THE FLOTATION OF FINES

by

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INTRODUCTION

Froth flotation has been an extremely successful method for the beneficiation of a variety of metallic and non-metallic minerals, ranging from coal to gold, and ranging in size from approximately 0.065 of an inch to 0.0004 of an inch. Ores coarser than 0.065 of an inch can be treated by gravity methods, but materials finer than 0.0004 of an inch have proven to be difficult to separate by conventional methods. Some ores, due to geological occurrence, have very small sizes of liberation and require extremely fine grinding to liberate the unwanted mineral from the wanted mineral. It has generally been agreed that flotation of minerals less than 0.0004 of an inch or ten microns in size has been difficult--if not impossible.

Flotation has been used since 1902 in Australia where it was first applied to sulfide ores to separate the sulfides from the gangue minerals. In a period of eighteen years it superseded, to a great extent, all other methods that required grinding the ores to less than 48 mesh Tyler. Froth flotation was first applied to sulfide ores but has found application to other types of materials such as coal, silicates, and other non-metallics.

The froth flotation process was designed to separate finely divided solids from each other. The process divides metallic and non-metallic minerals into two fractions; one a concentrate, and the other fraction a tailing. The first step in the process of froth flotation is severance, whereby the wanted mineral is freed from the unwanted mineral by crushing and grinding. The separation of the wanted mineral from the unwanted minerals takes place in an aqueous medium with adhesion of the wanted mineral to an air bubble which levitates the wanted mineral to the surface where it is removed.

The two reagents that are necessary for flotation are a collector and a frother. The collector may rarely take on the functions of a frother, but its usual function is that of a collector. A suitable collector reacts at the surface of the wanted mineral to give a surface which will adhere to an air bubble. The frother is a reagent that reduces the surface tension of the water so that a froth may be obtained. The froth carries the wanted mineral on top of the slurry of ore and water. The froth is continuously removed and treated further in other flotation cells so that when the process is completed the concentrates are of sufficiently high grade for consumption by a smelter or the ultimate consumer.

Previous work done on the flotation of minus ten micron material, sometimes referred to as slimes, has shown that generally poor results can be expected from the flotation of this very fine material. Various explanations have been put forth to explain this. The main field of study has been the effects at the interface of the air, water, and mineral. The author of this thesis will explore a new line of approach for the flotation of fines by investigating the effect of increasing gravity by the use of centrifugal force on the collection and separation of the finely ground ores in the minus ten micron range.

LITERATURE REVIEW

Introduction. Flotation has been utilized since the early 1900's for the treatment of particles too fine to be recovered by gravity methods. It is a process of attaching the wanted mineral to air bubbles, leaving the unwanted mineral unattached to any air bubble, thereby differentially separating the material. The wanted mineral concentrates on top of the slurry as a froth and is continuously removed from the mineral that sinks to the bottom of the cell.

There are two methods of recovering a mineral from a flotation operation. In one method the wanted mineral is floated from the gangue, and the gangue sinks to the bottom of the cell. In the second method the gangue is floated, and the wanted mineral is allowed to sink to the bottom of the cell. The choice of method is based on two considerations: first, the percentage of gangue and wanted mineral in the pulp, the procedure being generally to float the material that is less by volume. The second consideration of method to be used is when only one mineral can be coated with a surface which will adhere to an air bubble.

The flotation process rests primarily upon the attachment of a collector coating on the surface of the mineral in the pulp. Bubbles then attach themselves to this surface. This surface is formed with suitable reagents that react with the surface by various ways, such as chemical reaction or chemsorption, a combination of adsorption and chemical reaction. The theory behind this is yet not completely clear and is beyond the scope of this thesis.

It has been set as a goal of this thesis, a review of the literature on the flotation of extreme fines, as the writer believes this a direction in which mineral dressing has to advance. Each of the various factors connected with the flotation of extreme fines will be treated separately.

Fines and Slimes. The definition of fines and slimes is constantly being changed as better procedures are developed for the handling of finely ground ores. Before the advent of flotation, materials finer than approximately 150 mesh Tyler were considered to be slimes. In the early days of flotation, ores ground finer than twenty to thirty microns were considered to be slimes. (16)(35) Today the upper limit for slimes is ten microns.

The chief method today of handling slimes is either to disperse the slimes by modifying agents or remove them from the circuit mechanically and waste the fine material. The lower limit of flotation of slimes is held to be the point at which Brownian Movement occurs. (8)(14)(19) This movement is the random and irregular motion of very fine particles immersed in a fluid. These particles are small enough to be affected by the random motion of the molecules of the fluid. The difficulties met in floating slimes has been explained by Gaudin as due to the nonencounter of the fine particles with the bubbles in the slurry, and also the mutual coating of the wanted and unwanted mineral with ions from each of their respective surfaces.(9)

There are two important theories regarding fine particle coatings. The cementing hypothesis was advanced by del-Giudice in 1934 which postulated a chemical reaction between the slimes and larger minerals causing slime coatings of fines on the various minerals in the pulp.(6) The ionic hypothesis advanced by Sun explained slime coatings on the basis of electrical phenomena at the interfaces(29) so that the charges on the minerals were such as to cause attraction between particles or repulsion, depending on the charge at the interface.(34)(29)(6)

