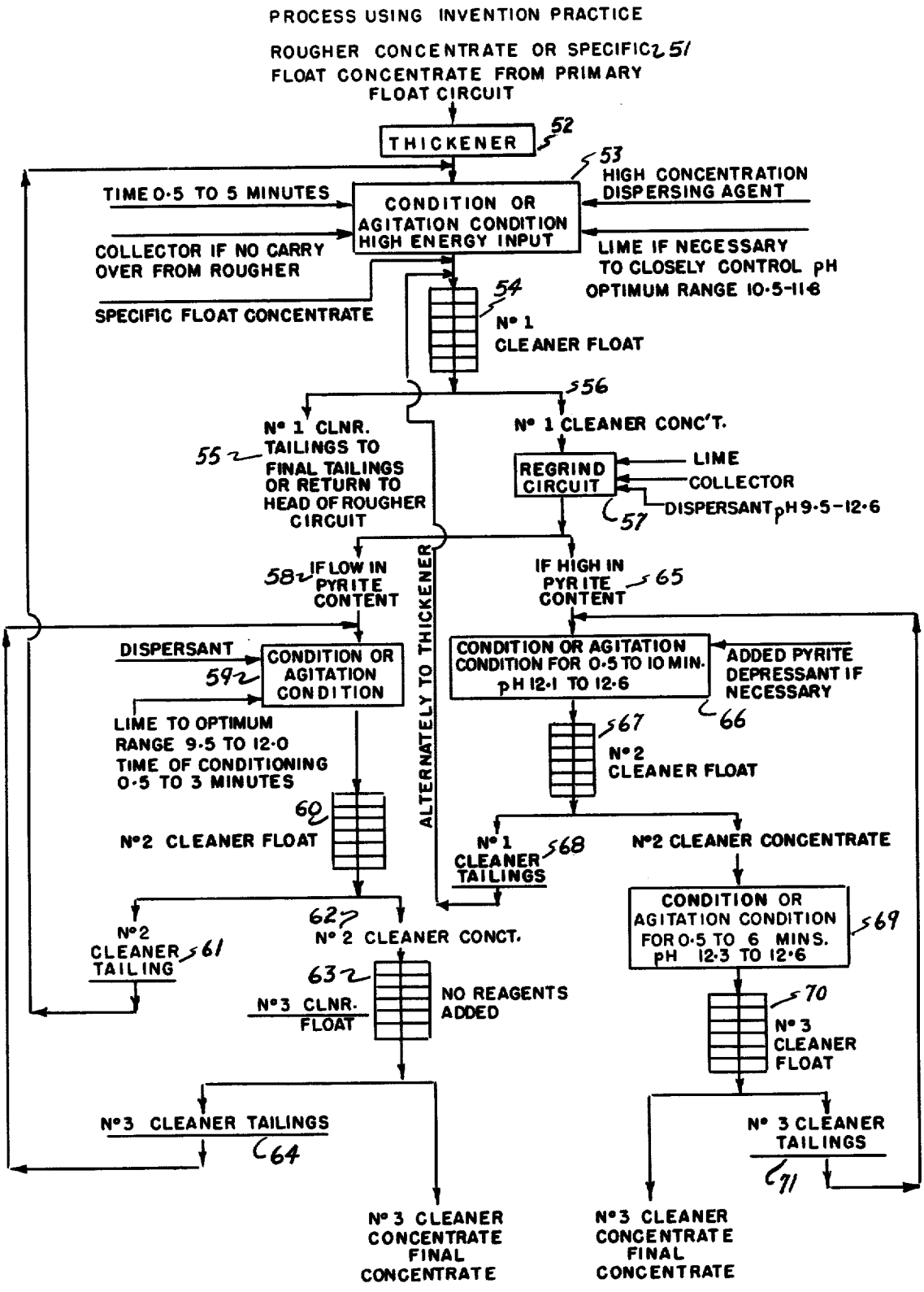

PROCESS USING INVENTION PRACTICE
ROUGHER CONCENTRATE OR SPECIFIC FLOAT CONCENTRATE FROM PRIMARY FLOAT CIRCUIT
51
THICKENER
52
53
HIGH CONCENTRATION DISPERSING AGENT
TIME 0·5 TO 5 MINUTES
CONDITION OR AGITATION CONDITION HIGH ENERGY INPUT
COLLECTOR IF NO CARRY OVER FROM ROUGHER
LIME IF NECESSARY TO CLOSELY CONTROL pH OPTIMUM RANGE 10·5-11·6
SPECIFIC FLOAT CONCENTRATE
54
N° 1 CLEANER FLOAT
56
N° 1 CLNR. TAILINGS TO FINAL TAILINGS OR RETURN TO HEAD OF ROUGHER CIRCUIT
55
N° 1 CLEANER CONC'T.
REGRIND CIRCUIT
LIME
COLLECTOR
DISPERSANT pH 9·5-12·6
57
IF LOW IN PYRITE CONTENT
58
IF HIGH IN PYRITE CONTENT
65
ALTERNATELY TO THICKENER
DISPERSANT
59
CONDITION OR AGITATION CONDITION
LIME TO OPTIMUM RANGE 9·5 TO 12·0
TIME OF CONDITIONING 0·5 TO 3 MINUTES
CONDITION OR AGITATION CONDITION FOR 0·5 TO 10 MIN. pH 12·1 TO 12·6
ADDED PYRITE DEPRESSANT IF NECESSARY
66
67
N°2 CLEANER FLOAT
60
N°2 CLEANER FLOAT
N° 1 CLEANER TAILINGS
68
N°2 CLEANER CONCENTRATE
62
N° 2 CLEANER CONCT.
N°2 CLEANER TAILING
61
CONDITION OR AGITATION CONDITION FOR 0·5 TO 6 MINS. pH 12·3 TO 12·6
69
63
N°3 CLNR. FLOAT
NO REAGENTS ADDED
70
N° 3 CLEANER FLOAT
N°3 CLEANER TAILINGS
64
N° 3 CLEANER TAILINGS
71
N°3 CLEANER CONCENTRATE FINAL CONCENTRATE
N°3 CLEANER CONCENTRATE FINAL CONCENTRATE

FIG 4

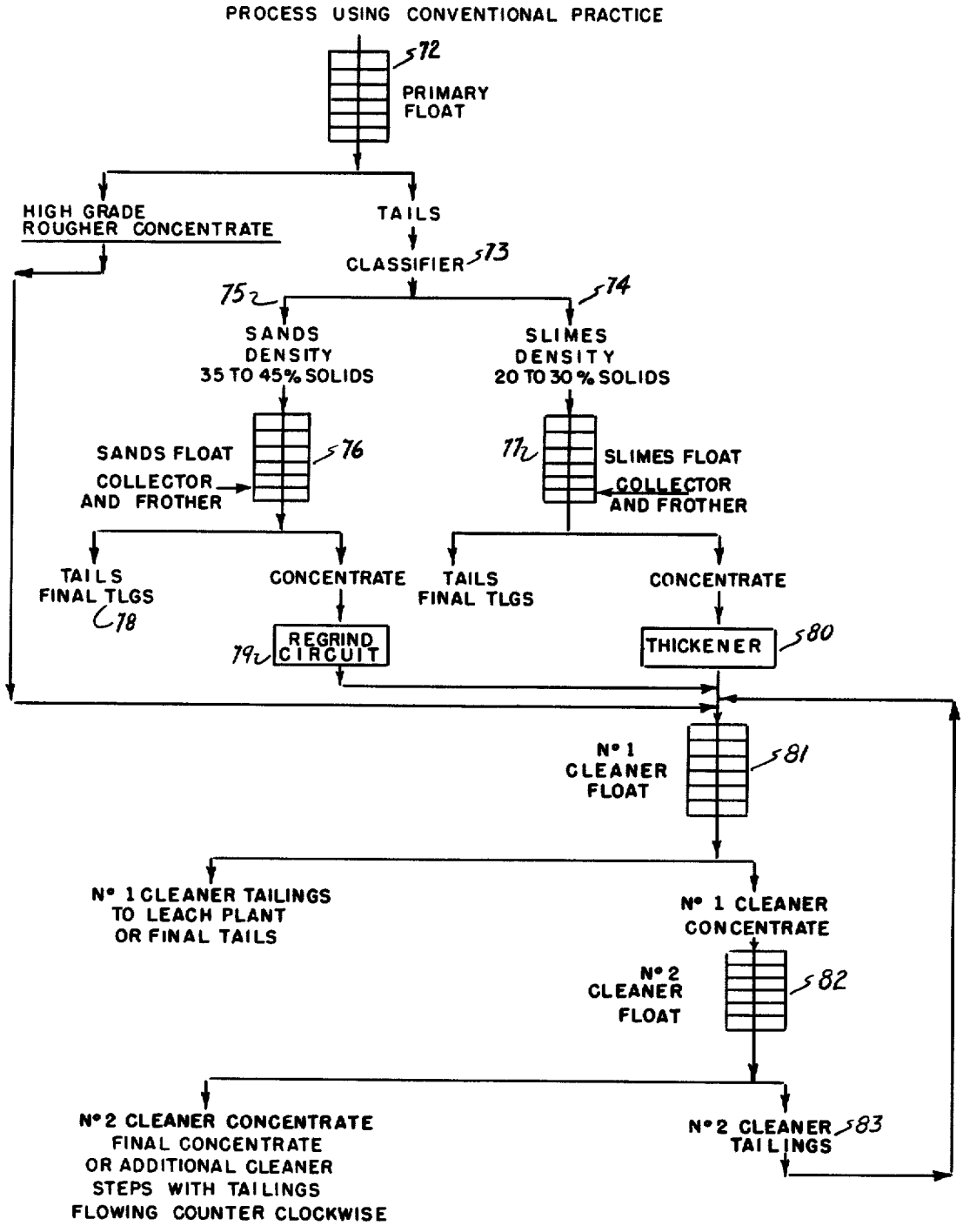

PROCESS USING CONVENTIONAL PRACTICE
72
PRIMARY FLOAT
HIGH GRADE ROUGHER CONCENTRATE
TAILS
CLASSIFIER 73
75
74
SANDS DENSITY 35 TO 45% SOLIDS
SLIMES DENSITY 20 TO 30% SOLIDS
SANDS FLOAT COLLECTOR AND FROTHER
76
77
SLIMES FLOAT COLLECTOR AND FROTHER
TAILS FINAL TLGS
78
CONCENTRATE
TAILS FINAL TLGS
CONCENTRATE
79
REGRIND CIRCUIT
THICKENER
80
N° 1 CLEANER FLOAT
81
N° 1 CLEANER TAILINGS TO LEACH PLANT OR FINAL TAILS
N° 1 CLEANER CONCENTRATE
N° 2 CLEANER FLOAT
82
N° 2 CLEANER CONCENTRATE FINAL CONCENTRATE OR ADDITIONAL CLEANER STEPS WITH TAILINGS FLOWING COUNTER CLOCKWISE
N° 2 CLEANER TAILINGS
83

FIG 5

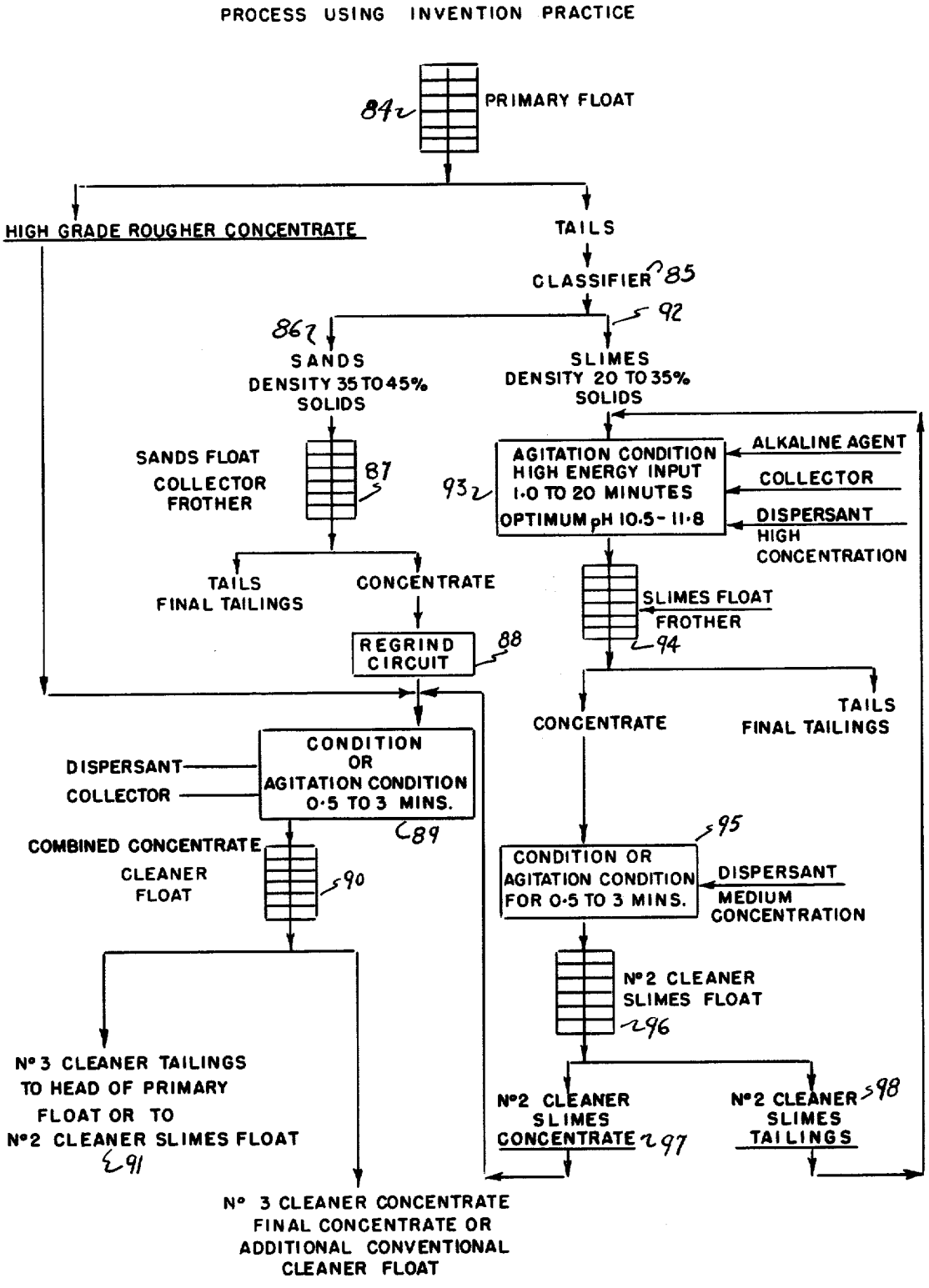

PROCESS USING INVENTION PRACTICE
84
PRIMARY FLOAT
HIGH GRADE ROUGHER CONCENTRATE
TAILS
CLASSIFIER 85
86
92
SANDS
DENSITY 35 TO 45% SOLIDS
SLIMES
DENSITY 20 TO 35% SOLIDS
SANDS FLOAT
COLLECTOR
FROTHER
87
AGITATION CONDITION
HIGH ENERGY INPUT
1.0 TO 20 MINUTES
OPTIMUM pH 10.5 - 11.8
93
ALKALINE AGENT
COLLECTOR
DISPERSANT
HIGH CONCENTRATION
TAILS
FINAL TAILINGS
CONCENTRATE
SLIMES FLOAT
FROTHER
94
REGRIND CIRCUIT
88
CONCENTRATE
TAILS
FINAL TAILINGS
DISPERSANT
COLLECTOR
CONDITION OR AGITATION CONDITION 0.5 TO 3 MINS.
89
95
COMBINED CONCENTRATE
CLEANER FLOAT
90
CONDITION OR AGITATION CONDITION FOR 0.5 TO 3 MINS.
DISPERSANT
MEDIUM CONCENTRATION
Nº2 CLEANER SLIMES FLOAT
96
Nº 3 CLEANER TAILINGS TO HEAD OF PRIMARY FLOAT OR TO Nº2 CLEANER SLIMES FLOAT
91
Nº2 CLEANER SLIMES CONCENTRATE
97
Nº2 CLEANER SLIMES TAILINGS
98
Nº 3 CLEANER CONCENTRATE FINAL CONCENTRATE OR ADDITIONAL CONVENTIONAL CLEANER FLOAT

FIG 6

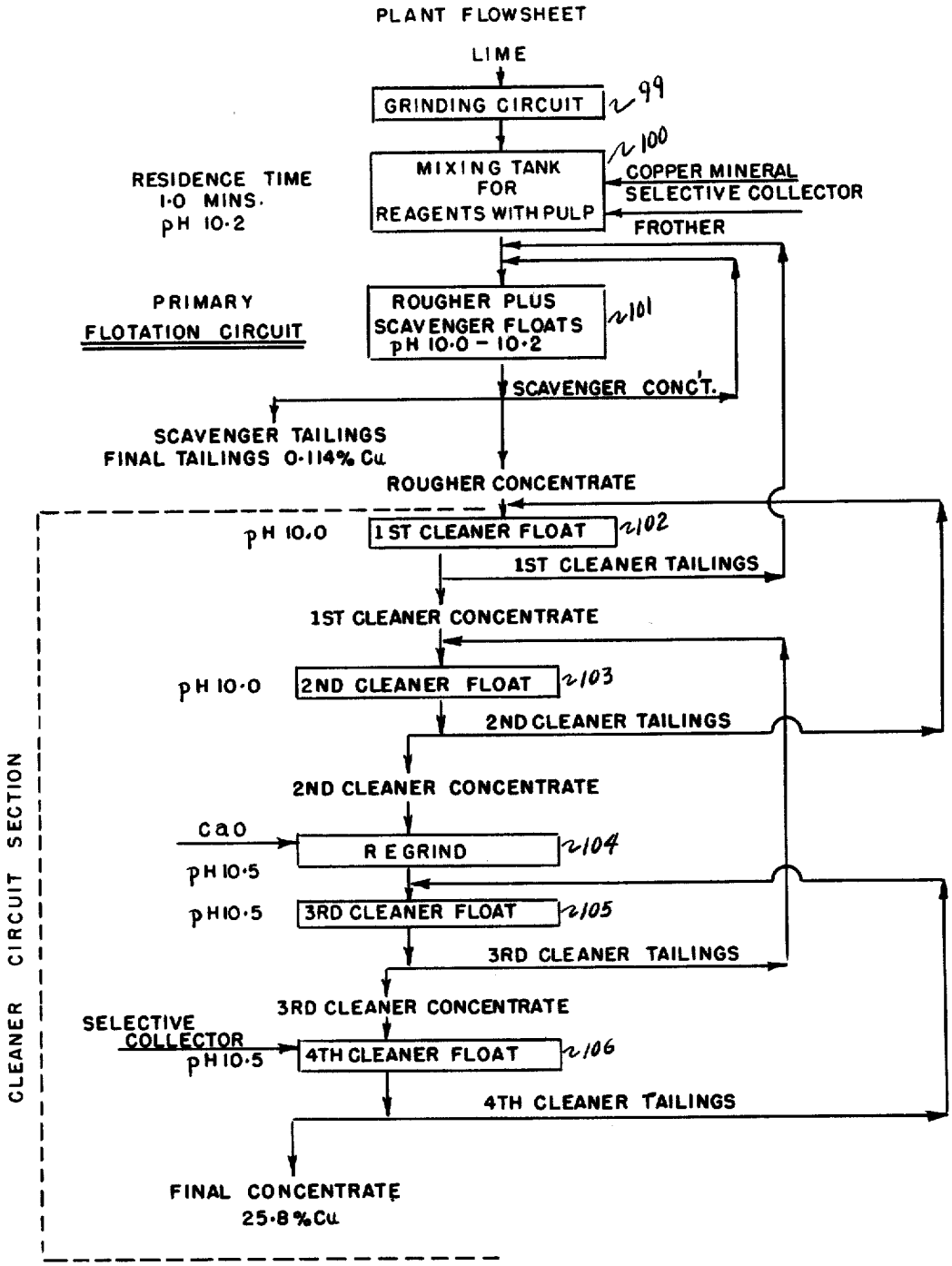

PLANT FLOWSHEET
LIME
GRINDING CIRCUIT
99
MIXING TANK FOR REAGENTS WITH PULP
100
COPPER MINERAL
SELECTIVE COLLECTOR
FROTHER
RESIDENCE TIME 1·0 MINS. pH 10·2
PRIMARY FLOTATION CIRCUIT
ROUGHER PLUS SCAVENGER FLOATS pH 10·0 – 10·2
101
SCAVENGER CONC'T.
SCAVENGER TAILINGS FINAL TAILINGS 0·114% Cu
ROUGHER CONCENTRATE
pH 10.0
1ST CLEANER FLOAT
102
1ST CLEANER TAILINGS
1ST CLEANER CONCENTRATE
pH 10·0
2ND CLEANER FLOAT
103
2ND CLEANER TAILINGS
2ND CLEANER CONCENTRATE
CaO
pH 10·5
REGRIND
104
pH10·5
3RD CLEANER FLOAT
105
3RD CLEANER TAILINGS
3RD CLEANER CONCENTRATE
SELECTIVE COLLECTOR
pH10·5
4TH CLEANER FLOAT
106
4TH CLEANER TAILINGS
FINAL CONCENTRATE 25·8%Cu
CLEANER CIRCUIT SECTION

FIG 7

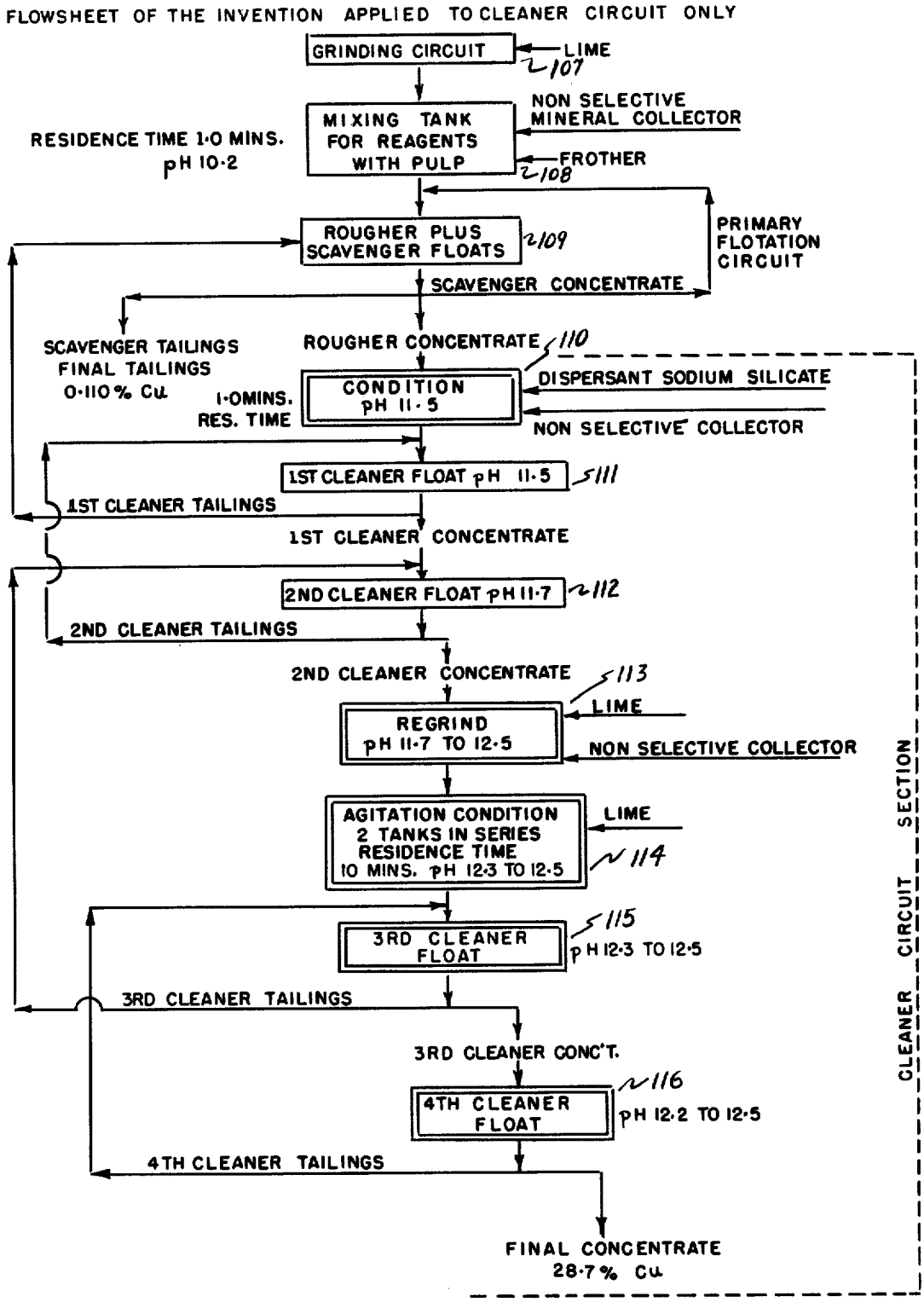
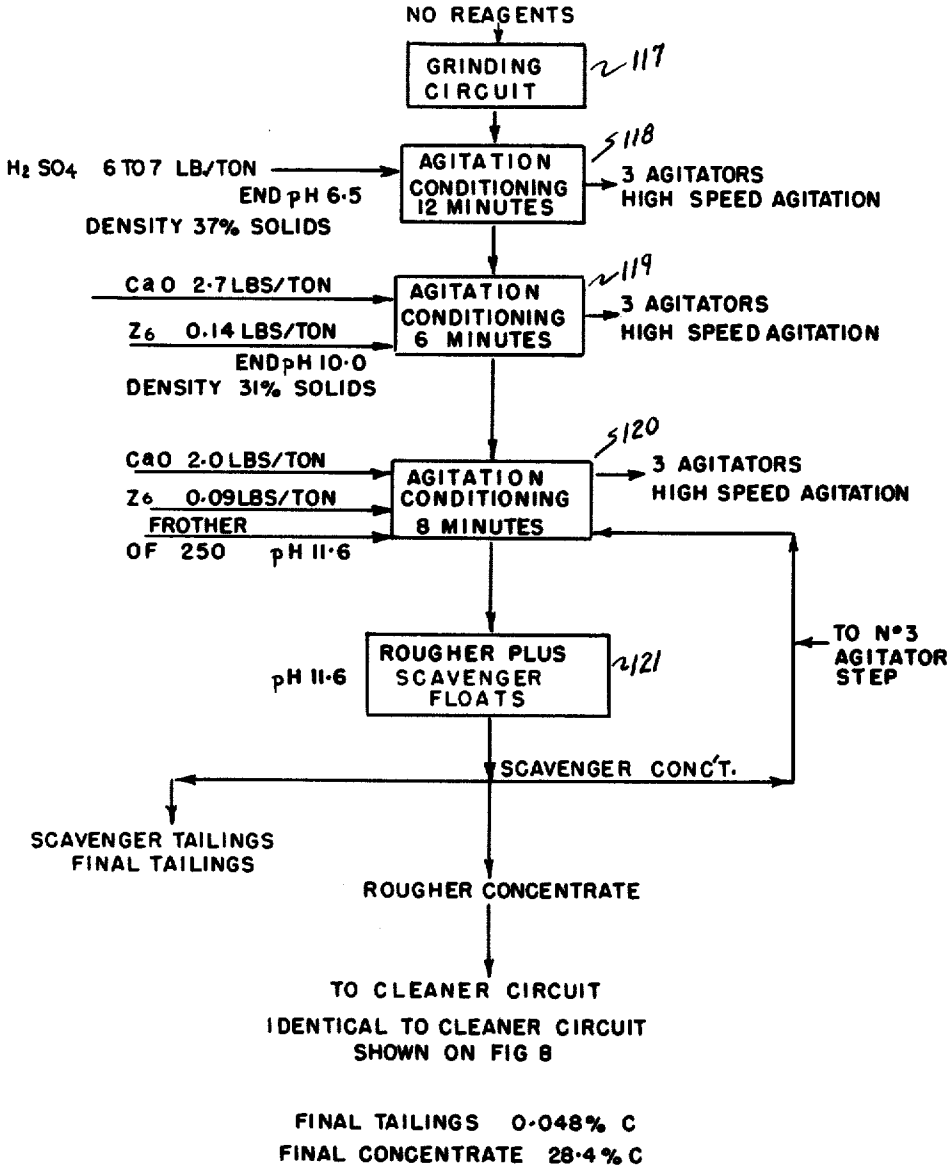

FLOWSHEET OF THE INVENTION APPLIED TO CLEANER CIRCUIT ONLY
GRINDING CIRCUIT
LIME
107
NON SELECTIVE MINERAL COLLECTOR
MIXING TANK FOR REAGENTS WITH PULP
FROTHER
108
RESIDENCE TIME 1·0 MINS. pH 10·2
ROUGHER PLUS SCAVENGER FLOATS
109
PRIMARY FLOTATION CIRCUIT
SCAVENGER CONCENTRATE
SCAVENGER TAILINGS FINAL TAILINGS 0·110% Cu
ROUGHER CONCENTRATE
110
CONDITION pH 11·5
1·0MINS. RES. TIME
DISPERSANT SODIUM SILICATE
NON SELECTIVE COLLECTOR
1ST CLEANER FLOAT pH 11·5
111
1ST CLEANER TAILINGS
1ST CLEANER CONCENTRATE
2ND CLEANER FLOAT pH 11·7
112
2ND CLEANER TAILINGS
2ND CLEANER CONCENTRATE
113
REGRIND pH 11·7 TO 12·5
LIME
NON SELECTIVE COLLECTOR
AGITATION CONDITION 2 TANKS IN SERIES RESIDENCE TIME 10 MINS. pH 12·3 TO 12·5
LIME
114
3RD CLEANER FLOAT
115
pH 12·3 TO 12·5
3RD CLEANER TAILINGS
3RD CLEANER CONC'T.
4TH CLEANER FLOAT
116
pH 12·2 TO 12·5
4TH CLEANER TAILINGS
FINAL CONCENTRATE 28·7% Cu
CLEANER CIRCUIT SECTION

FIG 8

FLOWSHEET OF THE INVENTION APPLIED TO CLEANER CIRCUIT ONLY WITH COPPER MINERALS HEAVILY ACTIVATED



FINAL TAILINGS 0·048% C
FINAL CONCENTRATE 28·4% C

FIG 9

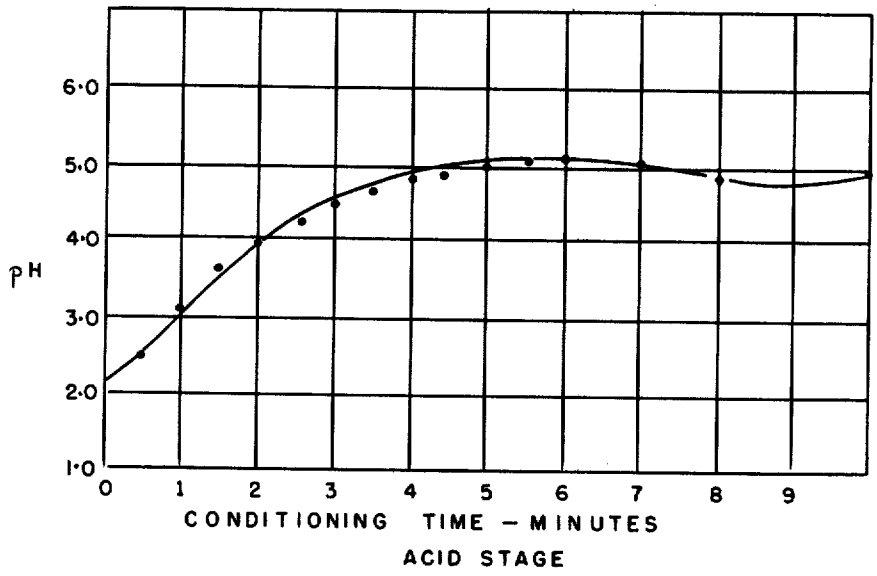

6·0
5·0
4·0
3·0
2·0
1·0
pH
0 1 2 3 4 5 6 7 8 9
CONDITIONING TIME – MINUTES
ACID STAGE

FIG 10

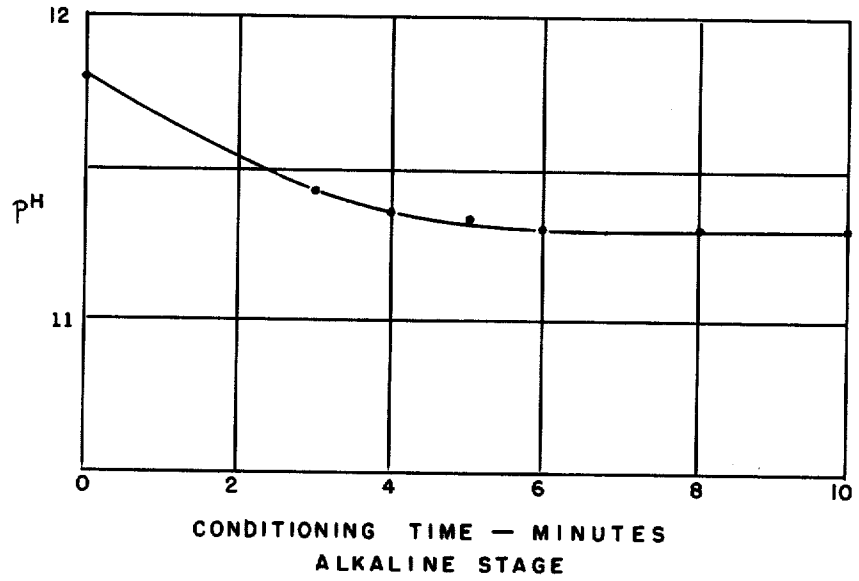

12
11
pH
0 2 4 6 8 10
CONDITIONING TIME – MINUTES
ALKALINE STAGE

FIG 11

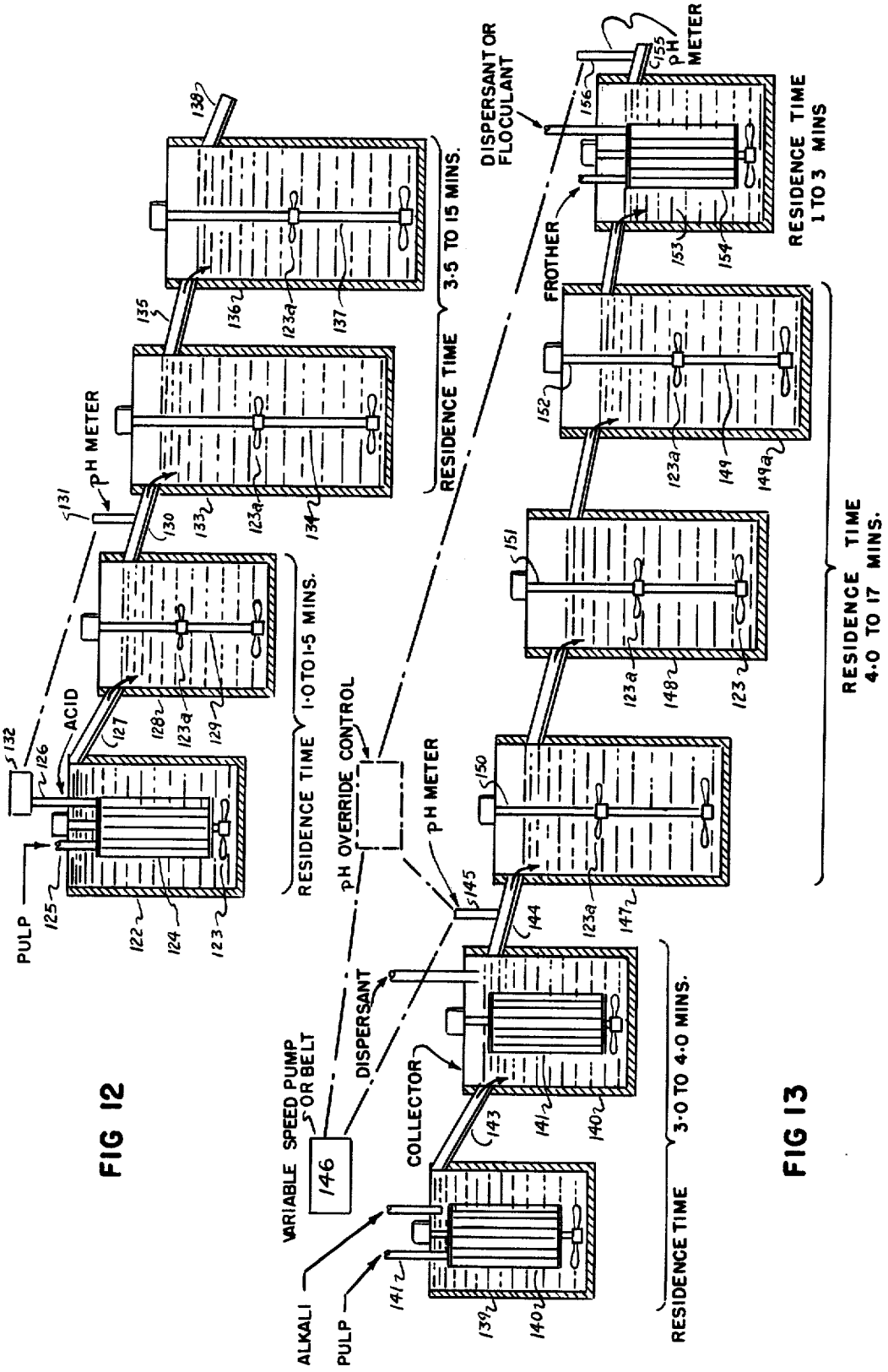

PULP
ACID
PH METER
PH OVERRIDE CONTROL
RESIDENCE TIME 1·0 TO 1·5 MINS.
RESIDENCE TIME 3·5 TO 15 MINS.
DISPERSANT OR FLOCULANT
FROTHER
PH METER
RESIDENCE TIME 1 TO 3 MINS
RESIDENCE TIME 4·0 TO 17 MINS.
ALKALI
PULP
VARIABLE SPEED PUMP OR BELT
146
DISPERSANT
COLLECTOR
RESIDENCE TIME 3·0 TO 4·0 MINS.

FIG 12

FIG 13

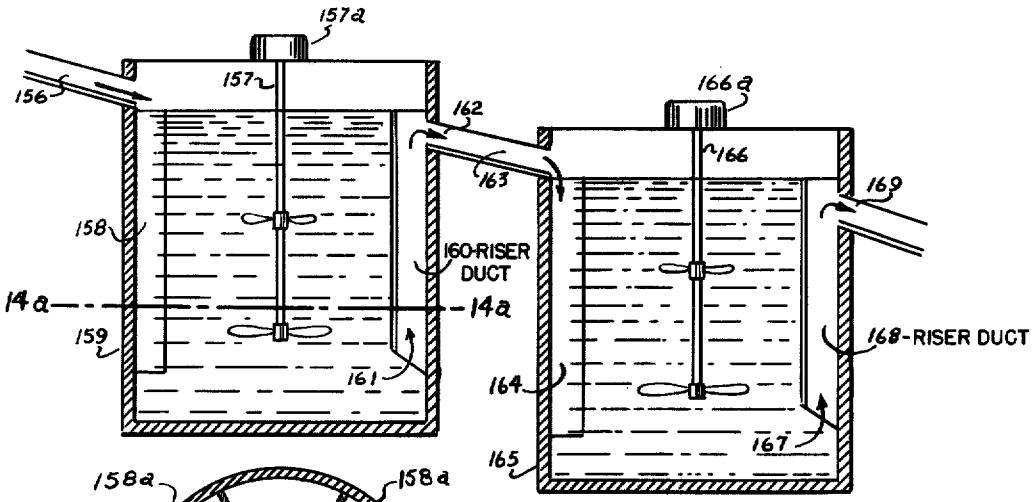


FIG. 14

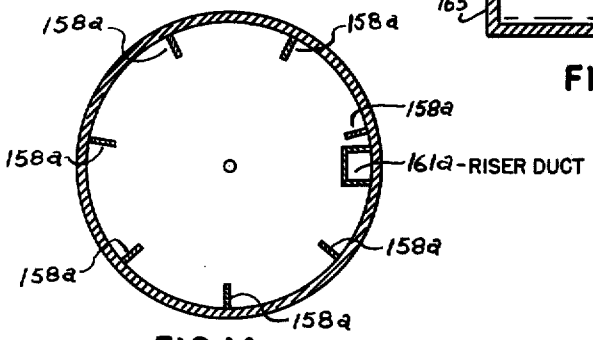


FIG. 14a

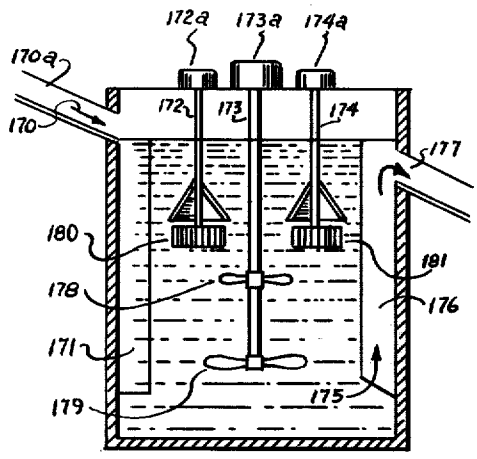


FIG. 15

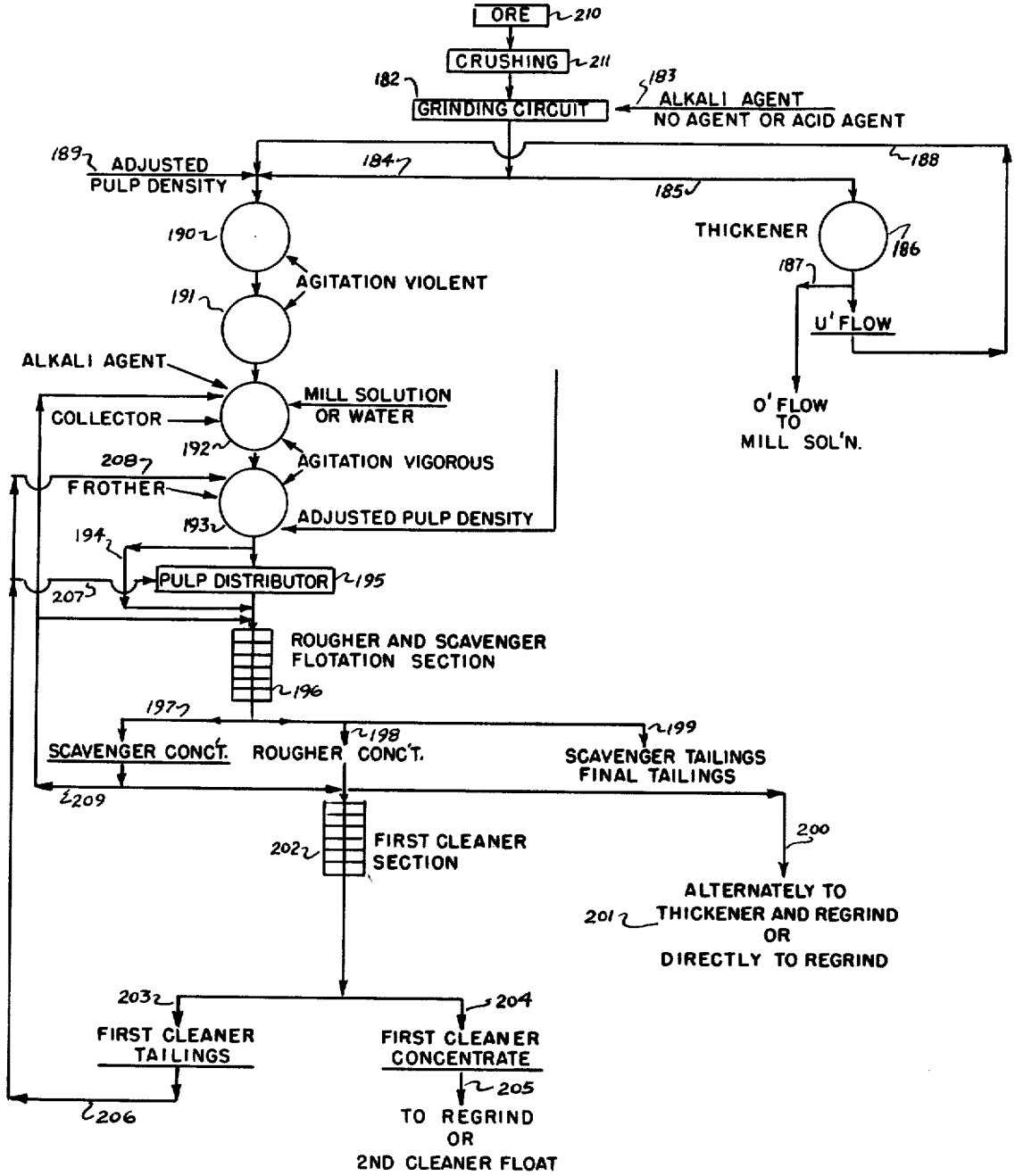

ORE
210
CRUSHING
211
182
183
ALKALI AGENT
NO AGENT OR ACID AGENT
GRINDING CIRCUIT
189
ADJUSTED
PULP DENSITY
184
188
185
THICKENER
186
187
U' FLOW
O' FLOW
TO
MILL SOL'N.
190
AGITATION VIOLENT
191
ALKALI AGENT
MILL SOLUTION
OR WATER
COLLECTOR
192
AGITATION VIGOROUS
208
FROTHER
193
ADJUSTED PULP DENSITY
194
PULP DISTRIBUTOR
195
207
ROUGHER AND SCAVENGER
FLOTATION SECTION
196
197
198
199
SCAVENGER CONC'T.
ROUGHER CONC'T.
SCAVENGER TAILINGS
FINAL TAILINGS
209
200
202
FIRST CLEANER
SECTION
ALTERNATELY TO
201
THICKENER AND REGRIND
OR
DIRECTLY TO REGRIND
203
204
FIRST CLEANER
TAILINGS
FIRST CLEANER
CONCENTRATE
205
TO REGRIND
OR
2ND CLEANER FLOAT
206

FIG. 16

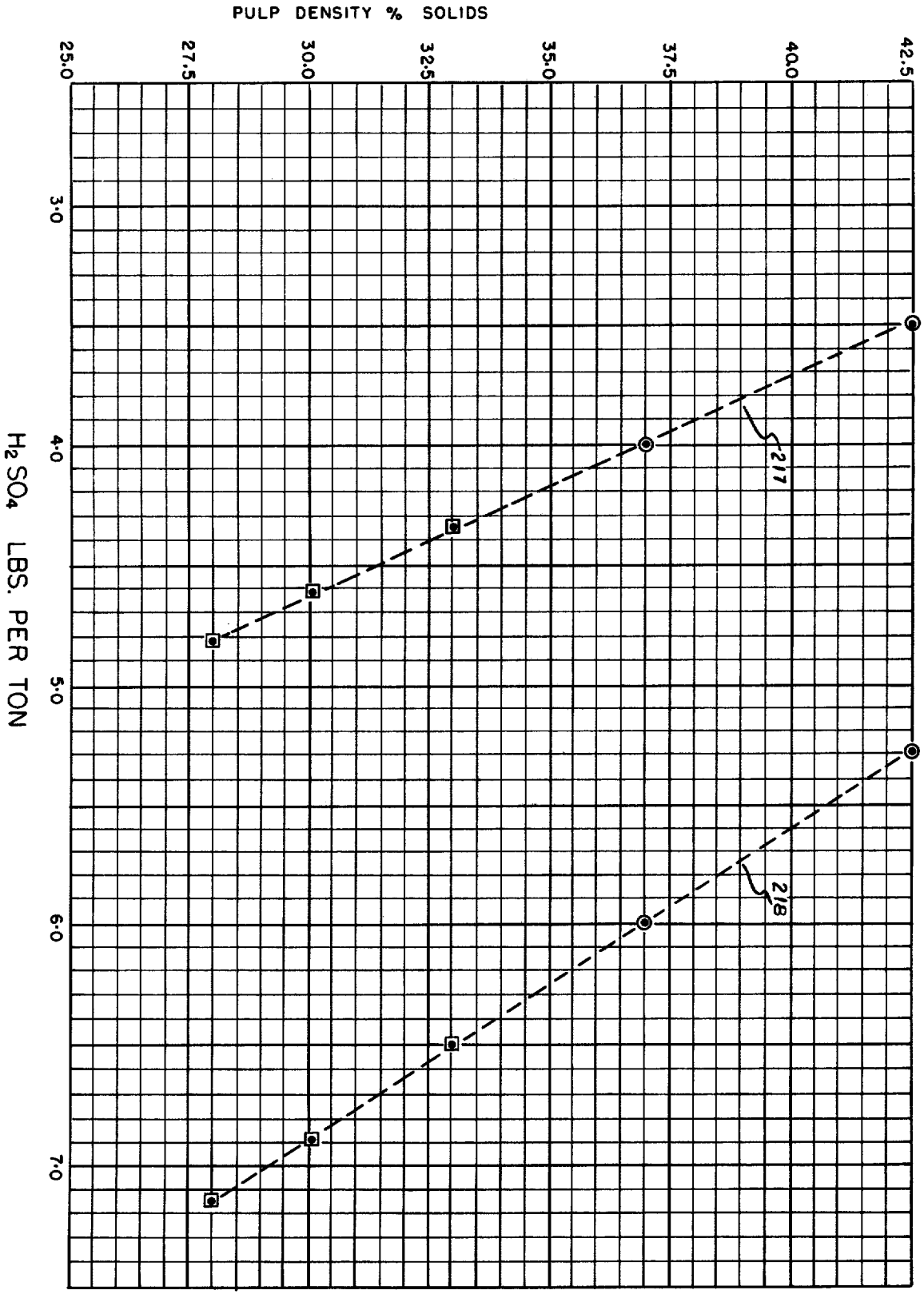

PULP DENSITY % SOLIDS
25.0
27.5
30.0
32.5
35.0
37.5
40.0
42.5
3.0
4.0
5.0
6.0
7.0
H2 SO4 LBS. PER TON
217
218

FIG 17

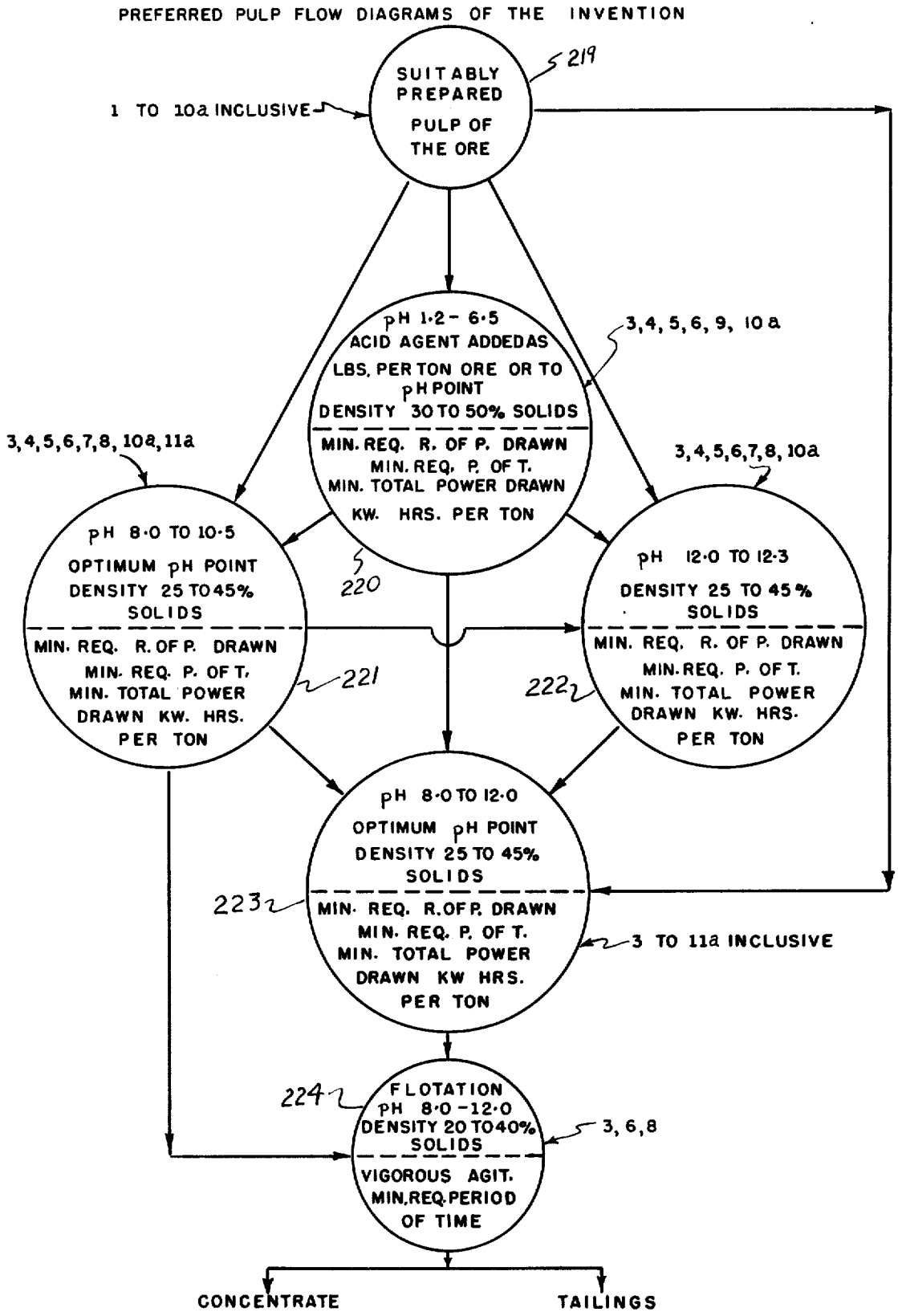

PREFERRED PULP FLOW DIAGRAMS OF THE INVENTION
SUITABLY PREPARED PULP OF THE ORE
219
1 TO 10a INCLUSIVE
pH 1·2 - 6·5
ACID AGENT ADDEDAS
LBS. PER TON ORE OR TO pH POINT
DENSITY 30 TO 50% SOLIDS
MIN. REQ. R. OF P. DRAWN
MIN. REQ. P. OF T.
MIN. TOTAL POWER DRAWN
KW. HRS. PER TON
220
3,4,5,6,9,10a
3,4,5,6,7,8,10a,11a
pH 8·0 TO 10·5
OPTIMUM pH POINT
DENSITY 25 TO 45% SOLIDS
MIN. REQ. R. OF P. DRAWN
MIN. REQ. P. OF T.
MIN. TOTAL POWER DRAWN KW. HRS. PER TON
221
3,4,5,6,7,8,10a
pH 12·0 TO 12·3
DENSITY 25 TO 45% SOLIDS
MIN. REQ. R. OF P. DRAWN
MIN. REQ. P. OF T.
MIN. TOTAL POWER DRAWN KW. HRS. PER TON
222
pH 8·0 TO 12·0
OPTIMUM pH POINT
DENSITY 25 TO 45% SOLIDS
MIN. REQ. R. OF P. DRAWN
MIN. REQ. P. OF T.
MIN. TOTAL POWER DRAWN KW HRS. PER TON
223
3 TO 11a INCLUSIVE
FLOTATION
pH 8·0 - 12·0
DENSITY 20 TO 40% SOLIDS
VIGOROUS AGIT.
MIN. REQ. PERIOD OF TIME
224
3, 6, 8
CONCENTRATE
TAILINGS

FIG 18

FLOTATION OF SULPHIDE MINERALS FROM SULPHIDE BEARING ORE

This invention is a continuation in part of application Ser. No. 267,205, now abandoned filed June 28, 1972, Application Ser. No. 270,961, now U.S. Pat. No. 3,735,931, filed July 19, 1972 and Ser. No. 304,540, now abandoned, filed Nov. 7, 1972.

BACKGROUND OF THE INVENTION

This invention relates to improvements in the flotation recovery of sulphide and a number of non-sulphide minerals from ores, particularly copper minerals, nickel minerals and molybdenum minerals, in which sulfhydryl anionic collectors are used as the collecting agent.

In conventional flotation practice as applied to the flotation of sulphide minerals from sulphide bearing ores, the largest tailings losses usually take place in the fines fractions, that is, the minus 325 mesh, or even finer, that is, the minus 20 micron size fraction, and in the coarsest fraction of the grind, which normally contains the highest percentage of middling particles.

My invention employs sulfhyoryl anionic collectors and, in probably its largest economic application, pertains to a comparatively large increase in recovery of sulphide minerals by being able to effectively recover the minerals values in these size fractions. The present invention is not generally dependent upon the particular collector employed, but is directed to the preparation of pulps for flotation recovery of at least the sulphide mineral values. Thus, where in the present application I use the expression "sulfhydryl anionic collectors" I intend the term to include any sulphur-connected anionic collector or combination of collectors which, by conventional testing procedures, has been selected for the particular flotation application.

The sulfhydryl anionic collectors are discussed and classified in "Flotation," Second Edition, A. M. Gaudin, McGraw Book Company Inc., New York, 1957, pages 181–184.

The term "Agitation Conditioning Circuit" may consist of one but preferably a number of individual agitation conditioning units wherein the units are arranged in series and the pulp flows from one agitator to the next, and whatever number may follow. When I use the term "Agitation Conditioning Stage" it may mean a single unit or a multiplicity of agitators. An agitation conditioning circuit may have anywheres from one to three or more agitation conditioning stages in which the agitators are arranged in series. The distinguishing feature between each agitation conditioning stage is where I add an acid agent or an alkaline agent to change the pH range within the agitation conditioning circuit. For instance, if prior to my final heavy activation agitation conditioning stage in the pH range of about 8.0 to 12.0 I prefer to use a preceeding acid conditioning stage where, following the grinding circuit, I add sulphuric acid to the pulp to lower the pH within the range of approximately 1.2 to 6.5, I would condition the pulp in normally a minimum of three agitators in series. In the No. 1 Agitator the pH of the pulp may be 1.2, while at the discharge of the pulp from the third agitator the pH may have risen to as high as 7.0. I would term this as the first agitation conditioning stage of the agitation conditioning circuit. The pulp will flow from Agitator No. 3 to No. 4 and this second agitation conditioning stage could be two agitators, that is, No. 4 and No. 5. I now add lime or calcium hydroxide to Agitator No. 4 and raise the pH to an optimum point within the range of 8.0 to 12.0. This would be the beginning of the Agitation Conditioning Stage 2. In Agitation Conditioner No. 4 in this example I also add a sulfhydryl anionic collector. At the beginning the pH in Tank 4 may be 10.5 and on the discharge from Tank 5 would have dropped to a pH of 10.0. The pulp from Tank 5 may now flow into three additional agitators in series, Tanks 6, 7, and 8. In Tank 6, which would be the beginning of the third agitation conditioning stage, I add further lime to raise the pH of the pulp to 11.6. I may also add additional collector to this tank. By the time the pulp flows through Tanks 6, 7, and 8 the pH of the pulp at the discharge of Tank 8 may have dropped to, say, 11.2. This would be the end of Agitation Conditioning Stage 3. The valuable sulphide minerals, and possibly pyrite in the pulp discharging from Tank 8 are in the heavily activated form of my invention. I add a frother to Tank 8 and the pulp would flow to the primary flotation circuit. It would be noted that in this illustrated case where there is continuous agitation conditioning, the distinguishing feature of the agitation conditioning stages is the addition of an acid agent or an alkaline agent to change substantially the pH range within the individual agitation conditioning stages.

I may use a combination of agitation conditioning tanks and flotation cells in an agitation conditioning circuit. For instance, following Tank 8 where the pulp flows or is pumped to flotation cells I may add frother or collector to one or more flotation units with the air turned off and the cells acting as the final period of agitation conditioning prior to flotation.

Where I use a number of flotation cells in series as agitation conditioning tanks, the same differentiation applies.

The term "Agitation Conditioning Step" refers to the specific agitator or flotation cell in the series wherein the first agitator the pulp enters in the agitation conditioning circuit is referred to as Number 1 step. Alternately it means a single agitator.

By "Acid Agent" is meant an agent selected from the group consisting of sulphuric acid, sulphurous acid, and sulphur dioxide, and is used to lower the pH in the alkaline pH range, or reduce the pH of the pulp to a desired point or range in the acid pH range of about 1.2 to 6.5.

By "Sulphidizing Agent" is meant an agent selected from the group consisting of sodium hydrogen sulphide, sodium sulphide or polysulphide, calcium sulphide, and hydrogen sulphide, from the use of which in my circuit in their application to copper ores a copper sulphide film is thought to form on certain non-sulphide copper minerals, resulting in such minerals to be collector coated with sulfhydryl anionic collectors and subsequently recovered by flotation.

Where I refer to "Sulphide Copper" or "Copper Sulphide" minerals I not only mean such minerals as Chalcocite and Covellite, but also the complex Copper Sulphide minerals such as Chalcopyrite, the Copper Iron Sulphide, and the other complex copper sulphides that commonly occur in copper deposits.

Where I refer to "Sulphide Nickel" or "Nickle Sulphide" minerals, the main mineral is "Pentlandite," the nickel iron sulphide, but may also include such nickel minerals as "millerite," the nickel sulphide.

Where I refer to "Sulphide Molybdenum" or "Molybdenum Sulphide" minerals, I am referring to Molyb-

denite.

Where I refer to "Sulphide Lead" or "Lead Sulphide" minerals, I am referring to Galena.

By "Alkaline Agent" is meant an agent selected from the group consisting of lime, calcium hydroxide, sodium carbonate and lime or calcium hydroxide in combination with sodium or potassium carbonate, sodium hydroxide and ammonium hydroxide, and is used for upward adjustment of the pH of the pulp.

By the expression "Heavy Activation" is meant a stage of activation in which at least the desired minerals are so heavily activated that surfaces of the sulphide minerals will take on and maintain a highly effective collector coating resulting in their floating in a much shorter time than in current conventional practice, and in addition, will not be depressed in the cleaner circuit wherein the pH may be appreciably higher than the pH used in the primary circuit, and appreciably higher than the accepted optimum pH for flotation of the desired recoverable minerals. "Heavy activation" does not preclude substantial activation and collection of minerals other than the valued sulphide minerals and the expression is used to distinguish from the present day conventional practice wherein the collection of the minerals is accomplished by stage addition prior to and during flotation of much smaller amounts of sulfhydryl anionic collector with a view, over the complete flotation cycle, to collecting more and more of the desired mineral in stages while ideally collecting none of the other minerals constituting the ore. The relatively high concentration of collector used at one or more points in my circuits is one of the features of the invention. For instance, where I heavily activate and effectively coat sulphide copper minerals in one or more agitation conditioning stages between the grinding circuit and the primary flotation circuit I may use as much as three times the amount of collector at this point alone as would be used in the total flotation circuit in applying current conventional practice to the same ore. In obtaining the optimum results of my invention the total amount of collector used as compared to conventional practice varies from about the same amount to as much as four hundred percent more dependent on the characteristics of the individual ore comparison. This phenomenom is not understood. Where I use the description of "sufficient concentration of collector" I mean that the amount of collector present at this stage in my circuit is in sufficient amount, in combination with other elements of my invention, to either produce or maintain the excellent floatability of the sulphide minerals which is a characteristic of my process.

When I use the term "pounds per ton" of various reagents, this is "pounds per ton of the total original feed," unless otherwise specified.

By "Dispersing Agent" is intended to be meant an agent which is nondeleterious to the flotation of sulphide minerals in the teachings of my invention, and which tends to create conditions of dispersion within the pulp at the prevailing pH to effect depression of the non-sulphide waste host rock minerals in subsequent flotation stage or stages. Where I use a dispersing agent the physical and chemical actions or reactions that take place under conditions of my invention and with relatively high concentration of dispersant compared to prior conventional use, is not understood, particularly the indicated effect of the dispersant as a strong activator for the collection of the sulphide minerals.

In describing the practice of my invention, the term "agitation conditioning" as distinguished from conventional practice is important. In my prior pending application Ser. No. 270,961, I have described an agitation conditioning circuit prior to rougher flotation. In plant practice the agitators consisted of cylindrical tanks using an impellor type mechanism with either a single impellor, or two impellors mounted one below the other on a single shaft.

In mixing reagents with the pulp to procure collector coating of sulphide minerals conventional practice uses the lowest possible agitation speeds with the main objects being to keep the solids in the pulp in suspension and distribute the reagents throughout the pulp. To quote Taggert, Section 12, page 20: "With fine pulps large tanks and slow agitation, as by slow sweeps, will serve, the principal consideration in this case being dispersion of the reagents."

With coarsely ground ores, due to the settling out characteristics of the coarse particles, small tanks are employed with more vigorous agitation to prevent the coarse particles from settling out.

By "Vigorous Agitation" I mean fast circulation of the pulp within the individual agitator using substantially higher power to the agitation mechanism or mechanisms than would be required to keep a finely ground product of the ore in suspension at a specific pulp density.

By "Violent Agitation" I mean not only fast circulation of the pulp within the individual agitator, but usually using in excess of 200 percent higher power to the agitation mechanism than would be required to keep the ground product of the ore in suspension at a specific pulp density, and of sufficient violence to cause splashing of the pulp within the agitator and entrainment of air within the pulp.

In my co-pending U.S. application Ser. No. 304,540, and earlier applications Ser. Nos. 267,205 and 270,961 (that is now allowed), I did not fully appreciate the important significance of the type of agitation conditioning produced being the resultant of the necessity for high energy input to be imparted to the pulp by the agitation mechanisms in order to obtain the optimum results from my invention. In all cases this high energy input produces a vigorous agitation which varies in relationship to the requisite amount of power required by the agitation mechanisms, to obtain the optimum metallurgical results of the invention. This requisite high energy input can be measured by the horsepower drawn by the motors driving the Agitation Conditioning mechanisms and converted to kilowatt-hours per ton of ore treated.

To achieve optimum results from my agitation conditioning circuits I prefer to use an agitator design described later in this patent application.

By "Agitation Mechanism" I mean the impellor or rotor or a number of impellors and rotors attached to a revolving shaft and by their design combined with other features for agitation such as stators, their speed of rotation, and the tank design bring about the agitation conditions of the pulp.

A single agitator may be equipped with one or more agitation mechanisms attached to their individual shafts, or a number of agitation mechanisms.

Each agitation mechanism may be driven by its own motor and drive arrangement, or alternately, a single motor may drive two or more mechanisms through such an arrangement as V-belts and pulleys.

In the laboratory work wherein the metallurgy was optimized, I used a duplicate mechanism of conventional flotation equipment, such as the Denver laboratory flotation cell in which the agitating mechanism consisted of a rotor and stator, and similarly, the Fagergren laboratory flotation cell which again consisted of a rotor and stator. In this research work I found the speed of the rotor was quite important for optimum metallurgy. In all cases in the laboratory research programme the speed of the rotors for optimum results had to be raised sufficiently high to produce vigorous agitation of the pulp.

In the earlier stages of the invention this factor of high energy input imparted to the solids in the pulp with the resultant comparatively high speed agitation was not fully appreciated until it had been checked out at a later date on a variety of sulphide-bearing ores.

It was noted that on some types of sulphide bearing ores in which the slimes content, after comminution, was comparatively low, recovery of the minus 200 mesh sized sulphide particles was little improved, or in fact, poorer in using a dispersing agent in an agitation conditioning circuit. This factor was noted in my earlier and pending patent applications. In working with these ores at a later date in establishing certain conditions in the pulp which are described in a number of examples, a surprising increase in recovery was obtained when the vigorousness of agitation was increased appreciably over that required to keep a finely ground pulp in suspension, and in the absence of a dispersant. Another surprising result was an increase in recovery of the coarse middling particles.

Where I add an alkali agent or acid agent to the conditioning circuit to raise or lower the pH, I prefer to use vigorous to violent agitation particularly for fast absorption of an acid agent. Pulp density is also a factor in obtaining optimum results and my agitation conditioning in the various stages is about 20 to 50 percent solids, with my preferred range being 30 to 50 solids. I use the higher percent solids at the head of the agitation conditioning circuit, with the lowest at the end of the circuit.

To establish the optimum agitation conditioning in the laboratory following optimizing of the other features of my invention wherein I use one rotor speed of the agitation mechanism, I then increase the speed of the rotor in steps until there is no further drop in the flotation tailings loss. I also check a lower and, if necessary, a higher rotor speed. By this means I arrive at the optimum speed range of the rotor for each agitation conditioning stage. I follow the same procedure in plant practice, equipping the agitators with various types of means described later in this application to rapidly vary the speed of the agitation mechanisms. By interpolation correlated with experience from plant practice the power necessary to drive the agitation mechanisms can be estimated as to the minimum power required, and optimum power required range to obtain maximum recovery of the desired minerals.

In plant practice, all other factors being equal, the conditions of agitation within the parameters of the agitation mechanism design can be measured by the horsepower drawn by the motors driving the agitation mechanisms. The horsepower drawn, in turn will be a relative factor to the energy imparted to the pulp by the agitation mechanisms. Although the phenomenon of this feature of the invention is not fully understood, it is thought that this imparting of energy to the pulp and particularly the ground ore particles is a most important part of the invention.

A surprising feature of the invention is that each agitation conditioning stage requires a specific minimum period of time of agitation at a minimum speed of rotation of the agitation mechanism with the motor driving the agitation mechanism drawing a minimum horsepower. If the speed of rotation of the agitation mechanism is substantially increased over this amount, one would conclude that the time factor would be substantially reduced. This is not generally the case, which would indicate that not only is the sufficient energy imparted to the pulp to obtain maximum recovery of the desired minerals important, but also that this approximate level of energy must be imparted to the pulp over a sufficiently long period of time.

In using shaped blade-type impellors for the agitation conditioning, this type of impellor is lower speed and the power input, in addition to the speed, depends upon the design and size, together with the tank design. In the first large scale plant operation using my process, this type of impellor design was used in the agitation conditioning circuit, as shown at FIG. 13. The power drawn by the various agitator mechanisms exceeded 50 percent to as much as 200 percent over the manufacturer's recommendation for agitation to keep the solids in suspension with a normal safety factor. The first two agitators in which violent agitation conditions were produced incorporated power input to the agitation mechanisms of approximately 400 percent over the agitation requirements to keep the solids in suspension. In plant practice, in addition to the type of agitation conditions which I have described, it is obvious that flotation cells can be used for agitation conditioning circuits with the speed of the rotors adjusted to an optimum. In plant practice I prefer to use a minimum of two cells or two tanks in series in any one agitation conditioning stage. I consider three tanks or three cells optimum in allowing for short-circuiting of the pulp and wherein the minimum required period of residence time of the pulp for agitation conditioning has a normal safety factor. I have found that any reasonable excess time over the required minimum is not deleterious to the metallurgy in any of my alkaline agitation conditioning stages. However, insufficient time will not produce the optimum desired results of my invention. For this reason, in the practical application of my invention I prefer to use a large safety factor which each alkaline agitation conditioning stage ensuring that the pulp has a sufficiently long period of residence time in each agitation conditioning stage to produce the optimum desired results.

Conventional practice recognizes the importance of the time required for collector coating of the desired recoverable minerals. One of the major differences of my invention in this respect is that the optimum time in conventional practice to obtain the maximum recovery at an acceptable grade of concentrate can be critical. The reference to these effects is "Handbook of Mineral Dressing" by Arthur F. Taggert, 1947 published by John Wiley & Sons Inc., New York, Section 12, page 19, to quote: "On the contrary, conditioning time may be too long, resulting in deterioration of recovery and/or grade of concentrate; for this reason, once the optimum time has been determined arrangements should be made to hold time as nearly as constant as possible." In many plants it is impossible to attain anywheres even near a constant time due to the large variation in ore

tonnage handled by the grinding circuit. With ore segregation in the bins and varying grindability of the ore, the tonnage from the grinding circuit may vary over a 24-hour period by as much as 40 percent.

In conventional practice for the flotation of sulphides such as copper sulphide minerals, this optimum time period for the collector coating of the minerals with sulfhydryl anionic collectors is of very short duration, normally involving only mixing of the collector with the pulp either with addition to the grinding circuit, to a mixing tank following the grinding cirucit, or as in one of the most modern plants simply to the pulp from the grinding circuit before distributing the pulp to the primary flotation circuit banks of cells.

In the application of my invention in respect to the pulp from the grinding circuit I use an agitation conditioning circuit with preferably an excess of conditioning time over the minimum for optimum results. For instance, in Example XIII the contact time of the collector with the pulp prior to flotation was 36 minutes and gave the lowest tailings loss of the various all-alkaline circuits used. Such a period of time, in this application to the circuit, is unheard of in conventional practice.

The minimum period of time for an agitation conditioning stage is important and may vary from about two to thirty minutes. In applying my invention to a pulp of the ore positioned in the flowsheet between the grinding circuit and primary flotation circuit, I use a minimum agitation conditioning time of about four minutes wherein the pulp density is in the range of about 25 to 50 percent solids.

Where I use an individual agitation conditioning stage on a flotation product to increase the activation of weakly activated minerals such as may be contained in a rougher concentrate, scavenger concentrate, or cleaner tailing the agitation conditioning time necessary may be less than four minutes, and at densities varying from 15 to 45 percent solids. In plant practice the residence time of the pulp can be calculated from the tonnage treated and pulp density. The residence time in the various agitation conditioning stages described in this application ignore the short-circuiting factor and in plant practice should be taken into consideration.

In the preferred embodiment of my invention in treating ores in which the major part of the values are in the form of sulphide copper minerals or sulphide molybdenum minerals, for optimum metallurgy the minimum agitation conditioning time is about 4 minutes using a minimum of two agitation conditioning steps. Where I use two alkaline agitation conditioning stages I prefer to use a minimum of four agitation conditioning steps.

Where sulphide nickel values are present in the pulp, for optimum metallurgy the minimum agitation conditioning time is about 8 minutes using a minimum of three agitation conditioning steps. In this application of my invention copper sulphate may be added to the grinding circuit during the preparation of the pulp.

Where weakly activated minerals show up in the various flotation circuits, such as a scavenger concentrate from the last float on the primary flotation circuit or the end float of the first cleaner circuit, a scavenger tailing or tailing from the first or other cleaners, part or all of such products may be beneficially returned to one or more agitation conditioning steps in the agitation conditioning circuit positioned in the plant flowsheet between the grinding circuit and the primary flotation circuit. The optimum place of return of such products can usually be determined by closed circuit laboratory work.

The second and third difference from conventional practice are the total amount of power input to the agitation mechanisms together with the rate or variable rate of power input to the individual mechanisms over the residence time period of the pulp in an agitation conditioning circuit.

The power drawn by the agitation mechanisms, that is, the impellors or rotors including losses in efficiency of the motors and driving mechanism such as a speed reducer, converted to kilowatt hours per ton of ore treated per four minutes of residence time is a minimum of 0.12 kilowatt hours. For instance, if a plant was treating 400 tons of ore per hour through my conditioning circuit with eight minutes residence time, the total power drawn by the motors driving the agitation mechanisms of all the agitators would be a minimum of

$$400 \times \frac{0.12}{0.746} \times \frac{8}{4} = 128.7 \text{ horsepower}$$

Where part or all of a flotation product such as a scavenger concentration from the primary circuit or from the first cleaner circuit, or part or all of any tailings from the various cleaner circuits are fed into an agitation conditioning circuit positioned in the flowsheet between the grinding circuit and the primary flotation circuit, the residence time of the new feed from the grinding circuit must still be maintained at the minimum time period of about 4.0 minutes.

The rate of power input to any one agitation mechanism is also important. It may vary within the circuit as shown in some of the illustrations, and the method of obtaining the optimum rate of power input is described in this application. If the rate of power input throughout the whole of the agitation conditioning circuit is too low a high tailings loss of the valued sulphide minerals ensues. If the rate of power input throughout the whole of the agitation conditioning circuit is appreciably too high, maximum recovery will not be obtained.

This latter condition would be impractical to practice as not only would there be a waste of power but the maintenance of the agitation mechanisms could become a major source of cost and lost operating time.

One of the most surprising features of this invention is, that to the inventor's knowledge, for the first time in the flotation of sulphide minerals the power used to drive the agitation mechanisms in an agitation conditioning circuit to obtain heavy activation of the desired sulphide minerals can be treated in the same manner as a reagent. For instance, where, in conventional practice a sulphide collector such as potassium amyl xanthane is stage added prior to and during rougher flotation, minimum amounts are required to activate the desired minerals. If the total amount is less than the requisite minimum, recovery of the desired minerals will suffer. If an excess is used, undesired minerals may be activated. Further, if a maximum is exceeded, particulrly just prior to rough floation, the froth is flattened and recovery of the desired mineral seriously affected within the normal rougher circuit floation time.

In obtaining heavy activation of the desired sulphide minerals in my agitation conditioning circuit, the power input impartted to the pulp through the agitation mechanisms can be interpreted as stage added over a mini-

mum period of time using a multiplicity of agitation mechanisms and using a minimum total amount of kilowatt hours per ton of solids treatd. If the amount of power used is less than the requisite minimum, recovery of the desired minerals will suffer. If a maximum is exceeded, particularly for a sufficient period of time prior to rougher floatation, recovery of the desired minerals is detrimentally effected.

One of the reasons why in plant practice I prefer to use a multiplicity of agitation conditioning units is that the power used and thus the energy input imparted to the solids particles in the pulp in each agitation conditioning step can be readily controlled with the agitation characteristics of the pulp a secondary factor, and optimized to obtain the heavy activation of the valued sulphide minerals. Where I use large diameter agitators I prefer to use a multiplicity of agitation mechanisms so that the maximum energy input imparted to the solid particles at any one particular period may not be exceeded, such as say, in using a single high speed mechanism. With my preferred agitator design, as I describe later, I am not concerned under certain conditions of operation with the possible settling out of the coarsest particles of the ground ore, which under some conditions of the operation require violent agitation, and thus could result in lower than maximum sulphide mineral recovery, due to the necessity of using too high power input to the pulp for too long a period of time.

Where I refer to an "optimum pH" point, I mean the practical pH point at which the pulp can be maintained. For instance, in a floatation circuit if the optimum pH point is 11.0, in practical plant application, due to changes in the core or plant operating conditions, it will vary between 11.0 plus or minus approximately 0.2, , with erratics due to changes in plant operating conditions, or poor operating plant control.

When I refer to the "Optimum pH Point" in an alkaline agitation conditioning stage, my control of the addition of the alkaline agent is usually based on the pH at the end of the stage. The optimum pH point within an agitation conditioning stage is actually a range of pHs that may vary as much as 0.6 to 0.9 from the initial point of the alkaline agent addition to the discharge point from the last agitator in the specific agitation conditoning stage. This is shown at FIG. XI.

In describing my invention the expression "appropriately prepared pulp of an ore" or a "suitably prepared pulp" of an ore when used herein is intended to mean that the pulp has been made up from ore wherein in its preparation at least one wet grinding circuit has been employed for comminution to the extent which by conventional standards would be considered sufficient for acceptable liberation of the desired mineral constituents and has, during such comminution or thereafter, been subjected to such treatment steps (such as adjustment of pulp density by dilution, thickening, thickening and dilution and adjustment of pH) as the operator may deem appropriate in the case of the particular ore being treated, to present the pulp for the stages of treatment comprising the process of the present invention. These treatment steps may be conventional and for conventional purposes, or they may, in conjunction with the basic process of the present invention, form one or more stages of a preferred embodiment of the invention.

It is the principal object of the present invention to provide a process for the flotation of economically valuable sulphide minerals from a broad spectrum of sulphide mineral bearing ores using sulfhydryl anionic collectors to give marked improvement in the valuable sulphide minerals recovery.

It is a further object of the invention to float in the same circuit such widely diverse copper sulphides as chalcopyrite, bornite and chalcocite.

It is a further object of the present invention to carry out such flotation to produce high grade concentrates and improved sulphide mineral value recoveries regardless of the presence in the ores of such deleterious materials as talc and clay.

It is a further object of the invention to provide integrated plant circuits whereby these procedures may be effectively and economically carried out not only on refractory types of ores, but also on what is considered normal metallurgical type ores, and particularly copper, molybdenum, and nickel sulphide bearing ores. As I consider this invention as basic to the flotation of sulphide minerals using sulfhydryl anionic collectors as the collector, the invention should not be construed as applicable to only copper, molybdenum, and nickel sulphide minerals. For instance promising results were obtained on lead sulphide bearing ores where the main lead sulphide mineral was galena.

SUMMARY OF THE INVENTION

I have invented a flotation process whereby slime sulphide minerals and sulphide minerals occurring as middlings in the coarse fraction of a prepared pulp of an ore are effectively recovered by froth flotation showing substantial improvements in valuable sulphide minerals recovery over conventional practices.

Broadly stated, the invention involves conditioning a suitably prepared pulp of the ore in at least one agitation condition circuit wherein the pulp of the ore is agitation conditioned in one or more agitation conditioning stages for minimum periods of time in each stage and the power used in each stage through the agitation mechanisms is in sufficient amount, and in combination with the other features of the invention achieves heavy activation of the sulphide minerals. The total power consumption of the agitation mechanisms is a minimum of 0.12 kilowatt hours per ton of solids fed to the circuit, and in at least the last step prior to flotation a sufficient amount of sulfhydryl anionic collector is present in the pulp, and the pulp is at an optimum pH point in the pH range of about 8.0 to 12.0. Following the last agitation conditioning step the pulp is subjected to flotation in the presence of a suitable frother, and if necessary, pulp density adjustment, to produce a flotation concentrate enriched in the valued sulphide minerals and a tailings impoverished in the said sulphide minerals.

The final agitation conditioning stage prior to flotation may be preceeded by one or more agitation conditioning stages wherein the pHs may be within the pH range of about 8.0 to 12.0, above 12.0 or below 8.0 Illustrations of such circuits are shown in the examples.

It is characteristic of the prepared pulps when treated according to my invention that the sulphide minerals are much more highly activated and effectively collector coated than in the case using conventional processes. This manifests itself in the fact that effective floatation can be carried out on a broad range of sulphide minerals and that these can be floated with high recovery in a single rougher float. Furthermore, the rougher concentrate produced by a primary flotation circuit can be subjected to a considerable variety of

conditions during cleaning, for instance, for depression of pyrite, without any substantial loss of mineral values.

Where a flotation product is produced containing some of the activated desired minerals which are still not sufficiently heavily activated to withstand the conditions of a cleaner circuit, such products as a scavenger concentrate from the rougher circuit or part or all of the various cleaner tailings from the cleaner circuits may be fed to the agitation conditioning circuit positioned in the flowsheet between the grinding circuit and the primary flotation circuit, or alternately, treated in one or more separate agitation conditioning circuits wherein the pH of the pulp is at an optimum pH point within the pH range of about 8.0 to 12.0.

The major application of the invention is in the use of an agitation conditioning circuit subsequent to the grinding circuit and prior to primary floatation wherein all of the desired recoverable sulphide minerals at this stage of flotation are floated in a single rougher or rougher plus scavenger concentrate. For instance, in an ore containing copper and molybdenum sulphide values, the copper and molybdenum sulphides would both be floated together. With an ore containing nickel and copper values the nickel and copper sulphides could be floated together, or alternately, the copper values in the first flotation stage and the nickel values conditioned in a second agitation conditioning circuit and floated in a second flotation circuit.

A secondary application of the invention is in treating a flotation concentrate, particularly a rougher, scavenger, or first cleaner concentrate prior to or after regrinding to increase the activation of the more weakly activated mineral particles. A further secondary application is to cleaner tailings where the original desired recoverable mineral drops from a cleaner float, and in a conventional circuit is finally lost in the plant final tailings.

FIG. **1**-shows one of the conventional flow sheets for copper ore flotation.

FIG. **2**-shows a preferred embodiment of the invention as applied to copper bearing ores having a high percentage of slimes.

FIG. **3**-shows another conventional flow sheet for pyrite depression, following a primary flotation circuit.

FIG. **4**-shows a preferred process following the primary flotation circuit with ores containing pyrite.

FIG. **5**-shows another conventional flow sheet using a sands-slime circuit.

FIG. **6**-shows a preferred embodiment as applied to a sands-slime circuit.

FIG. **7**-shows another conventional flow sheet for a copper ore.

FIG. **8**-shows a flow-sheet of a cleaner circuit in accordance with the invention.

FIG. **9**-shows one of the preferred embodiments of heavily activating copper minerals in the ore.

FIG. **10**-shows a typical acid conditioning stage prior to an alkaline stage according to the invention.

FIG. **11**-shows a typical alkaline agitation conditioning stage according to the invention.

FIG. **12**-shows another acid stage preceding an alkaline agitation conditioning circuit.

FIG. **13**-shows an embodiment of the invention using an all alkaline agitation conditioning circuit.

FIG. **14**-shows a vertical section of an agitator used in the circuit.

FIG. **14***a*--shows a sectional plan view of the agitator,

FIG. **17**-is a graph showing the affect of pulp density agitator.

FIG. **16**-shows another flow sheet of a preferred embodiment.

FIG. **17**--**is** a graph showing the affect of pulp density on the amount of acid used in an acid conditioning stage.

FIG. **18**-shows a pulp flow diagram with a pulp of the ore coming directly from the grinding circuit or thickening stage.

DESCRIPTION OF THE DRAWINGS

FIG. 1 - Shows one of the common conventional flow sheets for the flotation of copper ores. The alkaline agent, normally lime, is added to the grinding circuit, at **10,** to control the pH of the pulp for subsequent flotation. Following the grinding circuit the pulp is pumped, or flows by gravity, to a mixing tank, at **11,** consisting of a cylindrical tank with a shaped blade-type impellor attached to a shaft. The speed at which the impellor is driven is designed to keep the pulp in suspension and its main object is to distribute the reagents throughout the pulp. The residence time of the pulp in the tank is usually less than a minute. Collector and frother are normally added to this mixing tank. Following this tank, the pulp normally flows to a pulp distributor, at **12,** where it is fed to a number of rougher flotation banks of cells, at **13.** In the most recent conventional type of plant the mixing tank is eliminated, the alkali agent added to the grinding circuit and the collector and frother added to either the pulp distributor, or alternatively, to the cyclone classifier overflow. Reference to this recent type of technological thinking for treating the most common type of copper ores, the porphryes, is reported in "World Mining, "Volume 25, No. 8, July 192, pages 44 and 45, (World Mining, 500 Howard Street, San Francisco, Cal., 94105, U.S.A.).

During the rougher and scavenger flotation, additional collector is usually added. The total amount, including possible additional amount of collector to the cleaning stages, is usually less than 0.10 lbs. per ton.

In the rougher circuit there may be only a single concentrate produced, that is, a rougher concentrate, and in this case the rougher tailings is a final tailings to waste. In other flow sheets a high grade rougher concentrate is removed from the first few cells at the head of flotation and taken directly to the first cleaner float by-passing the regrind circuit, at **20.** This rougher flotation is followed by a scavenger float in the same or additional banks of cells, and in this case the tailings at the end of the scavenger float are termed the scavenger tailings and are normally final tailings to waste. The concentrate is termed the scavenger concentrate and is either returned to the head of the rougher float, at **19,** or alternatively, at **18,** may go either directly to the regrind circuit or to thickening prior to the regrind circuit. In using either circuit, or variations thereof, I refer to this circuit as the primary flotation circuit. Following either the use of the thickener, at **22,** on the rougher concentrate or the scavenger concentrate, or variations thereof, or alternatively, taking the primary circuit concentrates directly to a regrind circuit, at **28,** wherein normally the only agent added is lime, the product from the regrind circuit is fed to the No. 1 Cleaner Circuit, at **24.** Additional collector and frother may or may not be added during this cleaner float. Normally, the No. 1 Cleaner Tailings are final tailings, as the operators have found that the copper that is dropped in this No. 1

Cleaner Tailing is irrecoverable in using their conventional processes. An alternative circuit is to send the No. 1 Cleaner Tailing to a leaching plant to recover a small amount of acid soluble copper that, if it is floated in the primary circuit, a large portion will drop in the first cleaner tailing, due to its weak activation. The No. 1 Cleaner Concentrate is fed to a No. 2 Cleaner Circuit, at **25,** and this cleaner concentrate may be the final concentrate. In other cases additional cleaner steps ma be used. The No. 2 Cleaner Tailings, at **26,** are returned to the head of the No. 1 Cleaner Circuit.

FIG. **2** - Shows a preferred embodiment of the invention as applied to copper bearing ores containing a relatively high percentage of deleterious slime material with either the ore containing a negligible quantity of pyrite, or alternatively, a percentage of pyrite that could result in the production of a final low grade copper concentration due to the floating of the pyrite with the copper minerals present in the ore. In using the teachings of my invention, it is not necessary to add the alkaline agent, which is normally lime, to the grinding circuit, at **27.** Following the grinding circuit wherein my pulp density may be in the range of 25 to 50 percent solids and the preferred range at 30 and 50 percent solids, the prepared pulp of the ore is fed to an Agitation Conditioning Circuit, at **28.** I prefer to use a minimum of two agitators wherein the pulp is vigorously agitation conditioned. To obtain the best results from the agitation conditioning I prefer to have the sides of the agitators equipped with baffles (not shown). I prefer to use a minimum of two agitators in series, and dependent upon the residence time required, reagent addition, pH control, pulp dispersion, and tonnage of ore treated a number of additional agitation conditioners may be optionally used. The individual agitation conditioners may be of conventional design such as the Devereaux Agitator, using appreciably higher speeds than would be normally specified. My preferred design of agitator is shown at FIG. **14.**

As one simple embodiment of the process in agitation conditioning between the grinding circuit and the primary flotation circuit, I show three agitator conditioners, at **29, 30,** and **31,** in series wherein the pulp enters the first conditioner and flows from the first to the second, and the second to the third prior to the pulp distribution to the primary flotation circuit. In this alternative the alkaline agent is added to the first agitator conditioner, at **29,** and in practice would be controlled by the use of a pH meter recording the pH on the discharge of the third agitator conditioner, at **31.** I prefer to add the collector in this circuit to the first agitator conditioner, or only a part to the first agitator conditioner and a part to the second agitator conditioner, at **30.** A dispersing agent is shown as being added to the second agitator conditioner. In the use of the dispersing agent I prefer to have the collector agitation conditioned with the pulp either prior to or concurrent with the addition of the dispersing agent. This can be a major factor in maximizing the recovery of the sulphide values. Following the addition of dispersing agent I prefer to have at least one additional step of conditioning to ensure heavy activation of the sulphide values, and effective depression of the non-sulphide waste host rock minerals.

In place of individual tanks as agitator conditioners I may use part or one or more banks of flotation cells. Where flotation cells are used I prefer cells that are normally used in a primary flotation circuit in which the energy input is normally much higher than that of slower speed agitation which is used in cleaner circuits. In using the teachings of my process one or more banks of cells may be used with each bank made up of a number of flotation cell units. The air that would normally be fed to the cells for flotation is cut off and the cells used as agitation conditioners. Again, the reagents may be added at various points in the bank or banks of flotation cells used, and as shown in the flowsheet, at **32,** as an alternative to the use of individual circular type agitator conditioners.

Following the agitation conditioning circuit the pulp either is pumped or flows by gravity directly to flotations (not shown) or to a pulp distributor, at **33,** and to a number of banks of rougher and scavenger flotation cells, at **34.** Where a scavenger concentrate, a **35,** is produced, **1** prefer to return this concentrate to the last step of conditioning, at **31,** or in this case, the last agitator in the agitation conditioning circuit wherein a dispersing agent has been added to the second agitator, at **30.** Wherein the agitation conditioning circuit no dispersing agent is added to the second agitator, or prior to the second agitator, dispersing agent is added to the same agitator, at **30,** in which the scavenger concentrate is returned. Where the scavenger concentrate is returned to some other prior point in the agitation conditioning circuit, the dispersing agent may be added prior to or following its point of entry. Where one or more of the cleaner tailings are returned to any step in a prior agitation conditioning stage, and wherein one or more of the cleaner tailings have been produced by prior conditioning or agitation conditioning of a flotation concentrate there may be sufficient carryover of the dispersing agent to the agitation conditioning stage and no further addition of dispersant is required.

Where the scavenger concentrate is added to the head of the primary flotation circuit, prior to its return I prefer to condition the concentrate in one or more agitation conditioners, at **36,** with the addition of collector, particularly if there has been no carrying through of free collector from the primary flotation circuit. Where a single float is made and a rougher concentrate only is removed from the primary flotation circuit, I prefer to add all of this concentrate to either a thickener, at **37,** and then to a regrind circuit, at **38,** or alternatively, directly to a regrind circuit. In such a case the rougher tailings will be the final tailings. Where a rougher concentrate and a scavenger concentrate are produced in the primary circuit, the scavenger tailings will be the final tailings to waste. In the regrind circuit an alkali agent, usually lime, together with sulfhydryl anionic collector are added. Additional dispersant may be added to the grinding circuit, or alternately, to a conditioning or agitation conditioning stage, at **39,** following the regrind circuit. In the regrind circuit the alkaline agent may be added to control the pH below the optimum for conditioning or agitation conditioning ahead of the No. **1** Cleaner Float, at **40, I** also, or alternatively, will add the alkaline agent to this conditioning or agitation conditioning stage. The close control of all of the factors of pH, collector addition and dispersing agent concentration are important at this stage. Under normal conditions of application of one of the embodiments of the invention the No. **1** Cleaner Tailings will be low enough in copper values to reject as a final tailing together with the remaining amount of deleterious slimes. Alternatively if particularly there is a high percentage of combined middling particles, the No. **1**

Cleaner Tailings may be cycloned and the oversize returned to a regrind circuit (not shown), or alternatively, if there are some minerals such as oxide and silicate copper minerals which require further activation, I prefer to return either these tailings to a point within the original agitation conditioning circuit, at **28,** ahead of the primary flotation circuit, at **34,** or alternatively to a separate agitation conditioning stage with collector addition and prior to its return either at **28,** or to the head of the primary flotation circuit (not shown). The No. **1** Cleaner Concentrate may be conditioned, at **41,** with additional dispersing agent before being treated in the No. **2** Cleaner Concentrate may be the final concentrate, or additional cleaning steps carried out.

Where the ore contains appreciable amounts of pyrite, my preferred circuit is to carry out at least one agitation conditioing stage, at **43,** on the rougher concentrate, or on one or more flotation concentrates with or without added dispersant and collector at an optimum pH point within the pH range of about 8.0 to 12.0. This optimum pH point is generally in the pH range of 10.5 to 11.8. The agitation conditioning time is normally 1.0 to 6.0 minutes. My preferred method of adjusting the pH point is either with lime to raise the alkalinity to this point or with sulphuric acid to lower the alkalinity to this point.

With either the rougher plus one or more cleaners or one or more cleaner concentrates thus treated prior to the regrind circuit the final flotation concentrate prior to the regrind circuit is fed to the regrind circuit, at **44,** with or without further addition of dispersing agent. Following the regrind mill, I prefer to use at least one stage of agitation conditioning at **39,** with the addition of lime to bring the pH of the pulp up to a minimum of 12.3. If the collector is added to the head of the regrind mill, no further collector need be added to the cleaner flotation stage or stages following the regrind circuit. I prefer to use two stages of agitation conditioning following the regrind circuit where the pH is adjusted within the pH range of 12.3 to 12.6 using lime as the alkaline agent. A flotation stage is carried out following each agitation conditioning stage. In specific cases, such as where the pyrite has been activated with copper salts in the original ore deposit, it may be necessary to add a further pyrite depressant to the agitation conditioning stage or stages following the regrind circuit. In this case an alkali cyanide or zinc sulphate have been effective. The agitation conditioning stage or stages following the regrind mill are not to be confused with conditioning in an oxidizing atmosphere such as used with aeration at this point at high pHs wherein a sulfhydryl anionic collector such as xanthate is destroyed, deactivating the copper minerals and oxidizing the pyrite. Such a circuit, for instance where native copper is present, would also result in oxidation of the native copper and result in major copper value losses. In the use of air as an oxidizing agent during any of my agitation conditioning stages I have found that with the amount of air that is entrained such as in high speed agitation, it has little or no effect upon the metallurgy and thus it is thought that aeration in an agitation conditioning stage can play only a minor role in the results achieved. If the pulp is aerated to a high degree for prolonged periods of time, then detrimental effects on the metallurgy, particularly in recovery, have been experienced.

FIG. **3** - The flowsheet shown shows a conventional flowsheet for the depression of pyrite, following the primary flotation circuit.

FIG. **4** - The flowsheet shown shows my preferred process treatment following the primary flotation circuit with ores either containing a negligible amount of pyrite, or major amounts of pyrite.

In my preferred practice the concentrate or concentrates from the primary circuit are agitation conditioned at **53,** prior to the first cleaner float. If the primary concentrate has been prepared from a pulp that has been agitation conditioned according to my teachings, and the slimesfraction in the ore is low, a conditioning stage, at **53,** at this point in the circuit may be sufficient to substantialy reject the remaining slimes. If this agitation conditioning stage has not been employed I prefer to agitation condition a primary circuit concentrate either prior to or after regrinding on all types of ore.

If slime fraction is low in the ore, one cleaning stage, at **54,** prior to regrinding the cleaner concentrate, at **56,** is sufficient for good results.

However, if the slime fraction is high in the ore I prefer to use two or more cleaning stages (not shown), prior to regrind, at **57.**

Where the ore contains a negligible quantity of pyrite, at **58,** a conditioning stage, at **59,** may be sufficient to give acceptable results following the regrind circuit.

When the ore contains a significant quantity of pyrite, at **65,** I carry out at least one agitation conditioning stage, at **66,** or **69,** at a minimum pH of 12.3 in the presence of the collector and if necessary, an added pyrite depressant such as an alkali cyanide or zinc sulphate.

Where the regrind mill is in open circuit grinding, I prefer to have the pulp from the regrind mill fed to an agitation conditioning circuit, at **66,** containing at least two agitators in series with a minimum residence time of four minutes prior to at least one flotation stage.

Where the regrind mill is in closed circuit grinding, the pyrite, normally being much harder to grind than the copper minerals, circulates to a much higher degree and thus may be sufficiently conditioned to be depressed without further conditioning prior to being fed to the flotation stage, at **67,** following the regrind mill.

FIG. **5** - The flowsheet shown shows another conventional flowsheet using a Sands-Slime circuit.

In this case, following the preparation of the pulp, the pulp is classified into a sands fraction and a slimes fraction either before or after a primary float.

In my preferred practice as applied to such a circuit, my first agitation conditioning stage would be normally carried out on the slimes fraction only.

FIG. **6** - shows the preferred embodiment of my invention as applied to such a circuit following a primary float.

This primary float, at **84,** is normally what is termed a scalp float containing the readily flotable fraction of desired recoverable minerals. Following the primary float the tails are classified, at **85,** in either a mechanical or cyclone type classifier. The sands, at **86,** or coarse fraction are fed to a rougher flotation circuit, **87,** at **35** to **40** percent solids. The tailings from this circuit are final tailings and the concentrate is usually fed to a regrind circuit, at **88.** Following the regrind circuit this pulp, the concentrate at **84,** and the No. **2** Cleaner Slimes Concentrate at **97,** are brought together and either conditioned or agitation conditioned, at **89,** with usually the addition of dispersant and collector. If the majority of the slimes in the ore are not of a clay or talc nature, the dispersant is unnecessary

In the preferred embodiment of my invention I prefer to agitation conditioned for such flotation concentrate products 2 to 6 minutes at an optimum pH point within the pH range of about 8.0 to 12.0. The combined products from **89** are fed to a combined cleaner float to produce a No. **3** Cleaner Tailings at **91,** which I prefer to return to an agitation conditioning step within the agitation conditioning circuit at **93.** The No. **3** Cleaner Concentrate will normally be the final concentrate, or an additional cleaner step may be carried out. In this case the No. **4** Cleaner Tailings would be returned to the head of the combined cleaner float at **90** (not shown). The slimes product at **92** following the classifier would be agitation conditioned for preferably at least three minutes with the addition of an alkaline agent, collector, and if high in clay or talc, a high concentration of dispersant. I prefer to carry out two agitation conditioning steps at **93,** the first at an optimum pH point within the pH range of about 8.0 to about 10.5, and the second at an optimum pH point within the range of about 10.5 to about 12.0. Frother would normally be added to the last step of the final agitation conditioning step.

The pulp flows to the flotation stage **94** producing a concentrate and tails that are final tailings. The concentrate is fed to a conditioning or agitation conditioning stage at **95** with addition of a low concentration of dispersant, if initially used, together with additional collector if the carry through of collector **93** is insufficient. This pulp then flows to the No. **2** Cleaner Slimes Float at **96** producing a No. **2** Slimes Tailing at **98** which is returned to at least one step in the agitation conditioning circuit at **93.**

FIG. **7** - shows a conventional flowsheet for the flotation of a copper ore and the metallurgical results shown are described in Example XXXIII. It will be noted that lime is added to the grinding circuit at **99** and the pulp flows to a mixing tank where selective collector and frother are added, with the residence time of about 1 minute. From the mixing tank the pulp flows to a conventional circuit using a rougher plus scavenger float at **101.** The scavenger tailings are final tailings. In this case the scavenger concentrate is returned to the head of the rougher plus scavenger float circuit. The first cleaner tailings, for comparative reasons, were also returned to the float at **101.** Normally they would be rejected to waste. The rougher concentrate goes to a first cleaner float at **102,** and the pulp flow is conventional throughout. FIG. **8** - shows the flowsheet of my cleaner circuit only. It will be noted that at **110** in the laboratory closed circuit programme described in Example XXXIII, the rougher concentrate was conditioned for 1.0 minutes at a pH of 11.5 with the addition of sodium silicate and a powerful nonselective collector. potassium amyl xanthate. The pH was approximately 11.5 and controlled with the addition of sulphuric acid. This was necessary as the pH of the re-circulated pulps from the third and fourth cleaner stages were at 12.2 to 12.5, thus tending to raise the pH higher than the optimum pH point.

In the pilot plant run one conditioning stage only was used.

The next difference in the circuit was in the regrind at **113** where the pH was raised to about 11.7 with the addition of lime. Further collector was also added. The third significant difference is that following the regrind mill the pulp was agitation conditioned for ten minutes with two agitation conditioning tanks in series at a pH of 12.3 to 12.5. The fourth and fifth significant differences were that the third and fourth cleaner floats were carried out in a pH range of 12.3 to 12.5 and 12.2 to 12.5 respectively. In the closed circuit laboratory work described in Example XXXIII there was an additional agitation conditioning stage at **114** on the concentrate produced by the third cleaner. It will be noted that with the additional conditioning stage an appreciably higher grade concentrate was produced with no increase in copper value losses. It will be further noted that in using the conventional flowsheet for the production of the rougher concentrate, and with the limited application of my preferred cleaner circuit only an increase in copper value recovery of approximately 0.08 lbs. of copper per ton of ore and an appreciable increase in the copper concentrate grade was achieved. FIG. **9** - is part of the series of the pilot plant and laboratory work described in Example XXXIII and shows one of my preferred embodiments of heavily activating copper minerals in the ore. It will be noted that prior to the first alkaline conditioning stage the pulp from the grinding circuit at **117** flows to an acid conditioning stage at **118** with all steps using high speed agitation conditioning. Following the third agitation conditioning stage at **120,** the pulp flows to the rougher plus scavenger float at **121** and the flowsheet from this point on is identical to FIG. **8.**

It will be noted that as compared to the conventional flowsheet at FIG. **7,** not only is the grade of concentrate appreciably higher, illustrating the effective depression of the pyrite and insol, but also the increase in copper value recovery is approximately 1.3 pounds of copper per ton of ore. FIG. **10** - shows a typical acid conditioning stage prior to an alkaline stage, Where I use such a stage I prefer to add the acid agent on the basis of pounds per ton of ore and condition for a minimum period of 6 minutes in at least three agitation conditioning steps. FIG **11** - shows a typical alkaline agitation conditioning stage wherein the pH of the pulp in the first agitator starting at approximately 11.8, drops to approximately 11.3 in six minutes and levels out at this point. Where I use one alkaline agitation conditioning stage between the grinding circuit and rougher flotation I prefer to use a minimum of two agitation conditioning steps and consider three optimum. The agitation conditioning time at this point in the circuit will vary from a minimum of about 4 minutes to about a maximum of 45 minutes. Where I use an alkaline agitation conditioning stage, conditioning one or more concentrates either alone or with re-circulated cleaner tailings, a single agitator may suffice. In the application to these points in the circuit the agitation conditioning time is a maximum of 20 minutes. FIG. **12** - shows an acid conditioning stage preceeding an alkaline agitation conditioning circuit. The pulp from the grinding circuit is introduced into the first agitator **122,** at **125,** and the acid agent, usually sulphuric acid, is introduced through the line at **126.** At **123** is an impellor driven at high speed. The pulp discharges from the tank **122** through the launder at **127** into tank **128** where the pulp is agitation conditioned with two impellors at **123A** and **123,** mounted on the single shaft at **129.** The pulp continues to flow to tank **133** and to **136** where, as shown in FIG. **10,** the pH will have risen over the intial pH in tank **122.** The pulp will overflow tank **136** through the launder at **138** and into tank **139** through **141** in FIG. **13.** The Alkali agent, usually lime, is added to this tank and the amounts of lime and the pH con-

trolled through a pH meter, at **145.** I prefer to add the collector and dispersant if used, to tank **140** following tank **139.** The pulp from tank **140** overflows into tank **147,** from tank **147,** to tank **148,** from tank **148** to tank **149,** and from tank **149** to tank **153** wherein frother is added to the pulp and the pulp overflows tank **153** to the primary flotation circuit. Where two stages of alkaline agitation conditioning are used, additional alkaline agent would be added to tank **148** and the higher pH level controlled from the pH meter at **155.**

In using an essentially all alkaline agitation conditioning circuit FIG. **12,** eliminating the acid addition to Agitator **122,** may be used in a number of ways. Agitators No. **122** and **128** are small agitators with relatively high power input using a single agitation mechanism. The resultant agitation is vigorous to violent. In my preferred design these agitators are equipped with baffles as shown in FIG. **14** at **158** and **164** and FIG. **14***a* to **158***a*. The vertical pulp discharge line shown in FIG. **14** at **161** would be omitted and the pulp overflow Agitator **122** at **127** into Agitator **128** and overflow Agitator **128** at **130** to Agitator **133.** A combination of both overflow and sands discharge line may be used.

Agitators **133** and **136** are large agitators and my preferred design as shown in FIGS. **14** and **14***a* using a combination of baffles as shown at **158, 164,** and **158***a*, and the vertical discharge lines at **161** and **167.** Single agitation mechanisms may be used as shown in FIG. **14** at **157** and **166,** or multiple agitation mechanisms as shown in FIG. **15** at **172, 173** and **174.**

The resultant agitation is vigorous.

In the preferred embodiment of my invention part or all of the alkaline agent may be added to the grinding circuit (not shown) prior to the prepared pulp of the ore entering the agitation conditioning circuit, part or all to Agitators Nos. **122, 128** and **133.** I prefer to add only part of the alkaline agent to the grinding circuit wherein the pH of the pulp entering the agitation conditioning circuit at **125** in FIG. **12** is lower than the optimum pH point of the pulp in at least Agitator **136** wherein the pH is at an optimum pH point within the pH range of 8.0 to 12.0. Part or all of the collector may be added to the grinding circuit or to any agitator or combination of agitators. In this illustration of FIG. **12** where the pulp discharges from Agitator **136** at **138** and feeds to flotation there must be at least in Agitator **136** or in combination with at least one other agitator sufficient concentration of collector and a sufficiently long period of time to heavily activate at least the desired sulphide minerals. When a dispersing agent is used it may be added at the same points and in the same manner as the collector. Preferably it is added concurrently or following the at least first addition of collector, and in sufficient amount and for a sufficiently long period of time to not only assist in the heavy activation of the desired sulphide minerals but also the effective depression of the waste host rock minerals in subsequent flotation.

In the preferred embodiment of my invention where I use at least two agitation conditioning steps, in FIG. **12** the prepared pulp of the ore may be fed to Agitator **122** as shown at **125,** or alternately to Agitators **128** or **133,** wherein the discharge of the pulp from Agitator **136** at **138** is to flotation.

The following Table I, with an agitation conditioning circuit, treating 500 tons per hour. shows the preferred calculated residence time of the ground ore in the various agitators as shown in FIG. **12** and the minimum and maximum horsepower drawn by the agitation mechanisms.

Table II shows the various combinations of agitators that may be used (with by-pass launders for the pulp flow not shown), the total residence time of the ore, the minimum and maximum total power drawn by the agitation mechanisms, and the minimum and maximum total power consumption. As Agitators **122** and **128** will have approximately the same residence time of the ore, and similarly **133** and **136,** where I refer to a single agitator in the circuit as **122,** it could be **128,** and similarly **136** could be **133.**

The "Drawn Horse Power" referred to of the agitation mechanism or mechanisms may be the installed horse power, or lower than the installed horse power. The "Drawn Horse Power" is the actual horse power drawn by the agitation mechanisms motor, or motors.

TABLE I

Agitation Step No.		Approx. Residence Time of Ore	Drawn Horse Power Agitation Mechanisms	
No.	FIG. 12	Mins.	Min.	Max.
1	122	1.0	40	80
2	128	1.0	40	80
3	133	4.0	75	275
4	136	4.0	75	275

TABLE II

Agitation Conditioning Steps Used	Approx. Residence Time of Ore Minutes	Drawn H.P. Agitation Mechanisms		Power Consumption K.W. Hours per Ton of Ore Treated	
		Min.	Max.	Min.	Max.
1 and 4	5.0	115	355	0.17	0.55
1, 2 and 4	6.0	155	435	0.23	0.65
3 and 4	8.0	150	550	0.22	0.82
1, 3 and 4	9.0	190	630	0.28	0.94
1, 2, 3 and 4	10.0	230	710	0.34	1.06

FIG. **13** - shows a preferred embodiment of the invention in using an all alkaline agitation conditioning circuit.

In the preferred embodiment of the invention the pulp entering Agitator **139** at **141** would be pumped or flow by gravity directly from the grinding circuit (not shown), or from a thickener (not shown) between the grinding circuit and Agitator **139.** The pulp density is adjusted prior to or in Agitator **139** adding fresh water or mill solution. It may be further adjusted at any point in the circuit.

The pulp density to flotation may be adjusted at any one or more agitators, or following its discharge at **155.** In all cases it is a downward adjustment.

If the ore is low in deleterious slimes a dispersant or flocculant may be entirely omitted. In this agitation conditioning circuit where I prefer small agitators for violent agitation, Agitators **139, 140** and **153** would use violent agitation, and **147, 148** and **149***a* vigorous agitation.

All agitators would preferably be equipped with baffles (shown in FIGS. **14** and **15**), and Agitators **147, 148** and **149***a* with baffles and special internal discharge lines, as shown in FIGS. **14** and **15.**

The pulp discharges from the last agitator **153,** in the agitation conditioning circuit, and at **155** will either flow by gravity or be pumped to the head of the primary flotation circuit, or to a distributor before feeding to the flotation circuit.

FIG. 14 shows my preferred design of agitator in which varied agitation conditions can be produced together with being able to handle coarsely ground ores at the lower end of my density range without sanding up or necessitating excessive agitation conditions and thus excessive power to keep coarsely ground particles of the ore in suspension.

With this design of agitator I am able to adjust the speed of my agitation mechanisms to obtain optimum power input to the pulp to produce maximum sulphide mineral recovery immaterial of normal variances in pulp density or coarseness of the ground product.

The pulp enters agitator **159** through launder **156** and finally passes into the vertically enclosed discharge line **160** through the opening **161** and is discharged into launder **162** and to agitator **165.**

With the pulp discharging through **161,** sands will not build up in the agitator beyond this point.

The baffles **158** not only reduce short-circuiting but result in appreciably more power input to the ground ore particles.

The impellor shaft **157** may be driven by various conventional means at **157A.** I prefer a motor driving through a series of gears that may be quickly changed to give a higher or slower speed, or alternately, using V-belts or a variable speed motor (not shown).

Where I use small agitators, that is, less than about **12** feet in diameter and vigorous to violent agitation, I may omit the bottom discharge arrangement shown at **161,** and the pulp in this case will overflow into a launder (not shown) and flow to the next point in the circuit.

FIG. **14A** is a section through the agitator **159.**

FIG. **15** shows a preferred design for large diameter agitators to obtain my preferred optimum energy input and rate of energy input to the solids in the pulp.

Instead of using a single driven agitation means arrangement I may use a multiplicity of agitation means, and of different types. In this illustration the main agitation means is the driving of the impellor shaft **173** by the driving means at **173*a*** using shaped impellors **178,** and **179.**

The other agitation means are the high speed rotors **180** and **181,** attached to their respective shafts **172** and **174,** and driven through the driving mechanisms at **172*a*** and **174*a*** respectively.

In this illustration the shaped impellors **178** and **179** are sufficient to create vigorous agitation conditioning within the agitator. However, the energy input to the pulp as measured by the power drawn by the motors and for the residence time of the pulp in the agitator, is insufficient to obtain the optimum heavy activation of the sulphide minerals. By using agitation means in the same agitator such as the high speed rotors **180** and **181,** the required additional energy input to the pulp is obtained. The research work indicates that excess power from a single agitation mechanism will not produce the same optimum metallurgy as a number of units operating at an optimum speed and thus drawing an optimum amount of power. It will be appreciated that the speed of rotation of each agitation conditioning mechanism must reach a minimum rate of speed to achieve optimum results of the invention. It will be further appreciated that the power drawn by the motors can be measured and translated into kilowatt hours per ton of ore treated.

The relative figure of the total required horsepower drawn by the motors of the agitation conditioning mechanisms to the energy input to the pulp in an agitation conditioning circuit I describe as the sufficient power to produce heavy activation of the sulphide minerals when in combination with other features of the invention.

In using the number of individual agitation mechanisms in an agitation conditioning circuit, whether it is a single agitation mechanism in each of a number of agitators or multiple agitation mechanisms in one or more agitators I describe as stage agitationed conditioned.

By being able to use large diameter units, capital costs and floor space are reduced, particularly where a large tonnages and long agitation conditioning time periods are required to obtain optimum results from my invention.

FIG. **16** shows a flowsheet of a preferred embodiment of my invention. The ore **210,** is fed to a crushing plant **211,** and then to a wet grinding circuit at **182.** To the grinding circuit no reagents need be added, or alternately, part or all of the alkali or acid agent, part or all of the collector, and if an activator such as copper sulphate is used, part or all of the activator. If a wetting agent is used I prefer to add all of the wetting agent at this point. If a dispersant is used I prefer to add none or only part of the dispersant at this point.

Following the grinding circuit the pulp may flow or be pumped, at **184,** to the number one step, **190,** of the agitation conditioning circuit or alternately, **185,** be sent to a thickener, **186,** and from the thickener at **188,** be pumped to the number one agitation conditioning step. In either case the pulp density may be adjusted at **189** prior to entering the No. One agitation conditioning step. The pulp from No. **1** agitation conditioning step overflows to No. Two at **191,** from No. Two to No. Three at **192,** from No. Three to No. **4** at **196,** or through a pulp distributor at **195** to flotation at **196.**

In this example the first two agitation conditioning steps are violent, and the last two vigorous.

The main places in the circuit that I prefer to use a violent agitation conditioning step is where I add an acid agent or alkali agent to radically change the pH of the pulp. By such means the agent is more rapidly absorbed by the pulp and the pH of the pulp reaches equilibrium in a shorter residence time.

Where I use violent agitation conditioning steps they are normally small tanks with residence time of the pulp in an individual step not exceeding about 2 minutes.

As will be noted from the flowsheet a scavenger concentrate, scavenger tailing from the cleaner circuit, or any of the cleaner tailings may be returned to any desired agitation conditioning step in the agitation conditioning circuit.

FIG. **17** is a graph showing the effect of pulp density on the amount of acid agent used where an acid conditioning stage preceeds the final agitation conditioning stage prior to flotation in the pH range of about 8.0 to 12.0.

The first ore, graph line **217,** had been crushed, conventionally leached in situ with sulphuric acid, washed and then ground in the grinding circuit prior to my acid conditioning stage.

The graph shows optimum metallurgy obtained at various pulp densities with the amount of sulphuric acid at each density giving the lowest tailings loss.

The graph line at **218** shows the same data on feed from the same ore body which was crushed and ground without any prior metallurgical treatment.

Both of these copper ores were samples from the mill feed of a major producer in Arizona.

The surprising feature of these results was that the amount of sulphuric acid required to produce the lowest flotation tailings was a function of the pulp density, decreasing as the pulp density increased.

Although the pHs in the acid conditioning stage were all in the pH range of 1.2 to 6.5, the end pHs at the end of the acid conditioning stage varied appreciably.

For optimum results this data indicated that the acid agent should be added to the pulp on a pounds per ton basis at a specific pulp density.

FIG. **18** shows preferred embodiments of my invention with pulp flow diagrams in treating the suitably prepared pulp of the ore directly from the grinding circuit or following a thickener stage. The preferred agitation conditioning stages are shown at **220, 221, 222,** and **223.**

Reagents that may be fed to the various agitation conditioning stages are number **1** to **11***a* inclusive, and are as follows: **1** is no reagents; **2** is Wetting Agent; **3** is Frother; **4** Dispersing Agent; **5** Sulphidizing Agent; **6** Sulfhydryl Collector; **7** Alkali Cyanide; **8** Alkaline Agent; **9** Acid Agent; **10**(*a*) Copper Sulphate; **11**(*a*) Flocculating Agent other than lime or calcium hydroxide. Where the flocculating agent **11**(*a*) is added to the agitation conditioning stage **221,** the pulp flow from **221** is direct to flotation at **224.**

"Min. req. R of P drawn" is shortened for "Minimum required Rate of Power drawn." "Min. req. P of T" is shortened for "Minimum required Period of Time" of the ore residence in the specific agitation conditioning stage "Kw. Hrs. per Ton" is Kilowatt Hours per ton of ore treated.

It will be noted that each agitation conditioning stage requires a minimum total power drawn by the agitation mechanisms. In terms of the invention the prime object of the power drawn by the agitation mechanisms is to assist in heavy activation of the mineral particles and particularly the valued sulphide minerals. This total power drawn is usually in large excess over that required to keep the ground ore particles in suspension with the ore finely ground for sulphide mineral flotation and in the preferred pulp density range of 30 to 45 percent solids.

With coarsely ground ore, in the pulp density range of about 20 to 35 percent solids, the power drawn may be lower than required to keep the coarsest particles in suspension.

EXAMPLES OF THE INVENTION

A more detailed understanding of the invention may be had by reference to the following examples of laboratory testing on three different types of ores occurring in the same major copper deposit in the United States of America. These ores are identified as follows:

Lot 128	-High Clay Content
Cu (Total	0.95%
Cu (Acid -Soluble	0.23%
Fe	5.45%
S	4.22%
SiO_2	60%
CaO	0.25%
M_gO	0.35%
Al_2O_3	15.7%
Lot 126 — Similar to 128 but higher in pyrite content	
Cu	1.23%
Cu (Acid-Soluble)	0.17%
Fe	7.95%
S	10.1%

Lot **130** - this sample was a composite of daily concentrator feed samples, and could be described as medium in its clay and talc composition.

Cu (Total	1.09%
Cu (Acid-Soluble)	0.36%
Fe	7.55%
S	3.23%

The orebody from which these samples were taken is that of a major copper producer in the United States and the producer's standard laboratory tests gave the following metallurgy. All three samples contain varying amounts of talc and/or clay.

Lot 128	
Concentrate — % Cu	13.2%
% of Total Cu in Concentrate	75.8%
Rougher Tailing — % Cu	0.24%
Lot 126	
Concentrate — % Cu	8.85%
% of Total Cu in Concentrate	74.93%
Rougher Tailing — % Cu	0.35%
Lot 130	
Concentrate — % Cu	15.3%
% of Total Cu in Concentrate	75.56%
Rougher Tailing — % Cu	0.27%

The foregoing may be considered as indicative of the best metallurgy obtainable on these ores in accordance with the established practice of this operator and the recoveries indicated were accepted as the maximum possible recovery for these ores at the concentrate grades obtained.

In the examples, in all cases but one grinding was carried out on 800 gram charges in a laboratory rod-mill to a screen analysis of approximately 100 mesh, and at a pulp density of 50 percent solids. The pulp was transferred directly from the rod mill to a 600 gram laboratory Fagergren cell, and all conditioning and primary flotation was carried out in this cell.

In using the Fagergren laboratory cell the optimum speed for both the agitation conditioning and the flotation was 2000 rpm. For conventional flotation the generally accepted optimum speed was 1800 rpm. The increase in speed from 1800 to 2000 rpm is a comparatively large increase in energy input to the pulp. No desliming stage was used.

All cleaning steps were carried out in a Denver 250 gram cell.

Except where noted in the specific tests the rougher flotation time was 7 minutes, and in the cleaner the float was continued in each case to final clean-up with no set time. The average cleaner float time was about 4 minutes. The rougher float time is exceptionally low, as with this class of mineral, for any degree of recovery in a conventional circuit, a float time of fifteen minutes would be required.

With an 800 gram charge to a 600 gram Fagergren cell, the density of the pulp was approximately 30 percent solids. This is an unusual feature of the invention in view of the talc and clay contents of the ores treated.

In the following series of tests the rougher tailing and first cleaner tailing are shown only for comparative metallurgical reasons, and where the ratio of concentration at this point in the circuit is the range of about 10–12 to 1. In all cases on the three samples, complete circuits are described illustrating the production of concentrate grades in excess of 20 percent copper.

The copper analysis was done by the current standard atomic absorption method with a number of testing programme samples check analyzed by the supplier of the various ore samples. Where the term "corrected" is used, this indicates application of the correction factor necessary to relate the laboratory analysis to the check analysis of the supplier. To the combined rougher and first cleaner tailing, to correct to the supplier's analysis, the rule of thumb correction factor is plus 0.01 percent copper.

In both an all alkaline circuit and a circuit where a preliminary acid stage was used an average of 2½ percent of the total copper reports in solution. Except in the tests showing the solution analysis, all analyses are on the solids only.

LOT 128 - HIGH CLAY CONTENT

EXAMPLE I (TEST 8)

The sample was ground with 8.75 lbs. per ton of sodium carbonate producing a pH of 9.85. The pulp was then conditioned with 0.15 lbs. per ton of Potassium Amyl Xanthate (Z6) for 15 minutes.

20 lbs. per ton of sodium silicate was then added to the pulp for 5 minutes together with five drops of pine oil for the last 3 minutes. The final pH was 10.0.

The rougher float was conducted for seven minutes with the addition of 0.025 lbs. per ton of Z6 and one drop of pine oil.

The rougher concentrate was conditioned in the Denver cell for 3 minutes with 0.025 lbs. per ton of Z6 and 2 lbs. per ton of sodium silicate and floated to clean-up.

The rougher plus cleaner tailings was 91.8 percent by weight analyzing 0.077 percent copper. In comparison to the conventional test where the rougher tailings loss was 0.24 percent Cu, this represents a major increase in recovery.

It will be noted that in this ore, which contains a high clay content, prior to the rougher flotation an agitation conditioning stage was used wherein a large concentration of sodium silicate was added to the pulp in the presence of the sulfhydryl anionic collector and conditioned for a period of 5 minutes prior to flotation. Again, the rougher concentrate was agitation conditioned for a period of three minutes with the addition of 2 lbs. per ton of sodium silicate and 0.025 lbs. per ton of sulfhydryl anionic collector.

In the continuous plant practice the power consumption of the agitation mechanisms employed in the agitation conditioning circuit prior to rougher flotation would be an estimated 1.0 kilowatt hours per ton of dry ore fed to the circuit.

In this major application of my invention wherein the agitation conditioning circuit is located between the grinding circuit and rougher flotation, in treating ores containing copper sulphide values or ores containing copper and molybdenum sulphide values the maximum estimated power requirements for the total agitation mechanisms is about 5.0 kilowatt hours per ton of ore treated.

The power consumption of the agitation conditioning circuit treating the rougher concentrate would be an estimated 0.15 kilowatt hours per ton of solids fed to the circuit.

EXAMPLE II (TEST 5)

This test was identical to Example I with the exception of lime replacing the sodium carbonate in the first stage to bring the original pulp during agitation conditioning with the collector to the considered optimum pH. The pH at the end of the conditioning with Z6 (potassium amyl xanthate), (American Cyanamid) was 11.65. However, following the addition of the strong alkali dispersant, sodium silicate, the pH immediately dropped to 11.1 and the rougher flotation was carried out at a pH of 10.9. Why this drop in pH occurs is not known.

The rougher plus first cleaner tailing was 91.8 percent by weight analyzing 0.072 percent Cu. The rougher tailings was 88 percent by weight, and the first cleaner tailings 3.8 percent by weight. One of the surprising results of this test was that even though the activation cycle with Z6 was at the considered optimum pH the final pH to flotation was appreciably lower than the optimum for conventional practice, that is 11.6.

This test indicated that the previously determined optimum flotation pH for these minerals was radically changed.

Although the lime is a flocculant, in comparison to Example I where soda ash, a dispersant, was used, there was no significant difference in final metallurgy.

One of the features of this test was the large amount of sodium silicate that was used in producing such outstanding copper value recovery. In the first agitation conditioning stage between the grinding circuit and rougher flotation, 20 lbs. of sodium silicate per ton of ore was used, and in the second agitation conditioning steps on the rougher concentrate and ahead of the first cleaner float 16.6 lbs. of sodium silicate per ton of rougher concentrate was used.

Such concentration of dispersant in conventional copper ore treatment are unheard of and would be considered intolerable due to heavy copper value losses.

In treating such ores containing a relatively high percentage of slimes with clay or talcose type waste host rock minerals, I prefer to depress the slimes in at least one agitation conditioning stage between the grinding circuit and the rougher flotation circuit, and treat at least one flotation concentrate to a further agitation stage or conditioning stage in the presence of a dispersant to further depress the remaining slimes.

As will be seen in a number of ensuing illustrations of one of the preferred embodiments of the invention the amount of dispersant used in this illustration was in appreciable excess of the amount required for effective depression of clay and talcose slimes with resultant outstanding copper value recovery.

To determine the amount required I increase the dispersant to the point where copper value recovery is no longer increasing. This is usually in excess of the amount required for the effective depression of the waste non-sulphide host rock minerals. In steadily increasing the amount of dispersant I have already determined the minimum amount required for maximum copper value recovery and in plant practice prefer to use 5 to 20 percent in excess of this amount to allow for short-circuiting that takes place in the agitation conditioning units that are arranged in series. In describing

this amount of dispersant I use the term "sufficient amount." In plant practice this amount of dispersant addition may be reduced due to build-up of the dispersant in the circulated mill solution. Also the time of agitation conditioning is important particularly on the original pulp of the ore or following a primary scalp float.

If the time is too short neither heavy activation of the copper minerals nor effective depression of the non-sulphite host rock waste minerals will be achieved.

Reasonable excess time has no deleterious effect. The minimum period of time plus a reasonable excess, 10 to 25 percent for the residence time of the pulp in an agitation conditioning stage for heavy activation of at least the sulphite copper minerals and effective depression of the waste non-sulphide host rock minerals in subsequent flotation, I describe as "of sufficiently long duration."

The amount of total collector, Butyl Xanthate, used in the conventional circuit on this ore was reported to be 0.07 lbs. per ton. It will be noted that in my invention 0.15 lbs. Amyl Xanthate was used in the agitation conditioning stage prior to rougher flotation and 0.025 lbs. per ton in the agitation conditioning stage on the rougher concentrate prior to the first cleaner float.

This comparatively large concentration of collector, in the two agitation conditioning stages alone, is a usual characteristic of my process even in cases where I use lower overall amounts.

EXAMPLE III (TEST 1)

To the grinding stage 5.0 lbs. per ton of lime and 5.0 lbs. per ton of soda ash were added resulting in a pH of 11.4. 0.20 lbs. per ton of **Z6** was added and the pulp was conditioned for 30 minutes. Following this stage 0.75 lbs. per ton of a flocculant was added (a locust bean gum). The pulp was conditioned for seven minutes with the final addition of pine oil as a frother. The rougher float was 8 minutes with the addition of 0.025 lbs. per ton of **Z6** and one drop of pine oil. The cleaner used 4 minutes of pre-conditioning with 2.4 lbs. per ton of soda ash, 0.025 lbs. per ton of **Z6**, and 0.25 lbs. per ton of locust bean gum. The combined rougher and first cleaner tailing was 90.6 percent by weight analyzing 0.09 percent Cu. This was surprisingly good metallurgy for the substitution of a flocculant for this cycle in place of a dispersant, and although the metallurgy was poorer than Examples I and II, it was appreciably better than the conventional circuit.

This test showed that where an all lime circuit was not used in the initial stages, in the final stage either a dispersant or flocculant could be applied.

EXAMPLE IV (TEST 34)

In this example 0.5 lbs. per ton of soda ash was added to the grinding stage with the pH at the end of the grind being 6.45. To the first conditioning stage 6.8 lbs. of lime and 0.1 lbs. of NaCN per ton respectively were added for a conditioning time of twelve minutes. The pH at the beginning of the cycle was 11.8. The cyanide was added for pyrite depression. The next cycle was twenty minutes conditioning with 0.17 lbs. of **Z6** stage added.

In the last stage before floating 7.5 lbs. of sodium silicate per ton was added and conditioned for 5 minutes with three drops of pine oil for the last 3 minutes.

The rougher float was seven minutes with the addition of 0.025 lbs. of **Z6** per ton and one drop of pine oil.

The first cleaner float used four minutes pre-conditioning of the rougher concentrate with 0.375 lbs. of lime, 0.5 lbs. of sodium cyanide, and 1.25 lbs. of sodium silicate per ton respectively.

The second cleaner float used 6 minutes pre-conditioning of the first cleaner concentrate with 1.0 lbs. lime, 0.10 lbs. sodium cyanide and 0.5 lbs. sodium silicate per ton respectively.

The following "corrected" metallurgy was obtained.

Product	% Wt.	Analysis % Cu	% Cu Distribution
Concentrate	3.3	21.6	79.8
No. 2 Clnr. Tlg.	1.1	2.07	2.5
No. 1 Clnr. Tlg.	7.0	0.48	3.8
Rougher Tlg.	88.6	0.14	13.9
	100.0		100.0

In comparing the tailings loss to that in the previous example it is obvious that cyanide is detrimental to recovery when used in the first conditioning cycle. However, in comparing the open circuit metallurgy to the conventional standard metallurgical test wherein the concentrate grade was 13.2 percent and copper recovery 75.8 percent, the concentrate grade from this circuit is more than 50 percent higher with an increase in copper value recovery, even though the use of the cyanide as pyrite depressant had not been optimized.

With conventional flotation practice the insol in the final concentrate on this type of material will average 25 to 30 percent. The insol in the concentrate of this Example IV was 6.4 percent. This is an amazingly low percentage of insol, (waste non-sulphide host rock minerals), as it is appreciably lower than in conventional practice on ores with little or no slime problem. It will be noted that in Example I, a high speed agitation conditioning stage was carried out prior to the rougher flotation wherein the dispersing agent, sodium silicate, was added to the pulp in the presence of a sulfhydryl anionic collector, which again had been preagitation conditioned for a period of 20 minutes. It will be noted that ahead of each cleaner high speed agitation conditioning was used in the presence of additional dispersing agent. There was, in this case, sufficient sulfhydryl anionic collector carried over from the rougher circuit.

LOT 126 - HIGH TALC HIGH PYRITE CONTENT

Example V (Test 37)

In the grinding stage 2.5 lbs. per ton of soda ash was used with the pH at 7.0.

7.0 lbs. of lime, and 0.10 lbs. of sodium cyanide were added to the next stage for 12 minutes conditioning with a pH of 12.0 at the start and 11.85 at the end of the cycle.

In the next cycle 0.17 lbs. of **Z6** per ton was stage added and conditioned for 20 minutes. The end pH was 11.75.

In the final cycle before flotation, 7½ lbs. per ton of sodium silicate was added and three drops of pine oil for the last three minutes. The conditioning period was five minutes. The rougher flotation was twelve minutes with the addition of 0.025 lbs. **Z6** per ton and one drop of pine oil.

The first cleaner float used six minutes per-conditioning of the rugher concentrate with 0.75 lbs. of lime.

0.075 lbs. of sodium cyanide and 1.25 lbs. of sodium silicate per ton respectively. The pH was 11.7.

The second cleaner float used eight minutes pre-conditioning with 1.25 lbs. lime, 0.10 lbs. of sodium cyanide, and 0.0625 lbs. of sodium silicate per ton respectively. The pH was 12.2.

The following metallurgy was obtained:

Product	% Wt.	Analysis % Cu	% Cu Distribution
Concentrate	4.1	22.8	79.1
No. 2 Clnr. Tlg.	6.8	0.73	4.4
No. 1 Clnr. Tlg.	20.8	0.35	6.2
Rougher Tlg.	68.3	0.18	10.3

It will be noted that as in Example IV, the cyanide has not been optimally used. However, with the basic circuit of the invention, the concentrate grade is in excess of 100 percent higher than the standard, together with an increase in copper recovery.

The insol in the final concentrate was 1.6 percent. This is a fantastically low figure considering the excellent comparative metallurgy achieved. Further, only two cleaners are used. As in the previous examples it will be noted that prior to the rougher flotation an agitation conditioning stage was used wherein sodium silicate was added to the pulp in the presence of a sulfhydryl anionic collector and the period of conditioning was 6 minutes.

In this case the conditioning of the rougher concentrate was carried out in the presence of a dispersant, sodium silicate, a small amount of sodium cyanide, and the pH adjusted to 11.7 with lime. It was unnecessary to add further sulfhydryl anionic collector as there was sufficient carry through to free collector from the rougher stage. The first cleaner concentrate was conditioned in a similar manner prior to the second cleaner float for a period of eight minutea again without the requirement for further sulfhydryl anionic collector addition. The pH was adjusted with lime to 12.2.

I have found the optimum pH, prior to regrinding to a cleaner flotation concentrate is within a comparatively small range and can be important in the effect on both recovery and concentrate grade, and in the effective elimination of insol and pyrite. In the first cleaner stage where agitation conditioning or conditioning, is carried out prior to the first cleaner, the optimum pH range is generally about 10.5 to 12.2. This range applies to all cleaners prior to regrinding.

If pyrite is not a problem in producing a high grade concentrate, I prefer to agitation conditioning, or condition, with the use of a dispersant and free sulfhydryl anionic collector in the pulp in the pH range of about 9.5 to 12. This applies to circuits whether or not a regrind stage of one of the concentrates is required.

Where a regrind stage is required and pyrite in the ore is a problem in producing an acceptable grade of concentrate, following the regrind mill both in conditioning or agitation conditioning and flotation my preferred range for the pyrite depression is a pH of about 12.3 to 12.6 using lime alone as the alkali agent. Where other alkalies such as sodium or potassium hydroxide are used in combination with lime, the pH range may be extended to about 12.9.

In plant practice it is very often difficult to accurately measure these high pHs. For this reason I prefer to control the circuit at in the same pH range using the method of measurement by titrating the free lime with a known strength of sulphuric acid solution. This is a well known method of measuring free lime and once the amount of free lime in the solution is established in the equivalent pH range, it can be closely controlled.

EXAMPLE VI (TEST 42) - LOT 130

The following metallurgy was obtained using cyanide, and lime in the cleaners. The chemical analyses of the various products are corrected and were as follows:

Product	% Wt.	Analysis % Cu	% Cu Distribution
Concentrate	2.5	35.7	83.5
No. 3 Clnr. Tlg.	2.7	2.1	5.3
No. 2 Clnr. Tlg.	2.8	1.5	3.9
No. 1 Clnr. Tlg.	12.6	0.19	2.3
Rougher Tlg.	79.4	0.067	5.0
	100.0		100.0

Not only is the grade of concentrate increased by in excess of 100 percent over the standard but also the open circuit recovery in copper values is appreciably increased, even before taking into consideration the additional copper recovery that would be made on closed circuiting of the second and third cleaner tails.

The following circuit was used. To the grinding stage 0.625 lbs. per ton of soda ash was used to produce a pH of 7.55. The first conditioning cycle was 2 minutes with 8.5 lbs. of lime per ton resulting in a pH of 11.75. The second conditioning cycle was nineteen minutes with 0.225 lbs. Z6 per ton added in stages. The pH at the end was 11.6. The third and last conditioning cycle before flotation as 5 minutes with 6.25 lbs. per ton of sodium silicate and three drops of pine oil for the last 2 minutes. The sodium silicate dropped the pH to 11.4.

The rougher float was seven minutes with the addition of 0.025 lbs. of Z6 per ton and one drop of pine oil.

Prior to the first cleaner float the rougher concentrate was agitation conditioned for three minutes with a small addition of potassium amyl xanthate and approximately 1 pound per ton of sodium silicate.

Following the first cleaner, the cleaner concentrate was filtered and reground with lime to produce a pH of 12.15.

Following the regrind stage the pulp was placed in the 250 gram Denver Cell and conditioned with 0.1 lbs. per ton of NaCn and 0.625 lbs. per ton of Na_2SiO_3.

In the third cleaner the pH was raised to 12.35 with lime and 0.10 lbs. of NaCN per ton of ore and the pulp was conditioned before floating.

On both the second and third cleaners it was surprising that using a pH much higher than the optimum resulted in only low drops in copper values in the cleaner tailings. This test illustrates the "heavy activation" of the copper minerals that was obtained prior to the rougher float and maintained during cleaning.

The grade of concentrate produced in this test, which was in excess of **100** percent higher than in the conventional circuit with higher copper value recovery, is surprising in that only three cleaners were used. It will be noted that following the regrind the No. **2** Cleaner was at a pH of 12.15 and the No. **3** Cleaner at a pH of 12.35. In combination with the use of a dispersing agent and the specific pH ranges, together with sulfhydryl anionic collector to the first cleaner float, gave ex-

cellent rejection of both the slime waste non-sulphide host minerals and the pyrite to produce an amazingly high grade concentrate together with improvement in the copper value recovery.

EXAMPLE VII

A series of tests was run using different alkaline agents to control the pH to the conventional "optimum" of 11.6 in the activation cycle with xanthate.

The circuit was as follows:

Grinding Stage - 12 minutes with alkali agents shown in Col. 1

First Cycle - 2 minutes conditioning with lime bringing pH to approximately 11.6

Second Cycle - 19 minutes conditioning with stage addition of Z6

Third Cycle - 5 minutes conditioning with sodium silicate and pine oil

Flotation Stage - 7 minutes with addition of 0.025 lbs. per ton Z6 and one drop of pine oil

1st Cleaner - 3 minutes conditioning with 0.025 lbs. Z6 and 1.25 lbs. sodium silicate respectively followed by flotation to clean-up.

The lime cycle was 2 minutes followed by nineteen minutes conditioning with 0.225 lbs. Z6 stage added, and this cycle was followed with the addition of 7.5 lbs. per ton of sodium silicate for 5 minutes with three drops of pine oil added in the last 2 minutes. The rougher float was seven minutes with the addition of 0.025 lbs. per ton and one drop of pine oil.

The first cleaner was identical in all cases as to reagent balance and time of conditioning.

The following table shows the comparative results.

Test No.	Grinding Circuit Reagents Added	pH End	First Cycle Reagents Added	pH Begin	pH End &	lbs./ Ton	Rghr. + 1st Clnr. Tlg. Analysis % Wt.	% Cu
1	Soda Ash (47) 0.625 lbs.	7.7	H_2SO_4 2.5 lbs.	4.7	5.4	(1)	90.1	0.057
2	Lime (48) 0.20 lbs.	8.0	H_2SO_4 2.5 lbs.	4.6	5.5	(2)	90.8	0.065
3	H_2SO_4 2.5 lbs.	5.5	Nil	5.5	5.5	(3)	90.9	0.050

It will be noted that with the small amount of sulphuric acid added to the grinding stage there is a pronounced reduction in tailings loss. It was at first thought this could be due to more acid soluble copper going into solution. However, later work did not substantiate the theory. It was a marked improvement in the floated copper minerals.

It will be noted in Test 3 that there was no change in pH from the beginning to the end of the acid conditioning stage. With ores containing more acid soluble constituents the pH in the acid addition stage could rise to 7.0 alkaline, depending on the amount of acid used and the natural pH of the pulp.

Test No.	Reagent to Grinding Stage & Test No.	Reagent Lbs./Ton	After Grind	pH Z6 Cycle	Rghr. % Wt.	+ Clnr. Tlg. % Cu Analysis
1	Lime (40)	8.75	11.6	11.6	92.0	0.099
2	Lime (44)	0.5	7.9	11.6	91.3	0.072
3	Soda Ash (42)	0.625	7.55	11.6	92.0	0.073
4	Potassium Carbonate (43)	0.625	8.4	11.6	91.4	0.081
5	Sodium Hydroxide (45)	0.5	8.0	11.55	91.1	0.081
6	Ammonium Hydroxide (46)	0.5	7.5	11.52	91.4	0.076

It will be observed from the foregoing results that the pH at the end of the grinding is significant and that the copper recovery was significantly better when the pH at the end of grinding was considerably lower than the optimum pH for flotation. Normally one would expect the opposite to be true.

It will be noted that in all of these tests prior to the rougher flotation the last stage of conditioning was with a dispersing agent, sodium silicate, in the presence of the sulfhydryl anionic collector, potassium amyl xanthate, for a conditioning period fo five minutes.

EXAMPLE VIII

A series of tests was run which included an acid conditioning stage preceeding the activation and collection of the copper minerals with the collector.

In each case, after the acid stage, the pH was increased with lime to give a pH of approximately 11.6 in the activation and collecting stage with xanthate.

LOT 130 - EXAMPLE IX

The third test in Example VIII using an acid stage in the grinding circuit with ten minutes conditioning following the grinidng stage at pH of 5.5 resulted in a rougher plus first cleaner tailing rejection of 90.9 percent by weight analyzing 0.050 percent Cu. In this Example, following the acid cycle, the pH was raised to approximately 11.6, with lime and the dispersing agent added at the same time followed by the xanthate activation cycle and then flotation. The tailings rejection was 91.25 percent by weight analyzing 0.062 percent Cu. or a small increase in tailings loss compared to the circuit which added the dispersant following the xanthate activation cycle.

EXAMPLE X

In this example the sodium silicate dispersing stage was introduced following the grinding stage. To the

grinding stage 0.5 lbs. per ton of lime was added resulting in an end pH of 7.9. The pulp was then transferred to the Fagergren cell and conditioned for one minute with 6.25 lbs. per ton of sodium silicate. The addition of the sodium silicate raised the pH of the pulp to 8.9. In the next stage 8.25 lbs. per ton of lime was added raising the pH to 11.9. In the next cycle 0.225 lbs. per ton of Z6 was added and the pulp conditioned for nineteen minutes with three drops of pine oil added for the last 3 minutes. The end pH of the pulp was 11.55. The flotation cycle was 7 minutes with the addition of 0.025 lbs. per ton of Z6 and one drop of pine oil. The first cleaner was the standard used in previous tests, that is, three minutes conditioning with 0.025 lbs. per ton of Z6 and 1.25 lbs. per ton of sodium silicate. The tailings rejection was 92.25 percent by weight analyzing 0.089 percent Cu. In comparing this test to Example II in series 7 wherein the only change was in the position of addition of the sodium silicate or dispersing cycle, it will be noted that the tailings in this test was 91.3 percent by weight analyzing 0.072 percent Cu. In other words, addition of the sodium silicate prior to the collector resulted in higher tailing loss.

EXAMPLE XI

This example is a duplicate of the third test in series 8 with the exception that following the conditioning stage at a pH of 5.5 the pH was raised with lime to 7.7 and the pulp the dispersed with sodium silicate prior to the raising of the pH to 11.6, followed by addition of the xanthate. The tailings rejection in this test was 91.3 percent by weight analyzing 0.065 percent Cu. as against the tailings loss in the comparative test of 0.050 percent Cu. again illustrating the effect of the use of a dispersing stage prior to the addition of the xanthate collector.

EXAMPLE XII

In this example the sodium silicate has been added to the grinding stage, otherwise the test is a duplicate of test 2 in Example VII. The tailings rejection was 92.6 percent by weight analyzing 0.096 percent Cu. an appreciable increase in tailings loss as compared to the silicate addition in the last cycle before flotation rather than in the first stage.

EXAMPLE XIII (TEST 58)

In this example the xanthate collector was added to the grinding stage but the test is otherwise a duplicate of test 2 of Example VII. The tailings rejection was 91.1 percent by weight analyzing 0.062 percent Cu showing an improvement in recovery over the addition of the collector in the second last cycle of the comparative test. In this example the contact time of the xanthate collector with the ore was a total of 36 minutes and this gave the lowest tailings loss of the various all alkaline circuits used.

Contrariwise, where the grinding stage was carried out in an acid circuit the addition of the collector to the grinding stage resulted in lower Cu recovery than in the case where the xanthate was added following conditioning in the acid stage and raising of the pH to the alkaline side.

EXAMPLE XIV (TEST 60)

This test is a duplicate of the second test in Example VII with the exception that the sodium silicate was replaced by tetra sodium pyrophosphate. In the test using sodium silicate 6.25 lbs. per ton of sodium silicate was used as the dispersant. In the case of the pyrophosphate 0.625 lbs. per ton was used or only 10 percent of the sodium silicate. The tailings rejection was 91.9 percent by weight analzying 0.073 percent Cu. It will be seen that the metallurgy is practically identical showing that the family of phosphates can be effectively used in place of silicates, it being a matter of economics which would be applied.

The silicates would seem to have one major advantage over the use of phosphates in plant practice. The surprising factor in the use of sodium silicate was that in the pH range of 10 to 12, as soon as the effective amount was reached there is a sudden drop in pH which can be put to use by automatic controls while with the pyrophosphate there was no change in the pH.

EXAMPLE XV (TEST 61)

This is a duplicate of Example XIV wherein sodium lignin sulphonate was substituted for sodium silicate or pyrophosphate in the dispersal cycle. The tailings rejection was 90.9 percent by weight analysing 0.075 percent Cu which tailing, since the lignin sulphonate was not optimized, can be considered equivalent to both the sodium silicate and pyrophosphate as dispersing agents. In this example 0.625 lbs. per ton of lignin sulphonate was used. The change in pH using the lignin sulphonate was a drop of 0.05. However, as the natural pH of the 5 percent lignin sulphonate solution used was 8.1 this lower pH can account for the change.

EXAMPLE XVI (TEST 64)

During the grinding stage of this example 3½ lbs. per ton of sulphuric acid was added resulting in an end pH of approximately 5.1. Following the grinding stage the pulp was conditioned for ten minutes with no further reagent addition. There was no change of pH during this cycle. At the end of ten minutes 2.0 lbs. of lime and 0.225 lbs. of Z6 per ton respectively were added to the pulp raising the pH to approximately 6.5. The pulp was conditioned for ten minutes and at the end of this period lime was added bringing the pH of the pulp up to 11.55 and the pulp was conditioned in this cycle for nineteen minutes. The end pH was 11.45. Following this cycle 12.25 lbs. per ton of sodium silicate was added together with three drops of pine oil for the last 2 minutes of the 5 minute conditioning cycle. When the sodium silicate addition was made the pH of the pulp immediately dropped from 11.45 to 11.15. The flotation stage was the same as that used in the previous examples. The tailings rejection was 89.4 percent by weight analyzing 0.08 percent total Cu and 0.013 percent acid soluble Cu. Samples of the total solution from the rougher and first cleaner tailings was analyzed for copper and contained 2.2 percent of the total copper (6.7 percent of the acid soluble copper). As the solid tailings contained 3.2 percent of the acid soluble copper the recovery of the acid soluble copper in the cleaner concentrate was 90.1 percent. It will be noted that the amount of collector added to the agitation conditioning stages prior to rougher flotation was 0.225 lbs. per ton ore, or approximately three times the amount used in the conventional circuit for this ore.

EXAMPLE XVII (TEST 65)

This example is a duplicate of the previous example with the exception that following the first acid conditioning stage the pH of the pulp was raised to 8.0 and it

was conditioned with the xanthate at this point for ten minutes. The tailings rejection was 90.5 percent by weight analyzing 0.051 percent total copper and 0.012 percent acid soluble copper, and in the solution tailings is about 3.0 percent for a total of 10.6 percent so that 89.4 percent of the acid soluble copper is in the cleaner concentrate product.

EXAMPLE XVIII

This was a duplicate test of Example XIII with the exception that following the grinding stage a ten minute conditioning cycle was inserted at a pH of approximately 7.7. The tailing rejection was 90.8 percent by weight analyzing 0.061 percent Cu which is practically duplicate metallurgy to the prior example without the use of the additional conditioning cycle. The soluble copper content of the rougher plus first cleaner tailings solution was 10 percent of the total acid soluble Cu. It will be noted that the acid soluble Cu in solution in this alkaline circuit was appreciably higher than in the prior two examples wherein the original stages were in an acid circuit. At this time this phenomenon was unexplained.

EXAMPLE XIX (TEST 63)

For this sample Lot **128** ore was used and a 1000 gram sample was ground for twelve minutes at 50 percent solids with 2 lbs. per ton of sulphuric acid. After 10 minutes of conditioning following grinding without the addition of reagents the pH was 4.5. The pH was raised to 11.6 with the addition of 5.65 lbs. per ton of lime and 0.225 lbs. of **Z6** were added for a 19 minute conditioning period at the end of which the pH was 11.45. 12.25 lbs. per ton of sodium silicate was added together with three drops of pine oil and the pH dropped to 11.10. A flotation cycle similar to that of Example XVI, but carried out at 41 percent solids, produced a combined rougher plus first cleaner tailings containing 88.4 percent by weight of the solids and analyzing only 0.042 percent of the total copper, which is most remarkable metallurgy for such a high pulp density.

As previously mentioned in the foregoing tests about 2½ percent of the total copper reported in the tailings solution. In order to filter the tailings it is helpful to flocculate them by making them acid with sulphuric acid. Surprisingly although the total time involved from the acidulation of the flotation tailings in the flotation cell to the completion of filtration was 2 minutes or less I have found that this copper actually goes into solution during that period. As the following tests will illustrate the tailings solution at the end of flotation in the alkaline circuit contains no dissolved copper at all.

By employing the tailings filtrate as make up solution for the grinding of fresh feed material in the acid circuit it is possible to recover most of this sulphuric acid soluble copper by flotation since when the product of the wet grinding circuit is made alkaline the dissolved copper precipitates and floats with the copper sulphides.

This phenomenon presents a simple and effective manner for rendering sulphuric acid soluble copper minerals amenable to xanthate flotation regardless of whether or not the ore contains clay or talc slimes rendering necessary the use of dispersants and flocculants in the manner already described.

It appears that for effective flotation of the combined acid soluble and sulphide copper minerals present in ores such as those to which reference has been made it is not necessary that all of the acid soluble copper minerals should be put into solution and precipitated. It is sufficient if only a small percentage of the acid soluble copper minerals go into solution and are precipitated. Indeed since with appropriate conditioning as already illustrated a very substantial proportion of these acid soluble minerals float in a completely alkaline circuit it is to be assumed that some dissolution-precipitation mechanism operates even at pHs well on the alkaline side of neutral. As will be apparent however from the examples already given and those to follow the floatability of the acid soluble copper minerals is maximized by the establishment of acid conditions in at least one stage of the total conditioning process. Further, the use of dispersing agents during such stage can be beneficial in ensuring that the effect of the treatment reaches the maximum proportion of particles in the pulp which are amenable to the treatment.

Example XX

Stage I

A 1000 gram sample of the Lot **128** ore previously described was ground in a laboratory rod mill for 12 minutes at a pulp density of 50 percent solids with the addition of 2.0 lbs. per ton of H_2SO_4. Following grinding the pulp was transferred to a laboratory Fagergren where it was conditioned for 10 minutes without any further reagent addition. The pH at the beginning and end of this conditioning period was 5.0.

Stage II

13.0 lbs. per ton of $Ca(OH)_2$ was added and conditioned for 5 minutes. The beginning pH was 12.20 and the final pH 12.25.

Stage III

2.2 lbs. per ton of H_8SO_4 and 0.18 lbs. per ton of **Z6** were then added and the pulp was conditioned for 15 minutes. The beginning pH of this stage was 11.65 while that at the end was 11.40.

Stage IV

12.0 lbs. per ton of sodium silicate were then added as a dispersing agent together with 4 drops of pine oil and the pulp was conditioned for a further 5 minutes. Throughout this stage the pH was 11.0.

The rougher concentrate was then floated during a period of 7 minutes with the addition of 0.02 lbs. per ton of **Z6** and one drop of pine oil. A small amount of sulphuric acid and "Separan" (a synthetic flocculant manufactured by Dow Chemical Co.) was added to the rougher tailings and the latter was filtered.

The rougher concentrate was resuspended in a Denver Cell and conditioned for 3 minutes with the addition of 0.02 lbs. per ton of **Z6** and 1.2 lbs. per ton of sodium silicate. The thus conditioned pulp was then floated to cleanup.

The tailings were treated in the same manner as the rougher tailings and in each case there was less than 2 minutes total contact time in the cell and the filter together.

The head sample of this ore was 0.95 percent total Cu and 0.23 percent acid soluble Cu.

% Weight	87.7
% Total Cu analysis of solids	0.040
% Acid Soluble Cu analysis of solids	0.019
% of total Cu in the solutions from	

-continued

the rougher plus first cleaner tailings	3.0
% of original acid soluble copper in the solutions	12.5
% of total Cu in the first cleaner concentrate	93.3

For this type of ore the recovery of 93.3 percent of the total copper in the first cleaner concentrate is an amazingly high recovery considering that only 3.7 percent of the total copper was in the solids tailings.

It will also be noted that even with one of the conditioning stages at a pH in excess of 12.0, it had no harmful effect on the final flotation results.

This may be particularly important where molybdenum sulphide minerals may occur in the same deposits as the copper minerals, producing relatively high recovery of the molybdenum.

Examples XXI, XXII, and XXIII

This series of tests used the same ore and circuitry as Example XX, to determine the effect of various alkali agents in combination with $Ca(OH)_2$, and also omitting the acid addition in Stage III.

Stage II in all tests was taken as closely as possible to the same pH, that is, 11.65.

The following table shows the comparative results and the alkali agents used in Stage II.

Example No.	Alkali Agents lbs./ton CaO	Na_2CO_3	NH_4OH	HaOH	Rougher + 1st Cleaner Tailings Analysis Solids % Wt.	% Total Cu	Total A.S.*Cu	Sol'n % of Cu
XXI	8.2	4.0			88.0	0.063	0.028	2.0
XXII	8.6		2.0		87.6	0.055	0.018	4.3
XXIII	6.4			2.0	87.7	0.047	0.022	2.9

*A.S. — Acid Soluble Cu

It will be noted that all combinations are effective with the sodium hydroxide being outstanding.

EXAMPLE XXIV

In this example the same ore and circuitry was used as in Example XX. 3.2 lbs. per ton of H_2SO_4 was added to the grinding stage and 11.0 lbs. per ton of $Ca(OH)_2$ to Stage II.

The rougher and first cleaner tailings were settled with a flocculant only, no sulphuric acid being used. The solutions at all stages were sampled and analyzed for copper.

The following table shows the results obtained:

Stage	Description	pH Begin	pH End	Copper in Solution Mgs./L	Copper in Solution Total Mgs
				Begin	
I	Grinding and conditioning	5.4	5.1	1.8	2.7
				End	
				3.1	4.7
II	Lime addition	12.2	12.2	Nil	Nil
III	Collector addition	12.2	11.6	Nil	Nil
IV	Dispersant addition	11.25	11.2	Nil	Nil
End of Rougher Flotation				Nil	Nil
End of First Cleaner Float				Nil	Nil

The analysis of the rougher plus first cleaner tailings was 0.08 percent total Cu, and 0.057 percent acid soluble Cu.

The combined tailings weight was 90.5 percent.

The recovery of the acid soluble copper in the first cleaner concentrate was 77.6 percent.

The surprisingly low amount of copper in solution in Stage I followed by the high percentage of acid soluble copper retained in the first cleaner concentrate were the outstanding features of this test, together with total precipitation of the copper in Stage II that was in solution in Stage I.

Examples XXV, XXVI and XXVII

In this series of tests, the same ore and circuitry was used as in Example XX. In the test prior to Example XXV, 10 lbs. per ton of H_2SO_4 was added to the rougher tailings and 2.8 lbs. per ton of H_2SO_4 was added to the first cleaner tailings. Following filtering of these tailings, 1000 cc's of the solution was recovered and added to the grinding stage of Example XXV. The same procedure was followed with Examples XXVI and XXVII, with the combined tailings solution from Example XXV used in the grinding stage of Example XXVI, and the combined tailings solution of Example XXVI used in the grinding stage of Example XXVII.

The objects of these tests was to determine whether or not the copper dissolved in the tailings would precipitate and float with the solids copper in the flotation circuit.

The solutions were sampled in all of the stages of the three continuous tests.

The following were the results obtained:

Stage No.	Average pH Example No. XXV	XXVI	XXVII	Solution Cu. Mgs./L Example No. XXV	XXVI	XXVII	Solids Tailings Analysis % T Cu XXV	XXVI	XXVII	% A.S. Cu XXV	XXVI	XXVII
I	5.3	5.3	5.2	8.9	7.8	5.7						
II	11.8	11.8	11.8	0.0	0.0	1.2						
III	11.7	11.65	11.65	0.0	0.0	0.0						
IV	11.3	11.3	11.25	0.0	0.0	0.0						
Rougher Plus												

-continued

Stage No.	Average pH Example No. XXV	XXVI	XXVII	Solution Cu. Mgs./L Example No. XXV	XXVI	XXVII	Solids Tailings Analysis % T Cu XXV	XXVI	XXVII	% A.S. Cu XXV	XXVI	XXVII
1st Cleaner Tailings				144	135	140	.060	.057	.051	.026	.021	.01

The surprising phenomenon in the series of tests was the large precipitation of the copper dissolved in the tailings at a pH of 5.0 to 5.5 when re-circulated to the grinding circuit and transferred to the flotation cell.

The second surprising factor was the flotation of this copper following its precipitation with approximately 93 percent of the total acid soluble copper reporting in the first cleaner concentrate.

The third surprising factor was that with but an indicated 20 percent of the total acid soluble copper circulated to the grinding circuit from the tailings, approximately 93 percent of the total acid soluble copper reports in the first cleaner concentrate.

Further testing was carried out with pHs as low as 2.0 in Stage I, but gave no indication of improved results over the previous described circuit, although from a plant operational point of view, it may, in some instances, be more desirable to add the acid to the grinding circuit only, or to the first conditioning step following the grinding circuit.

EXAMPLE XXVIII

The same ore and circuitry as in the previous examples was employed using an all alkaline circuit.

The solutions in each stage. were sampled and analyzed for copper. Only traces of copper in one test showed up, and this was a neutral pH.

The ore sample used contained 1.09 percent Total Cu and 0.36 percent Acid Soluble Cu.

The typical metallurgical results were as follows:

% Wt. Rougher Plus First Cleaner Tailings	91.6
% Total Cu	0.11
% Acid Soluble Cu	0.067
% Recovery Sulphide Cu In 1st. Clnr. Conct.	94.6
% Recovery Acid Soluble Cu in 1st. Clnr. Conct.	83.0

The high recovery of the acid soluble copper in this circuit is outstanding, particularly as the only collector used was of the xanthate family, that is, Potassium Amyl Xanthate, which, in a current conventional circuit, has but little collecting effect on the family of acid soluble copper minerals, unless such modifiers as sulphidizing agents are used.

Two of the differences in preferred embodiments of my circuits over current conventional practice are a prolonged conditioning period prior to flotation, and a concentrated use of dispersants in the latter stages of my conditioning circuit with vigorous agitation.

By contrast, in current conventional practice in the flotation of copper ores the pulp goes directly from the grinding circuit to the flotation cells (being adjusted to optimum pH for flotation either on the way there or after arrival) and floated over a prolonged period of time with stage additions of the collector. In the process of the invention even where the grinding stage is used as stage one it is necessary to use a minimum conditioning cycle following the grinding stage of 10 to 15 minutes for optimum results. On ores with minimal slime interference and no dispersant required, I use a minimum of two vigorous to violent agitation conditioning steps where the calculated residence time of the pulp is a minimum of about 4.0 minutes.

EXAMPLE XXIX - Ore Lot **130** as previously described.

In this sample sulphur dioxide has been used in Stage I to replace sulphuric acid, with approximately 60 percent of the SO_2 added to the grinding circuit and 40 percent to the conditioning cycle following grinding.

In Stage II the alkaline agent ($Ca(OH)_2$ and the collector **Z6** were added simultaneously.

In Stage III the dispersing agent, Na_2SiO_3, was added together with the frother, followed by the rougher float.

The first cleaner float was preceeded by a 3 minute conditioning cycle using 0.75 lbs. of Na_2SiO_3 per ton, and 0.025 lbs. of **Z6** per ton.

The cleaner was insufficient to depress a satisfactory percentage of the remaining slimes in the rougher concentrate and a second stage cleaning was required using a 2 minute preconditioning period with 0.5 lbs. per ton of Na_2SiO_3.

This illustration may be considered as a three-stage preconditioning circuit to the rougher float, and an additional two stage dispersion circuit to the number **2** cleaner concentrate, to obtain satisfactory copper recovery in a concentrate containing an acceptably low clay content.

The following is the data on this example:

Stage I - To grinding circuit - 8.1 lbs./ton of SO_2-12 minutes; To conditioning - 4.7 lbs./ton of SO_2-10 minutes; pH end grinding - 5.9 pH begin and end conditioning - 4.8 and 5.4 respectively.

Stage II - **20** minutes conditioning with addition of 22 lbs./ton of $Ca(OH)_2$ and 0.225 lbs./ton **Z6**. pH begin and end conditioning - 11.7 and 11.5 respectively.

Stage III - 4 minutes conditioning with addition of 12.5 lbs./ton of Na_2SiO_3, and two drops of pine oil. Rougher Flotation.

The following results were obtained:

Rougher plus 1st Cleaner Tailings:

% Weight	87.5
% Total Cu Analysis	0.068
% Acid Soluble Cu analysis	0.033
% Total Cu in 1st Cleaner concentrate	94.5%
% Acid Soluble Cu in 1st Cleaner concentrate	89.5%
% Soluble Cu in solutions	Nil

Rougher plus 1st plus 2nd Cleaner Tailings:

% Weight	92.5
% Total Cu in second cleaner concentrate	93.7

This test illustrated the following:

a. The satisfactory substitution of SO_2 for H_2SO_4 in my process.

b. The simultaneous addition of the alkaline agent and the collector following the use of three-stage circuit prior to rougher flotation

c. The use of three-stage circuit prior to rougher flotation and the necessity of two stages of dispersion following the rougher flotation.

d. The complete precipitation of any acid soluble copper that had gone into solution in the acid cycle.

e. The high degree of flotability of the precipitated copper in conjunction with the sulphides.

EXAMPLE XXX - Ore Lot **130** as previously described.

This example illustrates the use of a three-stage conditioning circuit prior to rougher flotation, using SO_2 in place of H_2SO_4.

The following is a description and data on the circuit used: Grinding Circuit - 12 minutes - No reagents added pH at end - 7.2

Stage I - 10 minutes - Conditioning with additon of 12.8 lbs./ton SO_2 and 0.625 lbs./ton of dispersant, lignin sulphonate. pH begin - 4.0, and end - 4.1.

Stage II - As in Example XXIX

Stage III - As in Example XXIX Rougher float. Two cleaners as in Example XXIX.

At the end of Stage I the copper in solution was less than 5 percent of the total acid soluble copper and was totally precipitated in Stages II and III.

The metallurgy was practically identical to Example XXIX.

This test illustrated the following.

a. Using an acid circuit in the grinding stage would be eliminated, unless it was desirable from an operating point of view.

b. A dispersant could be used in the acid cycle where necessary without detrimental effects to the copper recovery.

EXAMPLE XXXI - Ore Lot **130** as previously described.

This example illustrates the use of a two-stage conditioning system wherein a single acid stage is used and a single alkaline stage is used, prior to rougher flotation.

Two cleaners were used with dispersal of the pulp between each cleaner.

Grinding Circuit - 12 minutes, no reagents added. pH at end - 7.2

Stage I - 20 minutes conditioning, added 12.5 lbs./ton H_2SO_4 and 0.625 lbs./ton dispersant, tetra sodium pyrophosphate. pH begin 1.8, end 3.6.

Stage II - 20 minutes conditioning, added 22.7 lbs./ton $Ca(OH)_2$, 0.25 lbs./ton Na_2SiO_3, and 2 drops of pine oil, at end prior to rougher flotation. pH begin 11.9, end 11.4.

Rougher flotation followed by two stages of cleaning as in Examples XXIX and XXX.

The rougher plus first cleaner tailings was 86.4 percent by weight analyzing 0.057 percent total Cu and 0.023 percent acid soluble Cu for a recovery in the first cleaner concentrate of 95.5 percent of the total copper and 94.5 percent of the acid soluble copper. There was no soluble copper in the tailings solutions.

This can be considered outstanding metallurgy for such a type of ore, particularly the flotation of the acid soluble copper.

In applying an acid conditioning stage to ores containing a large percentage of acid consuming constituents such as carbonates, it has, in some cases, been found necessary to use quantities of sulphuric acid exceeding thirty pounds per ton of ore and to lower the initial pH to as low as about 1.2.

EXAMPLE XXXII (Test **66**)

For this sample Lot **130** ore was used and a 800 gram sample was ground at 50 percent solids with 0.5 lbs. lime per ton and 0.15 lbs. **Z6** per ton. Following the grinding circuit the pulp was conditioned at approximately 42 percent solids for ten minutes without any further reagent addition. At the end of the ten minutes agitation conditioning the pH of the pulp was raised to 11.7 with lime and the pulp agitation conditioned for nineteen minutes at approximately 30 percent solids. At the end of this conditioning period. 6.25 lbs. sodium silicate per ton was added to the pulp, together with three drops of pine oil, and agitation conditioned for five minutes wherein the end pH prior to flotation was 11.3.

A rougher flotation was made using seven minutes flotation time together with the addition of 0.02 lbs. per ton **Z6** and one drop of pine oil. The rougher concentrate was agitation conditioned in the 250 gram Denver cell with the addition of 0.025 lbs. **Z6** and 0.7 lbs. per ton of sodium silicate. The rougher plus first cleaner tailing analyzed 0.061 percent total copper and the first cleaner concentrate analyzed 12.1 percent copper containing 91.1 percent of the total copper in the feed. This test shows the effect of adding the collector to the grinding circuit at a lower pH than the optimum pH for flotation. Two stages of agitation conditioning with the dispersant sodium silicate, and in the presence of the sulfhydryl anionic collector were used to produce excellent rejection of the deleterious host rock minerals together with high recovery of the copper minerals.

EXAMPLE XXXIII

The ore tested in this programme was the feed to a concentrator in British Columbia . This plant is considered one of the finest metallurgical operations in the treatment of copper ores.

The copper minerals in the ore are chiefly chalcopyrite with minor amounts of chrysocolla.

The ore contained a small amount of slimes, and approximately 1.0 percent pyrite.

The sample of ore tested was from a large tonnage sample taken of the mill feed, and the mill metallurgy for the period recorded. A pilot plant was set up to duplicate the plant and one of the pilot plant runs is the relative zero recorded in the comparative work.

The flow sheet of this circuit is shown at FIG. **7.**

The results were a very close check on the plant metallurgy.

The collector used was Aerofloat A-**242,** supplied by American Cyanamid Co. This collector is claimed to be much more selective to the copper minerals (wherein the ore contains pyrite), than potassium amyl xanthate, and this should have produced a higher grade concentrate than the xanthate used in may circuit.

The second pilot plant test in the series used a similar collector to Aerofloat A-**242,** Aerofloat **238.** In this pilot plant run the rougher and scavenger float was conventional with the reagents mixed with the pulp for one minute prior to rougher flotation.

However, the cleaner circuit treating the rougher concentrate was not conventional. It used one of the embodiments of my invention wherein the rougher concentrate was conditioned with added collector and approximately 0.3 lbs. sodium silicate per ton of original feed.

The pH was approximately 11.6.

The second cleaner concetrate was fed to a regrind mill with the addition of a small amount of collector.

Following the regrind mill the pH was raised to 12.4 with lime and the regrind mill product agitation conditioned in two agitation conditioning steps for 10 minutes prior to the third cleaner float.

The third cleaner concentrate was re-floated without conditioning to form the fourth and final cleaner concentrate.

The cleaner tailings were re-circulated counter current.

The flowsheet of this circuit is shown at FIG. **8.**

The third pilot plant run was duplicate of the second with the exception that the collector, Aerofloat **238,** was replaced by potassium amyl xanthate.

The fourth pilot plant run used potassium amyl xanthate and an agitation conditioning circuit described in my co-pending application 270,961 and 37,679 now abandoned, in which the pulp from the grinding circuit was agitation conditioned for 12 minutes with the addition of 6.5 lbs. sulphuric acid per ton of solids at 37 percent solids, followed by a second stage of agitation conditioning for 6 minutes at a pH of 10.0 with lime and a pulp density of 31 percent solids; 0.14 lbs. of **Z6** per ton was added to this stage.

In the final and last stage before flotation, the pulp was agitation conditioned for eight minutes at a pH of 11.6 with lime 0.09 lbs. per ton of **Z6** and the frother, Dowfroth 250, a product of the Dow Chemical Company.

Following rougher and scavenger flotation, the cleaner circuit was identical to the other three pilot plant runs.

The flow sheet of this circuit is shown at FIG. **9.**

The original work for the fourth pilot plant run was carried out in the laboratory using a Denver 1000 grm cell for the rougher float, and 500 and 250 gram cells for the cleaner floats.

In comparing the Weston Laboratory Standard to the Weston Pilot Plant Circuit, in the laboratory standard one minute's agitation conditioning was used ahead of the first cleaner with the addition of collector and dispersing agent, while in the pilot plant the conditioning only took place through a pump and pump line.

The second difference was that in the Pilot Plant the agitation conditioning stage between the third and fourth cleaner was omitted.

These two factors would account for the lower grade concentrate for the Pilot Plant run.

However, in both cases the grades of concentrate produced and the recoveries are appreciably better than the conventional flow sheet.

The increase in copper value is in excess of 1.0 lbs. of copper per ton of ore at a higher grade concentrate, even in using the non-selective collector, potassium amyl xanthate.

EXAMPLES XXXIV AND XXXV

These two examples are in using one of the teachings of my invention in the cleaner circuit only and its application is in one of the most simplified forms. The ore treated was a copper ore from a producing plant in Arizona in which the major copper minerals were chalcocite and chalcopyrite. The ore contained minor amounts of sulphuric acid soluble copper, and pyrite. Deleterious slime interference was at a minimum.

In Example XXXIV, agitation condition of the pulp was carried out according to my teachings in my pending application Ser. No. 270,961. Following the grinding circuit the pulp was agitation conditioned for 15 minutes with the addition of 6 lbs. per ton of sulphuric acid in which the pH of the pulp at the end of the conditioning stage was 4.7, and the pulp density approximately 42 percent solids. At the end of this conditioning stage the pulp density was lowered to approximately 25 percent solids, and the pH raised with lime to 10.5, 0.13 lbs. per ton of potassium amyl xanthate was added to the pulp and it was agitation conditioned for a period of 6 minutes. At the end of this period the pH was raised with lime to 11.2 with the addition of 0.14 lbs. of **Z6** together with a conventional frother, and conditioned for nine minutes. At the end of this period a rougher float was made producing a rougher concentrate and final rougher tailing.

For the complete conditioning and rougher float a 1000 gram Denver laboratory cell was used at a rotor speed of 2100 rpm. The rougher concentrate was transferred to a 250 gram Denver cell and agitation conditioned at a rotor speed of 1350 rpm for two minutes, with the addition of 0.02 lbs. per ton **Z6** and 0.3 lbs. per ton sodium silicate.

A first cleaner concentrate and first cleaner tailings were produced. The rougher tailings was 96.13 percent by weight and analyzed 0.050 percent total copper. The first cleaner tailing was 2.05 percent by weight and analyzed 0.26 percent copper. The first cleaner concentrate was 1.82 percent by weight and analyzed 33.01 percent copper and contained 91.77 percent of the total copper in the feed. The head value of the ore was 0.65 percent Cu.

The production of a high grade concentrate, that is, 33 percent copper in a single stage of cleaning was quite amazing. It was found on this ore that in practicing my teachings of at least one agitation conditioning step on at least one flotation concentrate what seems to be additional activation of the copper minerals in this activation conditioning stage on this ore was at an optimum pH point of approximately 11.0. The pH below 10.5 showed much poorer results and pHs above 11.5 showed lower grade concentrates.

The ore in the following Example XXXV was from the same orebody as in Example XXXIV. The preparation of this ore for rougher flotation was by conventional means. Lime was added to the grinding unit which, in this case, was a laboratory rod mill and at the end of the grinding stage the pH was 10.5. The ore charge was 1000 grams and following the grinding stage three drops of a collector designated Minerec 898 and supplied by Minerec Corp. was added to the pulp along with a conventional frother and conditioned for two minutes prior to rougher flotation.

During rougher flotation 0.02 lbs. of potassium amyl xanthate was added to the pulp and a total float time of ten minutes was used. A single cleaner was used. The rougher concentrate was agitation conditioned in a duplicate manner to Example XXXIV. The rougher tailing was 96.06 percent by weight and analyzed 0.065 percent copper. The first cleaner tailing was 2.02 percent by weight and analyzed 0.10 percent copper. The cleaner concentrate was 1.92 percent by weight and analyzed 24.26 percent copper.

There were two surprising features of these results. The first was the low analysis of the first cleaner tailing, that is, 0.10 percent copper, with the grade in the first cleaner concentrate analyzing 24.26 percent copper from an original head value of 0.53 percent copper. In

comparing this test to Example XXXIV where the original copper values were heavily activated prior to the rougher flotation, whereas in this conventional float the mineral would be only lightly activated, the conditions of the two minute agitation conditioning stage were such that the copper minerals could only have been more heavily activated to produce a concentrate grade of approximately 24 percent Cu in a single cleaner with a cleaner tailing analyzing the extraordinarily low figure of 0.10 percent Cu.

EXAMPLE XXXVI

The ore used in this programme was from the same orebody as in Examples XXXIII and XXXIV. It contained higher sulphuric acid soluble copper values than in the other two examples. The treatment of the ore to produce a rougher float was the same, that is, the ore was ground with lime to produce a pH in the final grinding circuit pulp of 10.4 Following the grinding stage the pulp was transferred to a 1000 gram Denver flotation cell. Three drops of Minerec 898 were added to the cell, together with conventional frother and conditioned for 2 minutes. The rougher float time was ten minutes with 0.02 lbs. Z6 added after 5 minutes flotation time. The rougher concentrate was transferred to a 250 gram Denver flotation bowl and the machine operated at 1350 rpm. The rougher concentrate was conditioned in this bowl for 2 minutes with the addition of 0.03 lbs. per ton Z6 and 0.15 lbs. per ton sodium silicate. The first cleaner float was 5 minutes. The first cleaner concentrate was transferred to the same 250 gram flotation cell and conditioned for 2 minutes with the addition of 0.03 lbs. per ton Z6 and 0.15 lbs. per ton sodium silicate.

The second cleaner float was floated for a period of 4 minutes. The rougher tailings were 96.64 percent weight and analyzed 0.123 percent copper. The first cleaner tailings were 1.75 percent by weight and analyzed 0.75 percent copper. The second cleaner tailing was 0.3 percent by weight and analyzed 1.43 percent copper. The second cleaner concentrate was 1.31 percent by weight and analyzed 40.8 percent copper.

Again, it will be noted that with this ore containing comparatively high oxide copper values, not only was it possible to produce a high grade concentrate in two cleaners, but also additional activation of the copper minerals had taken place wherein the open circuit approximately 80 percent of the total copper values reported in the concentrate at a grade of 40.8 percent.

Where pyrite occurs in such ores, before regrinding, my preferred circuit is to use two or more such cleaners ahead of the regrind circuit, eliminating a very high percentage of any slime material that may be present. For effective depression of the pyrite following the regrind circuit, I have found that the elimination of these slimes is an integral part of the process in effectively depressing the pyrite in the subsequent treatment steps.

EXAMPLE XXXVII

The following programme was carried out on a copper ore from British Columbia. Deleterious slimes were present in this sample. The copper minerals contained in the ore were chalocite, chalcopyrite, and chrysocolla. The ore also contained pyrite.

In this test 1500 gram charges were used, ground in a laboratory rod mill with specific reagents, and a rougher concentrate produced.

The second test was run in the same manner, varying one of the reagents only, and the rougher concentrate from this test combined with the first rougher concentrate for cleaning and further treatment.

A 1000 gram Denver flotation cell was used with a rotor speed of 2100 rpm for the conditioning and flotation of the rougher concentrate.

In the first test 0.6 grams of lime was added to the mill together with 0.05 lbs. per ton of Z200 (a sulfhydryl anionic collector sold by the Dow Chemical Co.), and one pound per ton of sodium silicate. Following the grinding circuit the pulp was transferred to the Denver cell and conditioned for 0.5 minutes at a pH of 10.1 in the presence of a conventional frother. The rougher float time was 10 minutes and the rougher tailing analyzed 0.171 percent copper.

The second test in which the rougher concentrate was produced was identical to the previous test with the exception that the sodium silicate to the grinding unit was reduced to 0.33 lbs. per ton of ore. The rougher tailings analyzed 0.204 percent copper. The two rougher concentrates were combined and placed in a 500 gram Denver cell bowl which was operated at 1600 rpm and the pH was 10.5. Added to the cell was 0.003 lbs. per ton of Z200 and 0.33 lbs. per ton of sodium silicate. The pulp was agitation conditioned at a rotor speed of 1600 rpm for 1½ minutes and a first cleaner concentrate produced. The first cleaner concentrate was placed in a regrind mill, 0.25 grams of lime was added together with 0.04 lbs. per ton of Z200.

Following the regrind the pulp was transferred from the regrind mill to a 250 Denver cell which was operated at 1350 rpm. To the bowl was added 0.1 lbs. per ton of sodium silicate and the pH raised to 12.5 with lime. The agitation conditioning was carried on for ten minutes prior to the second cleaner float. The second cleaner concentrate was transferred to the 250 gram bowl, operated again at 1350 rpm, and the make-up solution was a solution from No. 2 Cleaner Tailings. No further reagents were added to the pulp, and there was no conditioning. The No. 3 Cleaner Concentrate was floated to clean up and produced a concentrate analyzing 50.1 percent copper.

This was a surprising high grade of concentrate to be produced in three cleaners from an ore that contained both deleterious slimes and pyrite.

EXAMPLE XXXVIII

The following series of tests were run on a nickel ore with a head value of 0.55 percent nickel and 0.50 percent Cu. The nickel occurred as pentlandite and the copper as chalcopyrite.

In each test the ore was ground to approximately 95 percent minus 100 mesh in a laboratory rod mill at 50 percent solids.

The ground ore was transferred from the rod mill to a laboratory Fagergren flotation cell, vigorously conditioned with no air to the cell, and finally floated in the same cell.

Each ore charge was 800 grams, and the conditioning and flotation was carried out at 33 percent solids and 30 percent solids respectively.

In all tests the reagents added were in the following form - CaO and Na_2CO_3 as grams or to a pH; $CuSO_4$ - 5 percent solution; SO_2 - saturated solution; Z6 - 1 percent solution; Lignin and Sodium Silicate - 10 percent solution; Frother - Pine Oil, and M.I.B.C./65 - drops.

The conditons of the test were as follows:

Test No. 1								
	Time of	Reagents Added and Place of Addition						
Units	Condit. Mins.	CaO	Na_2CO_3	$CuSO_4$ cc's	SO_2 cc's Set	Z6 cc's	Pine Oil	pH
Rod Mill No. 1		0.5	0.5	3.0				10.0
Agit. Condit. No. 2	10							9.95
Agit. Condit. Float to Clean-up	20					6.0	3	9.90
Test No. 2								
Rod Mill No. 1			0.75	6.0				9.7
Agit. Condit. No. 2	10							9.7
Agit. Condit. Float to Clean-up	20				4.0	6.0	3	8.5
Test No. 3								
Rod Mill No. 1			3.0	4.5				10.2
Agit. Condit. No. 2	10							10.1
Agit. Condit. Float to Clean-up	20					6.0	3	10.0

The rougher tailings from the three tests analyzed as follows:

Test No.	Rougher Tailings % Wt.	% Ni.	% Cu.	Calculated % Recovery Ni.	Cu.
1	80.4	0.043	0.014	93.7	97.8
2	80.5	0.046	0.020	93.3	96.8
3	80.0	0.038	0.014	94.5	97.8

In using 30 minutes vigorous agitation conditioning in the conditioning circuit, outstanding results were obtained in all of the tests. No dispersant was used.

In plant practice the power consumption of the agitation conditioning circuit in all three tests would be approximately 1.5 kilowatthours per ton of ore fed to the circuit. In treating nickel bearing sulphide ores in my agitation conditioning circuit located between the grinding circuit and primary flotation the estimated maximum power consumption of the total agitation mechanisms used is about 6.0 kilowatt hours per ton of ore treated.

In Test No. 2, sulphur dioxide as sulphurous acid and free sulphur dioxide was used as the acid agent to lower the pH. On first addition the pH dropped to 8.5 and increased to 9.0 by the end of the conditioning stage. Sulphuric acid may also be used as an acid agent to lower the pH in an acid or alkaline conditioning stage preceeding the final alkaline conditioning stage in the pH range of 8.0 to about 11.5.

I have found that in addition to adding part or all of the alkali agent to the grinding circuit, the activator for the pentlandite, copper sulphate, and the collector to the grinding circuit, that more collector is usually necessary for optimum results. For this reason I prefer to add all, or at least part of the collector to the agitation conditioning circuit.

In plant practice I prefer to use a minimum of three vigorous agitation conditioning steps following the grinding circuit. With slow speed agitation conditioning there is a marked drop off in nickel recovery.

The minimum residence time of the pulp in the agitation conditioning circuit for good recovery of the nickel minerals is approximately 8 minutes.

In the preferred embodiment of my invention the alkali agent is a combination of lime or calcium hydroxide with soda ash (sodium carbonate) or sodium carbonate alone.

EXAMPLE XXXIX

The following series of tests were run on the same ore as Example XXXVIII.

The same ore charge, conditions of grind, and percent solids to conditioning were used. The major differences were in the type of agitation, or aeration period, and the use of dispersant. On the Fagergren laboratory cell used, the lower end of the vigorous agitation conditioning was 1800 rpm, and the upper, 2100 rpm. The lower end of violent agitation conditions wherein splashing of the pulp against the sides of the bowl and major entrenchment of air was 2150 rpm, and the upper limit was 2500 rpm wherein the agitator was so violent that splashing of the pulp could not be contained within the bowl.

At 1800 to 2100 rpm there was fast circulation of the pulp with no splashing and only minor entrainment of air in the pulp. At 2100 rpm there was fast circulation of the pulp with the beginning of splashing against the sides of the bowl and some entrainment of air in the pulp.

The following were the conditions of the test:

Test No. 1 Units and R.P.M.	Time of Condit. Mins.	Reagents Added and Place of Addition CaO	Na_2CO_3	$CuSO_4$ cc's	Z6 cc's	Lignin cc's	Pine Oil	pH
Rod Mill			0.5	6.0				9.35
No. 1 Agit. Condit. 2000 rpm	10 Air off							
No. 2 Agit. Condt. 2500 rpm	25 Air on							8.85
No. 3 Agit. Condit. 2000 rpm	3 Air off				4.0	2.0	2.0	
Float to Clean-up								8.85
Test No. 2								
Rod Mill		0.75		6.0				9.35
No. 1 Agit. Condit. 2000 rpm	10 Air off							9.7
No. 2 Agit. Condit. 2000 rpm	60 Air on							8.7 End
No. 3 Agit. Condit. 2300 rpm	15 Air off				4.0			
No. 4 Agit. Condit. 1800 rpm.	5 Air off					2.0	2.0	8.7
Float to Clean-up								8.7

The rougher tailings from the two tests analyzed as follows:

Test No.	Rougher Tailings % Wt.	% Ni.	% Cu.	Calculated % Recovery Ni.	Cu.
1	83.0	0.064	0.027	90.3	95.5
2	83.2	0.067	0.028	89.9	95.3

The dispersant used in these tests was a lignin supplied by the American Cyanamid Company, and designated as the 600 Series.

It will be noted that in comparing the tailings loss to the previous example the violent agitation conditioning period with aeration resulted in markedly higher tailings losses in both nickel and copper values, although the recoveries are reasonably good.

EXAMPLE XXXX

This series of tests was run on a copper ore from two properties of a major producer in Arizona. The ores contained both sulphide copper minerals and acid soluble copper minerals. The latter minerals were mainly chrysocolla and malachite.

In each case the ore was ground to approximately 75 percent minus 100 mesh in a laboratory rod mill at 50 percent solids. The ground ore was transferred from the rod mill to a laboratory Denver flotation cell, vigorously conditioned with no air to the cell, and finally floated in the same cell.

Each ore charge was 1000 grams. The conditioning was carried out at approximately 40 percent solids, and the flotation at 25 percent solids.

The conditions of the tests were as follows:

Test No. 1 Units	Time of Condit. Mins.	Reagents Added CaO	Na_2CO_3	Collector cc's - 1% Sol'n.		MIBC 65 Drops	NaHS cc's - 1% Sol'n.	pH
Rod Mill		2.5		404	3			
No. 1 Agit. Condit.	5			Z6	3			10.2
No. 2 Agit. Condit.	3					3	4.0	
Test No. 2								
Rod Mill								
No. 1 Agit. Condit.	20		To pH of 9.9	Z6	12	6		End pH 9.4
No. 2 Agit. Condit.	0							
Test No. 3								
Rod Mill		2.5		404	−1.5			
No. 1 Agit. Condit.	5			Z6	2.5	6		10.5
No. 2 Agit. Condit.	0							

The rougher tailings from the three tests analyzed as follows. The heads were approximately 0.78 percent Total Cu.

Test No.	Rougher Tailings % Wt.	% Total Cu.	% A.S. Cu.	% Recovery Total Cu.
1	91.8	0.034	0.016	96.1
2	93.2	0.026	Not Analyzed	96.9
3	93.6	0.038	0.010	95.3

The collector **404** is a sulfhydryl anionic collector supplied by the American Cyanamid Company. The frother M.I.B.C./**65** is a mixture of methyl isobutyl carbinol and the **65** an alcohol frother supplied by the American Cyanamid Company. The mixture ratio was **65%** M.I.B.C. to 35 percent alcohol frother **65.**

The agitation conditioning in all tests was vigorous, with the air cut off the cell. The copper recovery from all tests was excellent with the longest vigorous agitating conditioning period, that is 20 minutes in Test No. **2,** showing the highest copper recovery.

The sodium hydrogen sulphide, used in Test No. **1** as a sulphidizing agent may be used in both an acid and alkaline conditioning stage. Other sulphidizing agents, such as calcium sulphide, sodium sulphide and hydrogen sulphide may also be used to advantage on some types of acid soluble copper minerals that may show little response to activation by no other means. This is a surprising feature of the invention, as in previous conventional practice both high speed agitation conditioning and prolonged period of sulphidizing have been carefully avoided, with the sulphidizing agent stage added in the flotation circuit.

In the preferred embodiment of my agitation conditioning circuits for copper ores containing sulphide copper minerals, there are two preferred circuits. The first uses a minimum of two vigorous agitation conditioning steps. The second uses a minimum of one violent agitation conditioning step with at least two vigorous agitation conditioning steps.

The simplest such circuit is a single violent agitation conditioning step followed by a single vigorous agitation conditioning step.

The pH of the pulp in the last agitation conditioning step prior to flotation for copper ores, nickel ores, molybdenum ores and lead ores, is in the range of about 8.0 to 12.0. This final step may be preceeded by one or more agitation conditioning stages in the pH ranges of about 1.2 to 8.0, 12.0 to about 12.3, and 8.0 to about 10.5. If the latter stage is used in at least a two-stage all-alkaline conditioning circuit the preferred stages are in the pH ranges of about 8.0 to about 10.5, and about 10.5 to 12.0.

EXAMPLE XXXXI

The following series of tests were carried out on a molybdenum ore from British Columbia. The molybdenum reportedly occurred as molybdenite, the molybdenum sulphide, and also some degree of oxidation of the molybdenite.

The first ore contained talc and the operator of the property, one of the largest Canadian mining companies, reported that using all available technology, they were unable to produce either reasonable recovery or a concentrate that was of sufficiently high grade to be saleable. With the inventor's process good recoveries were obtained together with concentrate grades that were higher than that required to meet their market specifications.

The second ore from the same deposit contained only minor amounts of detrimental slimes. Both recovery and final concentrate grade, using conventional processes, was poor.

With various preferred embodiments of my invention tailings losses were greatly reduced and the final concentrate grades produced were outstanding.

On both types of ore final concentrate grades were produced analyzing 56.0 to 58.5 percent MoS_2. The normal market specification is 55.0 percent MoS_2.

The following series of tests were run. In each test 1500 gram charges of the ore were ground in a laboratory ball mill at 50 percent solids, and transferred to a Denver laboratory cell, conditioned in the same cell, and floated in the rougher float for 6 minutes. Where an acid conditioning stage was used prior to the final alkaline conditioning stage in the pH range of 8.0 to 12.0, the density of the pulp was approximately 42 to 45 percent solids, and in the alkaline conditioning stages, 30 to 35 percent solids.

Density to flotation was 30 percent solids in all cases. This is a surprisingly high density for high recoveries and talc depression.

In all cases the conditioning was at the high end of the vigorous agitation conditioning range.

The grind was coarse, approximately 30 percent plus 100 mesh, and the coarse fraction contained a high percentage of middling particles.

In Test No. **1,** with such a coarse grind, and high percentage of middlings, the outstanding rougher recovery will be noted.

This is typical of one of the preferred embodiments of my invention on other ores such as copper ores wherein in using a prior acid conditioning stage to the final conditioning stage in the pH range of 8.0 to 12.0, the coarse middling particles are sufficiently activated and collector coated, and float in the routher flotation circuit, and by regrinding the rougher concentrate, or first cleaner concentrate, are liberated to produce a final high grade concentrate with outstanding recoveries.

Tests No. 1 and 2 were on the talc bearing ore.

Test No. 1	Reagents and Where Added							
Units	H_2SO_4 ccs 10% Soln.	CaO to pH	Fuel Oil Drops	Z6 ccs 1% Soln.	Na_2SiO_3 ccs 10% Soln.	Dow Froth 250- Drops	Time Mins.	pH
Rod Mill								8.9
No. 1 Agit.								4.8
Condit.	52.5						12.0	Begin
								6.7
								End

-continued

Test No. 1	Reagents and Where Added							
Units	H_2SO_4 ccs 10% Soln.	CaO to pH	Fuel Oil Drops	Z6 ccs 1% Soln.	Na_2SiO_3 ccs 10% Soln.	Dow Froth 250- Drops	Time Mins.	pH
No. 2 Agit. Condit.		10.5		20.0	15.0		6.0	10.2
No. 3 Agit Condit.		11.6		10.0		12	6.0	11.5 End
Float							6.0	
The Dowfroth 250 is a frother supplied by the Dow Chemical Company.								
Test No. 2								
Rod Mill			6.0					8.5
No. 1 Agit. Condit.	25.0				10.0	8	12.0	4.8 Begin 7.0 End
No. 2 Agit. Condit.		11.2					9.0	11.1 End
Float							6.0	
Tests No. 3 and 4 were on the ore sample containing no talc, but refractory as far as both grade and recovery were concerned.								
Test No. 3								
Rod Mill		11.3	3.0			Pine Oil 2 drops		
Agit. Condit.							3.0	11.3
Float							6.0	
Test No. 4								
Rod Mill		11.2	3.0			Pine Oil 2 drops		
Agit. Condit.					3.0		4.0	11.2
Float							6.0	

A laboratory rod mill was used for the initial preparation of the pulp and regrinding of the second cleaner concentrate. The ore charges were 1500 grams and the initial agitation conditioning stages were at a rotor speed of 2100 rpm.

In the laboratory the rougher concentrate was agitation conditioned in the 500 gram cell at a speed of 1650 rpm for 1.0 minutes with the addition of 0.03 lbs. per ton of **Z6**, and 0.3 lbs. per ton of sodium silicate at a pH of about 11.7.

The agitation conditioning period ahead of the third cleaner treating the reground second cleaner concentrate was at a pH of 12.4 to 12.45 for 6 minutes, and four minutes ahead of the fourth cleaner treating the third cleaner concentrate at the same pH.

In the laboratory the products were closed-circuited for four cycles. This closed-circuited laboratory work is referred to as the Weston Laboratory Standard.

The following is the summary of the metallurgical data obtained.

Circuit Description	Concentrate % Total Cu.	Tailings % Total Cu.
Plant Standard, using Aerofloat 242	25.8	0.114
Plant Standard for Rougher and Scavenger using Aerofloat 238 and Weston Cleaner Circuit	28.7	0.110
Plant Standard for Rougher and Scavenger using Z6 and Cleaner Circuit	26.5	0.098
Weston Circuit Throughout using Z6	28.4	0.048
Weston Laboratory Standard using Z6	30.1	0.048

It will be noted that in using the Weston Cleaner Circuit in combination with the conventional method of producing a rougher and scavenger concentrate, in each case both recovery and concentrate grade is improved.

The following metallurgy was obtained:

Test No.	Rougher Tailings % Wt.	% Mo	% Recovery Mo	Heads
1	93.4	0.002	99.1	0.21
2	96.6	0.006	97.1	0.21
3	98.5	0.037	87.9	0.30
4	97.0	0.019	93.8	0.30

In Test No. **1** potassium amyl xanthate was used as the collector alone and showed the highest recovery.

The lower weight rejection in the rougher tailings was due to the picking up of the large percentage of the middling particles in the coarse fraction of the feed to the flotation circuit.

Test No. **2** used Fuel Oil alone as the collector and showed excellent results but not equal to the xanthate.

In molybdenum ores these collectors may be used singly or together, it being a question of economics.

The vigorous agitation conditioning time in Test No. **3** was below the minimum for optimum metallurgy on this ore.

In using my preferred agitation conditioning circuit in treating molybdenum ores the minimum power consumption is about 0.25 kilowatt hours per ton of ore and the maximum is about 1.0 kilowatt hours per ton of ore.

In using 4 minutes in Test No. **4**, the recovery is substantially higher, and excellent metallurgy for this refractory type ore.

It will be noted that both Tests No. **3** and **4** used all-alkaline circuits. and Test No. **4** used a small amount of dispersant.

EXAMPLE XXXXII

The following comparative tests were carried out on a sample of the mill feed to a concentrator of a major copper producer in the western United States.

The ore was a copper ore in which the main sulphide copper mineral was chalcopyrite and the minor copper minerals, native copper and chrysocolla.

The ore contained a comparatively high percentage of pyrite and deleterious slimes.

In the two comparative tests carried out on this ore the time periods of conditioning in the agitation conditioning circuit were identical as was also the reagent balance. The only difference was in the speed of the rotor of the flotation machine in both the conditioning circuit and in the rougher flotation circuit. In the first test the speed of the rotor was 1550 rpm, and in the second test, 2100 rpm.

1500 gram charges of the ore were ground in a laboratory rod mill to approximately 30 percent plus 100 mesh, and transferred from the rod mill to a laboratory Denver cell using a 1000 gram bowl.

68 cc's of 10 percent H_2SO_4 were added to the cell and the pulp conditioned for 12 minutes at about 42 percent solids.

The pH of the pulp following grinding was 5.7, following the addition of the H_2SO_4, 2.7, and at the end of the acid conditioning stage, 4.2.

At the end of the acid conditioning stage the pulp density was reduced with tap water to about 30 percent solids, lime added to the pulp raising the pH to 10.8, 4 cc's of 1 percent sodium cyanide added to the pulp and agitation conditioned for 3 minutes. At the end of this conditioning, 23 cc's of 10 percent solution silicate was added and the conditioning continued for 6 minutes. The pH at the end was 10.4.

The pulp was further conditioned for 6 minutes with 6 cc's of 1 percent solution of sodium iso-propyl xanthate. At the end the pH of the pulp was raised to 11.6 with lime, 3 cc's of 10 percent Na_2SiO_3, 4 cc's of 1 percent Z11, and 4 drops of pine oil added to the pulp and conditioned for 6 minutes. The end pH was about 11.4.

A rougher float for 12 minutes was made and the rougher concentrate agitation conditioned in a 500 cc bowl for 4 minutes with the addition of 2½ cc's 10 percent Na_2SiO_3 and 2½ cc's of 1 percent Z11 and floated to produce a first cleaner tailing.

For comparative purposes the rougher tailings and first cleaner tailings are combined to show the relative metallurgy obtained.

In the test where the speed of the flotation machine rotor was 1550 rpm, the weight and analysis of the rougher plus first cleaner tailings was 84.0 percent and analyzed 0.149 percent Cu, while in the second test with the rotor speed at 2100 rpm, the comparative combined tailings weight was 87 percent and analyzed 0.073 percent Cu. In the case of the first test the agitation was amply sufficient to keep the solids in the pulp in suspension with the coarse grind employed. The outstanding metallurgical results in both rejecting a higher percentage of the waste minerals with an appreciable increase in recovery using the higher speed agitation (and thus higher energy imparted to the pulp) of the invention is most marked.

From the description of the invention it is obvious that in the first test with the lower speed of agitation and the resultant lower amount of horsepower used, and thus lower kilowatt hour consumption per ton of ore, the speed of the agitation mechanism, in this case the rotor, was too low to achieve the optimum results of the invention.

In plant practice wherein two agitation conditioning stages were used prior to the final agitation conditioning stage in the pH range of about 8.0 to 12.0, the power consumption in the individual agitation conditioning stages increased as the pH was increased using lime as the alkaline agent.

It will be appreciated that the power consumption of the agitation conditioning mechanisms will not only be a factor of speed of the mechanisms but also such factors as the conditions of flocculation of the pulp, pulp density, agitator design, and the impellor or rotor design.

In the research work where an acid conditioning stage is used in the pH range of 1.2 to 6.5 and preceeding the final agitation conditioning stage in the pH range of about 8.0 to 12, on some ores optimum metallurgical results were obtained in using the same rotor speed throughout the agitation conditioning circuit while in other cases to obtain optimum results a lower speed was necessary in the acid conditioning circuit than that used for the final alkaline conditioning stage. This phenomenon is not understood.

In applying the preferred embodiment of my invention to all types of sulphide bearing ores I prefer to treat all of the pulp of the appropriately prepared pulp of the ore in my agitation conditioning circuit either immediately following the grinding circuit, or following a thickening step wherein the pulp density has been adjusted or is within the range of about 25 to 50 percent solids, and wherein in at least the last agitation conditioning step prior to rougher flotation the density of the pulp has been adjusted to an optimum within the pulp density range of 20 to 40 percent solids, or alternatively, is adjusted subsequent to the last agitation conditioning step and either prior to or during flotation, or a combination of these conditions.

In the treatment of some ores, particularly in an acid conditioning stage, the pulp density may be as high as 70 percent solids.

It will be appreciated that the factors of power consumption to obtain the optimum metallurgy from my invention in at least one agitation conditioning circuit may be expressed in terms of the minimum rate of power drawn over a specific period of residence time of solids in the agitation conditioning circuit. This minimum is about 0.10 kilowatt hours per ton for a residence time period of about 4.0 minutes.

The total power consumption is represented by the total power drawn of all the agitation mechanisms in at least one agitation conditioning circuit, Thus the total power may be expressed in terms of kilowatt hours per ton of ore treated.

In the treatment of sulphide bearing ores that contain deleterious slimes the following table shows the preferred minimum residence time of the ore (as solids in the prepared pulp of the ore) in an agitation conditioning circuit wherein there is at least one alkaline agitation conditioning stage in the pH range of about 8.0 to 12.0. The table also shows the preferred total minimum kilowatt hour consumption per ton of ore treated as used by the agitator mechanism which in combination with the minimum residence time of the ore and other factors of the invention achieve heavy activation of the desired sulphide minerals at that particular stage in the flotation circuit.

The table further shows the preferred minimum of sulfhydryl anionic collector present in the suitably prepared pulp of the ore in at least the last agitation conditioning step of the agitation conditioning circuit prior to flotation.

The alkaline metal xanthates are treated as one part of the group of sulfhydryl anionic collectors, and the remaining members of the group, the thiols, thiocarbonates, monosubstituted thiorureas, and thiophosphates are described as "others."

Preferred Minimum Factors — Agitation Conditioning Circuit

Designated Type of Sulphide Ore	Residence* Time of Ground Ore Minutes	Kw. Hr.** Consumption Per Ton Ore	Collectors — lbs/Ton Ore Xanthates Alone	Others Alone	Mixed Z-Others
Copper	8	0.24	0.10	0.04	0.05
Molybdenum	6	0.18	0.05	0.02	0.03
Nickel	16	0.48	0.08	0.03	0.04

*The residence time of the ground ore in the agitation conditioning circuit is the calculation based on the available volume of the individual agitators to the pulp discharge level, and the density of the pulp in the individual agitator.

**The Kw. Hr. Consumption per ton of ore is the total horsepower drawn by the agitation mechanisms converted to kilowatts and divided by the tons of ore per hour treated in the agitation conditioning circuit.

The following table shows the preferred operating range of kilowatt hour consumption per ton of ore in the at least one alkaline agitation conditioning stage in the pH range of about 8.0 to 12.0, preferred range of rate of power consumption per 4 minutes of agitation conditioning time, and preferred range of conditioning time.

Preferred Operating Ranges

Designated Type of Sulphide Ore	Range Residence Time of Ore Minutes	Power Cons. Per 4 Minutes Residence Time	Total Power Cons. Range per Ton Ore
Copper	8 to 20	0.15 to 0.40	0.30 to 2.0
Molybdenum	4 to 8	0.15 to 0.30	0.15 to 0.6
Nickel	16 to 40	0.15 to 0.40	0.60 to 4.0

This table and the preceeding table are primarily applicable to treating all of the prepared pulp of the ore following the grinding circuit, or after a thickener stage following the grinding circuit, or following a scalp float, or in treating the slimes fraction of a sands-slime separation.

In treating a flotation product such as a rougher or scavenger concentrate, or cleaner tailings product where the object of an agitation conditioning circuit is to either increase the activation of the weakly activated minerals, or increase the recovery of the same minerals that due to weaker activation have dropped out of a concentrate into the cleaner tailings, my preferred agitation conditioning time range is about 2.0 to 10.0 minutes and the rate of power consumption of the agitation conditioning mechanisms is in the range of 0.05 to 0.15 kilowatt hours per ton of solids fed to the agitation conditioning circuit per 2.0 minutes residence time of the solids in the pulp. The preferred pH range is at an optimum pH point within the pH range of 8.0 to 12.0.

The preferred density of the pulp range is 15 to 35 percent solids.

I claim:

1. A process for the recovery by froth flotation of minerals from ores containing at least one sulphide mineral and wherein said at least one sulphide mineral is a major economic component of the ore and wherein a pulp of the ore has been prepared consisting of a pulp density selected from the range of about 20 to 60 percent solids and said solids of said ore have been ground to flotation feed size comprising: subjecting said prepared pulp of the ore to conditioning in at least one agitation conditioning circuit consisting of at least one alkaline agitation conditioning stage and at least three agitation conditioning steps wherein in said at least three alkaline agitation conditioning steps the conditioning is carried out in the presence of an alkaline agent selected from the group consisting of lime or calcium hydroxide, sodium hydroxide, sodium carbonate, and ammonium hydroxide and wherein the pH of the pulp is at at least one optimum pH point within the pH range of about 8.0 to 12.0 and wherein at least the final agitation conditioning step prior to flotation a sufficient concentration of collector selected from the group of sulfhydryl anionic collectors is present and said concentration of collector is in the range of about 0.02 to 0.40 pounds per ton of ore and wherein said pulp is agitation conditioned with agitation conditioning mechanisms with sufficient power consumption per ton of ore treated said power consumption being in the range of about 0.10 to 0.45 kilowatt hours per ton of ore per 4.0 minutes residence time of the ore in said at least one agitation conditioning stage and for a sufficiently long period of time and wherein said period of time is in the range of about 4.0 to 40 minutes to produce heavy activation of said at least one sulphide mineral to achieve high recovery of said at least one sulphide mineral in at least one subsequent froth flotation stage: subsequently in the presence of a suitable frother subjecting said agitation conditioned pulp to froth flotation to produce at least one flotation concentrate enriched in said at least one sulphide mineral and a tailings impoverished in said at least one sulphide mineral.

2. The process of claim 1 wherein said prepared pulp of the ore is the remaining pulp following one or more flotation stages.

3. The process of claim 1 wherein said prepared pulp of the ore is the slimes product produced in a sands-slime

4. The process of claim 1 wherein said agitation conditioning circuit contains at least three agitation conditioning steps and wherein in at least two of said steps said sufficient power consumption per ton of ore treated is employed to produce heavy agitation of at least the one said sulphide mineral to achieve high recovery of said at least one sulphide mineral in said at least one subsequent froth flotation stage.

5. The process of claim 1 wherein there is a dispersing agent present in the pulp in at least the last agitation

conditioning step prior to flotation.

6. The process of claim **1** wherein said at least one agitation conditioning stage in the pH range of about 8.0 to 12.0 is preceeded by at least one agitation conditioning stage in the pH range of about 1.2 to 7.9.

7. The process of claim **1** wherein said at least one agitation conditioning stage in the pH range of about 8.0 to 12.0 is preceeded by at least one agitation conditioning stage in the pH range of 12.0 to 12.3.

8. The process of claim **1** wherein there are at least two agitation conditioning stages in the pH range of about 8.0 to 12.0.

9. The process of claim **1** wherein said agitation conditioning steps are arranged in series.

10. The process of claim **1** wherein there is a sulphidizing agent present in the pulp in at least the last agitation conditioning step prior to flotation and the sulphidizing agent has been selected from the group consisting of hydrogen sulphide, sodium sulphide, calcium sulphide, and sodium hydrogen sulphide.

11. The process of claim **1** wherein in addition to treating said prepared pulp of the ore a scavenger flotation concentrate produced from said froth flotation is fed to at least one step of said agitation conditioning circuit.

12. The process of claim **1** wherein in addition to treating said prepared pulp of the ore at least part or all of one or more cleaner tailings produced from said froth flotation is returned to at least one step of said agitation conditioning circuit.

13. The process of claim **1** wherein in addition to treating said prepared pulp of the ore at least one flotation cleaner tailing produced from said froth flotation is subjected to agitation conditioning in at least one independent agitation conditioning step prior to being returned to at least one step in said agitation conditioning circuit.

14. The process of claim **1** wherein in addition to treating said prepared pulp of the ore at least one flotation cleaner tailing produced from said froth flotation is subjected to agitation conditioning in at least one independent agitation conditioning step prior to being fed to a flotation stage.

15. The process of claim **1** wherein said at least one agitation conditioning stage in the pH range of about 8.0 to 12.0 is preceeded by an acid conditioning stage wherein the pH of the pulp has been lowered by an acid agent selected from the group consisting of sulphuric acid, sulphurous acid, and sulphur dioxide to within the pH range of about 1.2 to 6.5.

16. The process of claim **15** wherein said acid agent is sulphuric acid and is control added to the pulp of the ore on a pounds per ton of ore basis.

17. The process of claim **15** wherein said acid agent is sulphuric acid and is control added to the pulp of the ore on a "pounds per ton or ore-density of the pulp" basis.

18. The process of claim **15** wherein said acid agent is sulphuric acid and is control added to the pulp of the ore to maintain an optimum pH point.

19. The process of claim **1** wherein said at least one alkaline agitation conditioning stage consists of at least two agitation conditioning steps and said power consumption of said agitation conditioning mechanisms is in range of about 0.17 to 0.53 kilowatt hours per ton of ore treated per 5 minutes residence time of the ore in the said at least one agitation conditioning stage.

20. The process of claim **1** wherein said at least one alkaline agitation conditioning stage consists of at least two agitation conditioning steps and said power consumption of the agitation conditioning mechanisms is in the range of about 0.22 to 0.82 kilowatt hours per ton of ore treated per 8 minutes residence time of the ore in said at least one agitation conditioning stage.

21. The process of claim **1** wherein said at least one alkaline agitation conditioning stage consists of at least three agitation conditioning steps and said power consumption of the agitation conditioning mechanisms is in the range of about 0.23 to 0.65 kilowatt hours per ton of ore treated per 6 minutes residence time of the ore in said at least one agitation conditioning stage.

22. The process of claim **1** wherein said at least one alkaline agitation conditioning stage consists of at least three agitation conditioning steps and said power consumption of the agitation conditioning mechanisms is in the range of about 0.28 to 0.94 kilowatt hours per ton of ore treated per 9 minutes residence time of the ore in said at least one agitation conditioning stage.

23. The process of claim **1** wherein said at least one alkaline agitation conditioning stage consists of at least four agitation conditioning steps and said power consumption of the agitation conditioning mechanisms is in the range of about 0.34 to 1.06 kilowatt hour per ton of ore treated per 10 minutes residence time of the ore in said at least one agitation conditioning stage.

24. A process for the recovery by froth flotation of copper minerals from copper bearing ores containing at least one sulphide copper mineral and wherein said at least one sulphide copper mineral is a major economic component of the ore wherein a pulp of the ore has been prepared consisting of a pulp density selected from the range of about 20 to 60 percent solids and wherein said solids of said ore have been ground to flotation feed size comprising: subjecting said prepared pulp of the ore to conditioning in at least one agitation conditioning circuit consisting of at least one alkaline agitation conditioning stage and at least three agitation conditioning steps wherein in said at least three alkaline agitation conditioning steps the conditioning is carried out in the presence of an alkaline agent selected from the group consisting of lime or calcium hydroxide, sodium hydroxide, sodium carbonate and ammonium hydroxide and wherein the pH of the pulp is at at least one optimum pH point within the pH range of about 8.0 to 12.0 and wherein in at least the final agitation conditioning step prior to flotation a sufficient concentration of collector selected from the group of sulfhydryl anionic collectors is present and said concentration of collector is in the range of about 0.04 to 0.40 pounds per ton of ore and wherein said pulp is agitation conditioned with agitation conditioning mechanisms with sufficient power consumption per ton of ore treated said power consumption being in the range of about 0.12 to 0.40 kilowatt hours per ton of ore per 4.0 minutes residence time of the ore in said at least one agitation conditioning stage and for a sufficiently long period of time and wherein said period of time is in the range of about 4.0 to 30 minutes to produce heavy activation of said at least one copper mineral to achieve high recovery of said at least one copper mineral in at least one subsequent froth flotation stage subsequently in the presence of a suitable frother subjecting said agitation conditioned pulp to froth flotation to produce at least one flotation concentrate enriched in said at least

one copper mineral and a tailings impoverished in said at least one copper mineral.

25. The process of claim **24** wherein said prepared pulp of the ore is the remaining pulp following one or more flotation stages.

26. The process of claim **24** wherein said prepared pulp of the ore is the slimes product produced in a sands-slime split of the ore.

27. The process of claim **24** wherein said agitation conditioning circuit contains at least three agitation conditioning steps and wherein in at least two of said steps said sufficient power consumption per ton of ore treated is employed to produce heavy activation of said at least one copper mineral to achieve high recovery of said at least one copper mineral in said at least one subsequent froth flotation stage.

28. The process of claim **24** wherein there is a dispersing agent present in the pulp in at least the last agitation conditioning step prior to flotation.

29. The process of claim **24** wherein said at least one agitation conditioning stage in the pH range of about 8.0 to 12.0 is preceeded by at least one agitation conditioning stage in the pH range of about 1.2 to 7.9.

30. The process of claim **24** wherein said at least one agitation conditioning stage in the pH range of about 8.0 to 12.0 is preceeded by at least one agitation conditioning stage in the pH range of 12.0 to 12.3.

31. The process of claim **24** wherein there are at least two agitation conditioning stages in the pH range of about 8.0 to 12.0.

32. The process of claim **24** wherein said agitation conditioning steps are arranged in series.

33. The process of claim **24** wherein there is a sulphidizing agent present in the pulp in at least the last agitation conditioning step prior to the flotation and the sulphidizing agent is selected from the group consisting of hydrogen sulphide, sodium sulphide, calcium sulphide, and sodium hydrogen sulphide.

34. The process of claim **24** wherein in addition to treating said prepared pulp of the ore, a scavenger flotation concentrate produced from said froth flotation is fed to at least one step of said agitation conditioning circuit.

35. The process of claim **24** wherein in addition to treating said prepared pulp of the ore at least part or all of one or more cleaner tailings produced from said froth flotation is returned to at least one step of said agitation conditioning circuit.

36. The process of claim **24** wherein in addition to treating said prepared pulp of the ore at least one flotation cleaner tailing produced from said froth flotation is subjected to agitation conditioning in at least one independent agitation conditioning step prior to being fed to a flotation stage.

37. The process of claim **24** wherein at least one flotation cleaner tailing is subjected to agitation conditioning in at least one independent agitation conditioning step prior to being fed to a flotation stage.

38. The process of claim **24** wherein said at least one agitation conditioning stage in the pH range of about 8.0 to 12.0 is preceeded by an acid conditioning stage wherein the pH of the pulp has been lowered by an acid agent selected from the group consisting of sulphuric acid, sulphurous acid, and sulphur dioxide to within the pH range of about 1.2 to 6.5.

39. The process of claim **37** wherein said acid agent is sulphuric acid and is control added to the pulp of the ore on a pounds per ton of ore basis.

40. The process of claim **39** wherein said acid agent is sulphuric acid and is control added to the pulp of the ore on a "pounds per ton of ore-density of the pulp" basis.

41. The process of claim **38** wherein said acid agent is sulphuric acid and is control added to the pulp of the ore to an optimum pH point of the pulp of the ore.

42. The process of claim **24** wherein said at least one alkaline agitation conditioning stage consists of at least two agitation conditioning steps and said power consumption of the said agitation conditioning mechanisms is in range of about 0.17 to 0.53 kilowatt hours per ton of ore treated per 5 minutes residence time of the ore in said at least one agitation conditioning stage.

43. The process of claim **24** wherein said at least one alkaline agitation conditioning stage consists of at least two agitation conditioning steps and said power consumption of the agitation conditioning mechanisms is in the range of about 0.22 to 0.82 kilowatt hours per ton of ore treated per **8** minutes residence time of the ore in said at least one agitation conditioning stage.

44. The process of claim **24** wherein said at least one alkaline agitation conditioning stage consists of at least three agitation conditioning steps and said power consumption of the agitation conditioning mechanisms is in the range of about 0.23 to 0.65 kilowatt hours per ton of ore treated per 6 minutes residence time of the ore in said at least one agitation conditioning stage.

45. The process of claim **24** wherein said at least one alkaline agitation conditioning conditioning stage consists of at least three agitation conditioning steps and said power consumption of the agitation mechanisms is in the range of about 0.28 to 0.94 kilowatt hours per ton of ore treated per 9 minutes residence time of the ore in said at least one agitation conditioning stage.

46. The process of claim **24** wherein said at least one alkaline agitation conditioning stage consists of at least four agitation conditioning steps and said power consumption of the agitation conditioning mechanisms is in the range of about 0.34 to 1.06 kilowatt hours per ton of ore treated per 10 minutes residence time of the ore in said at least one agitation conditioning stage.

47. The process of claim **24** wherein said at least one alkaline agitation conditioning stage consists of at least two agitation conditioning steps and said total power consumption of said agitation conditioning mechanisms is in the range of 0.24 to 1.5 kilowatt hours per ton of ore treated and the residence time of the ore in said at least one agitation conditioning stage is in the range of about 8.0 to 20 minutes.

48. The process of claim **24** wherein said collector is an alkali metal xanthate and said sufficient concentration of collector is a minimum of about 0.10 lbs. per ton of ore treated.

49. The process of claim **24** wherein said collector is selected from the group consisting of thiols, thiocarbonates, monosubstituted thioureas, and thiophosphates and said sufficient concentration of collector is a minimum of 0.03 lbs. per ton of ore treated.

50. The process of claim **24** wherein said collector is a combination of alkali metal xanthate and collector selected from the group consisting of thiols, thiocarbonates, monosubstituted thioureas, and thiophosphates and said sufficient concentration of collector is a minimum of **0.05** lbs. per ton of ore treated.

51. A process for the recovery by froth flotation of nickel minerals from nickel bearing ores containing at least one sulphide nickel mineral and wherein said at

least one sulphide nickel mineral is a major economic component of the ore and wherein a pulp of the ore has been prepared consisting of a pulp density selected from the range of about 20 to 60 percent solids and said solids of said ore have been ground to flotation feed size comprising: subjecting said prepared pulp of the ore to conditioning in at least one agitation conditioning circuit consisting of at least one alkaline agitation conditioning stage and at least three agitation conditioning steps wherein in said at least one alkaline agitation conditioning stage the conditioning is carried out in the presence of an alkaline agent selected from the group consisting of lime or calcium hydroxide, sodium carbonate, sodium hydroxide, and ammonium hydroxide and wherein the pH of the pulp is at at least one optimum pH point within the pH range of about 8.0 to 12.0 and wherein in at least the final agitation conditioning step prior to flotation a sufficient concentration of collector selected from the group of sulfhydryl anionic collectors is present and said concentration of collector is in the range of about 0.03 to 0.40 pounds per ton of ore and wherein said pulp is agitation conditioned with agitation conditioning mechanisms with sufficient power consumption per ton of ore treated said power consumption being in the range of about 0.12 to 0.40 kilowatt hours per ton of ore per 4.0 minutes residence time of the ore in said at least one agitation conditioning stage and for a sufficiently long period of time and wherein said period of time is in the range of about 4.0 to 40 minutes to produce heavy activation of said at least one sulphide nickel mineral to achieve high recovery of said at least one nickel mineral in at least one subsequent froth flotation stage: subsequently in the presence of a suitable frother subjecting said agitation conditioned pulp to froth flotation to produce at least one flotation concentrate enriched in said at least one sulphide nickel mineral and a tailings impoverished in said at least one sulphide nickel mineral.

52. The process of claim **51** wherein said prepared pulp of the ore is the remaining pulp following one or more flotation stages.

53. The process of claim **51** wherein said agitation conditioning circuit contains at least three agitation conditioning steps and wherein in at least two of said steps said sufficient power consumption per ton of ore treated is employed to produce heavy activation of at least the one said sulphide nickel mineral to achieve high recovery of said at least one sulphide nickel mineral in at least one subsequent froth flotation stage.

54. The process of claim **51** wherein said sufficiently long period of time to produce heavy activation of said at least one sulphide nickel mineral to achieve high recovery of said at least one sulphide nickel mineral in at least one subsequent froth flotation stage is a minimum period of eight minutes.

55. The process of claim **51** wherein there is a dispersing agent present in the pulp in at least the last agitation conditioning step prior to flotation.

56. The process of claim **51** wherein said at least one agitation conditioning stage in the pH range of about 8.0 to 12.0 is preceeded by at least one agitation conditioning stage in the pH range of about 1.2 to 7.9.

57. The process of claim **51** wherein there are at least two agitation conditioning stages in the pH range of about 8.0 to 12.0.

58. The process of claim **51** wherein said agitation conditioning steps are arranged in series.

59. The process of claim **51** wherein in addition to treating said prepared pulp of the ore a scavenger flotation concentrate produced from said froth flotation is fed to at least one step of said agitation conditioning circuit.

60. The process of claim **51** wherein in addition to treating the prepared pulp of the ore at least part or all of one or more cleaner tailings produced from said froth flotation is returned to at least one step of said agitation conditioning circuit.

61. The process of claim **51** wherein in addition to treating said prepared pulp of the ore at least one flotation cleaner tailing produced from said froth flotation is subjected to agitation conditioning in at least one independent agitation conditioning step prior to being returned to at least one step in said agitation conditioning circuit.

62. The process of claim **51** wherein in addition to treating said prepared pulp of the ore at least one flotation cleaner tailing produced from said froth flotation is subjected to agitation conditioning in at least one independent agitation conditioning step prior to being fed to a flotation stage.

63. The process of claim **51** wherein said prepared pulp of the ore contains copper sulphate.

64. The process of claim **51** wherein copper sulphate is present in said prepared pulp of the ore prior is at least the last agitation conditioning step prior to flotation.

65. The process of claim **51** wherein said at least one agitation conditioning stage in the pH range of about 8.0 to 12.0 is preceeded by an acid conditioning stage wherein the pH of the pulp has been lowered by an acid agent selected from the group consisting of sulphuric acid, sulphurous acid, and sulphur dioxide to within the pH range of about 1.2 to 6.5.

66. The process of claim **65** wherein said acid agent is sulphuric acid and is control added to the pulp of the ore on a pounds per ton of ore basis.

67. The process of claim **65** wherein said acid agent is sulphuric acid and is control added to the pulp of the ore on a "pounds per ton or ore-density of the pulp" basis.

68. The process of claim **65** wherein said acid agent is sulphuric acid and is control added to the pulp of the ore to an optimum pH point of the pulp of the ore.

69. The process of claim **51** wherein said at least one alkaline agitation conditioning stage consists of at least two agitation conditioning steps and said power consumption of said agitation conditioning mechanisms is in the range of about 0.17 to 0.53 kilowatt hours per ton of ore treated per 5 minutes residence time of the ore in said at least one agitation conditioning stage.

70. The process of claim **51** wherein said at least one alkaline agitation conditioning stage consists of at least two agitation conditioning steps and said power consumption of the agitation conditioning mechanisms is in the range of about 0.22 to 0.82 kilowatt hours per ton of ore treated per 8 minutes residence time of the ore in said at least one agitation conditioning stage.

71. The process of claim **51** wherein said at least one alkaline agitation conditioning stage consists of at least three agitation conditioning steps and the said power consumption of the agitation conditioning mechanisms is in the range of about 0.23 to 0.65 kilowatt hours per ton of ore treated per 6 minutes residence time of the ore in said at least one agitation conditioning stage.

72. The process of claim 51 wherein said at least one alkaline agitation conditioning stage consists of at least three agitation conditioning steps and said power consumption of the agitation conditioning mechanisms is in the range of about 0.28 to 0.94 kilowatt hours per ton of ore treated per 9 minutes residence time of the ore in said at least one agitation conditioning stage.

73. The process of claim 51 wherein said at least one alkaline agitation conditioning stage consists of at least four agitation conditioning steps and said power consumption of the agitation conditioning mechanisms is in the range of about 0.34 to 1.06 kilowatt hours per ton of ore treated per 10 minutes residence time of the ore in said at least one agitation conditioning stage.

74. The process of claim 51 wherein said at least one alkaline agitation conditioning stage consists of at least two agitation conditioning steps and said total power consumption of said agitation conditioning mechanisms is in the range of 0.48 to 3.0 kilowatt hours per ton of ore treated and the residence time of the ore in said at least one agitation conditioning stage is in the range of about 16 to 40 minutes.

75. A process for the recovery by froth flotation of molybdenum minerals from molybdenum bearing ores containing at least one sulphide molybdenum mineral and wherein said at least one molybdenum mineral is a major economic component of the ore and wherein a pulp of the ore has been prepared consisting of a pulp density selected from the range of about 20 to 60 percent solids and said solids of said ore have been ground to flotation feed size comprising: subjecting said prepared pulp of the ore to conditioning in at least one alkaline agitation conditioning circuit consisting of at least one alkaline agitation conditioning stage and of at least three agitation conditioning steps wherein in said at least one alkaline agitation conditioning stage the conditioning is carried out in the presence of an alkaline agent selected from the group consisting of lime or calcium hydroxide, sodium carbonate, sodium hydroxide, and ammonium hydroxide and wherein the pH of the pulp is at at least one optimum pH point within the pH range of about 8.0 to 12.0 and wherein in at least the final agitation conditioning step prior to flotation a sufficient concentration of collector selected from the group consisting of sulfhydryl anionic collectors and fuel oil is present and said sufficient concentration of sulfhydryl anionic collectors is in the range of about 0.02 to 0.30 pounds per ton of ore and wherein said pulp is agitation conditioned with agitation conditioning mechanisms with sufficient power consumption per ton of ore treated said power consumption being in the range of about 0.12 to 0.30 kilowatt hours per ton of ore per 4.0 minutes residence time of the ore in said at least one agitation conditioning stage and for a sufficiently long period of time and wherein said period of time is in the range of about 4.0 to 20 minutes to produce heavy activation of said at least one sulphide molybdenum mineral to achieve high recovery of said at least one molybdenum mineral in at least one subsequent froth flotation stage: subsequently in the presence of a suitable frother subjecting said agitation conditioned pulp to froth flotation to produce at least one flotation concentrate enriched in said at least one sulphide molybdenum mineral and a tailing impoverished in said at least one sulphide molybdenum mineral.

76. The process of claim 75 wherein said prepared pulp of the ore is the remaining pulp following one or more flotation stages.

77. The process of claim 75 wherein there is a dispersing agent present in the pulp in at least the last agitation conditioning step prior to flotation.

78. The process of claim 75 wherein said at least one agitation conditioning stage in the pH range of about 8.0 to 12.0 is preceeded by at least one agitation conditioning stage in the pH range of about 1.2 to 7.9.

79. The process of claim 75 wherein said at least one agitation conditioning stage in the pH range of about 8.0 to 12.0 is preceeded by at least one agitation conditioning stage in the pH range of 12.0 to 12.3.

80. The process of claim 75 wherein there are at least two agitation conditioning stages in the pH range of about 8.0 to 12.0.

81. The process of claim 75 wherein said agitation conditioning steps are arranged in series.

82. The process of claim 75 wherein in addition to treating said prepared pulp of the ore a scavenger flotation concentrate produced from said froth flotation is fed to at least one step of said agitation conditioning stage.

83. The process of claim 75 wherein in addition to treating said prepared pulp of the ore at least part or all of one or more cleaner tailings produced from said froth flotation is returned to at least one step of said agitation conditioning circuit.

84. The process of claim 75 wherein in addition to treating said prepared pulp of the ore at least one flotation cleaner tailing produced from said froth flotation is subjected to agitation conditioning in at least one independent agitation conditioning step prior to being returned to at least one step in said agitation conditioning circuit.

85. The process of claim 75 wherein in addition to treating said prepared pulp of the ore at least one flotation cleaner tailing produced from said froth flotation is subjected to agitation conditioning in at least one independent agitation conditioning step prior to being fed to a flotation stage.

86. The process of claim 75 wherein said selected collector is a combination of a sulfhydryl anionic collector and a fuel oil.

87. The process of claim 75 wherein said selected collector is fuel oil.

88. The process of claim 75 wherein said at least one agitation conditioning stage in the pH range of about 8.0 to 12.0 is preceeded by an acid conditioning stage wherein the pH of the pulp has been lowered by an acid agent selected from the group consisting of sulphuric acid, sulphurous acid, and sulphur dioxide to within the pH range of about 1.2 to 6.5.

89. The process of claim 88 wherein said acid agent is sulphuric acid and is control added to the pulp of the ore on a pounds per ton of ore basis.

90. The process of claim 88 wherein said acid agent is sulphuric acid and is control added to the pulp of the ore on a "pounds per ton of ore-density of the pulp" basis.

91. The process of claim 88 wherein said acid agent is sulphuric acid and is control added to the pulp of the ore to an optimum pH point of the pulp of the ore.

* * * * *

UNITED STATES PATENT OFFICE

CERTIFICATE OF CORRECTION

Patent No. 3,919,079 Dated November 11, 1975

Inventor(s) David Weston

It is certified that error appears in the above-identified patent and that said Letters Patent are hereby corrected as shown below:

Claim 3, line 58, after "slime" add --split--.

Claim 19, line 66, following "in" insert --the--.

Claim 45, line 30, delete first "conditioning".

Claim 45, line 32, insert --conditioning-- after "agitation".

Signed and Sealed this

Twenty-seventh Day of July 1976

[SEAL]

Attest:

RUTH C. MASON
Attesting Officer

C. MARSHALL DANN
Commissioner of Patents and Trademarks

UNITED STATES PATENT AND TRADEMARK OFFICE

CERTIFICATE OF CORRECTION

PATENT NO. : 3,919,079

DATED : November 11, 1975

INVENTOR(S) : DAVID WESTON

It is certified that error appears in the above-identified patent and that said Letters Patent are hereby corrected as shown below:

Column 58, line 63, delete "agitation" and insert in lieu thereof -- activation --.

Signed and Sealed this

Thirty-first Day of August 1976

[SEAL]

Attest:

RUTH C. MASON
Attesting Officer

C. MARSHALL DANN
Commissioner of Patents and Trademarks