Most operators using flotation in plants remove the slimes and send them to waste.(7) It has been shown that particles of extreme fineness can be floated if they are flocculated.(5)(25) The writer is led to the view that practically any size of mineral can be floated if it can be flocculated.(25)(16)

Conditions that control flocculation also control slime coatings on larger particles.(1) Good results have been achieved by adding the collector at the grinding stage so that immediate coating of the fresh mineral surface with the collector is attained. When this is done the mutual coating of gangue and wanted mineral is prevented and higher recovery is achieved.(10)

Flocculation. Various authors have stated that flocculation is necessary for the flotation of extremely fine sizes--sizes as fine as 2100 mesh Tyler have been floated.(27) They postulate that this is due to flocculation of one particular mineral together so that particle-gas collision can be attained. Dispersion of the extremely fine sizes cause flotation of the selected mineral to cease. Dispersion can readily be obtained by the use of liberal amounts of gelatin added to the slurry. Flocculation is enhanced by organic and inorganic reagents in the slurry which reduce the surface charges on the mineral to enable them to come close enough together to form larger particles. Thus, flocculation takes place without the binding forces of minute bubbles between the slime particles.(30)

The three phases, as commonly used in flotation terminology, are solid, liquid, and gaseous. Bubbles of air introduced into the slurry by various means are one of these three main phases. Flotation engineers are primarily concerned with the attachment of these bubbles to the desired mineral in such a way that it will be levitated to the surface and removed. The mechanism of attachment has been theorized to be: (1) by precipitation,(31) that is, by the precipitation of the gas from the liquid onto the mineral, and (2) the collision theory whereby the particle encounters a gas bubble and is stuck onto the gas bubble due to the particle having a preferential affinity for air.(27)

For the flotation of fine sizes, the size of the bubble is of importance because of the laminary layer and a rather large interfacial free energy that surrounds the bubble. The small particle has to overcome this laminary layer and interfacial free energy to attach itself to a bubble. For the flotation of extremely fine sizes, it would seem imperative that small bubbles be used to float small particles.(27) This can be visualized as due to the low interfacial free energy that might occur when a rubber ball hits a building at extreme speed versus a sand grain hitting the rubber ball. The rubber ball, which is analogous to an air bubble, will deform when hitting the building. The sand grain at the same velocity would have difficulty deforming the rubber ball. This is the same case as where a small bubble hits a small but larger particle, or where a small bubble hits a smaller particle.(22)(24)(34)

The attachment of bubbles to particles is also dependent upon the number of particles per bubble. The larger the number of bubbles per particle, the greater the probability of contact. Some authors have recommended that the pulp density for extreme fines be greater than that for coarser material.(15)

<u>Collector coating.</u> The collector coverage of mineral surfaces has been investigated and shown to be incomplete in cases of even good flotation results. Various investigations have shown collector coatings on successful flotation in the range of 3 to 100 per cent, certainly a wide variation, but points up the difficulty of actually measuring the collector coating on the mineral or the small amount of collector coating required for flotation.(11)(23) Incomplete

collector coating of the fine particles has been one of the hypotheses advanced for poor flotation of fines. The fines that have more surface area use so much collector that exorbitant amounts of collector are required for the flotation of extreme fines. If this hypothesis were followed, it would seem that all that would have to be done to float extreme fines would be to increase the amount of collector used, and the fines would float. This is not the case however, because when this method is applied, gangue and mineral both float.(28) Because of this fact, it becomes impossible to separate the wanted mineral from the gangue by ordinary flotation processes.

It has been advanced that when fines are present in a liquid there is an extreme amount of surface area so that both the gangue and the mineral contribute ions from their surfaces to the liquid. These ions then coat the gangue and mineral with a surface that is essentially the same and renders flotation impossible. Gaudin and Mallozemoff have postulated the addition of collector to the grinding circuit to coat the surfaces immediately as they are formed.(10)

Specific Flotation Rates of Fines. The kinetics of flotation, which measures the specific rate with the dimension of reciprocal time, has been investigated by various researchers. The conclusion drawn was that the specific rate of flotation is at a maximum at approximately fifty microns; falling off rapidly as the size increases or decreases.(5)(13)(21)(16)(26)

Effect of Gases. Contradictory results have been obtained for the effect of gases upon surfaces of minerals. Some authors report that oxygen was necessary for activation of mineral surfaces, and others showed that nitrogen had beneficial results.(12)(3)(18)

Increasing Gravity. The only reference to the effect of gravity on the flotation of particles was contained in a Russian article entitled "Flotation and the Dimension of Particles" where a formula for the lower limit in size of flotation was given.(2)

$T\,1/2$ = surface tension at the boundary of oil and water

θ = angle of contact in degrees

d_1 = density of the particle

d_2 = density of the water

g = acceleration of gravity

The lower limit of particle size is:

$$r\ \min = \sqrt{\frac{1/2\ \ T\,1/2\ (1-\cos\theta)}{g\,(d_1 - d_2)}}$$

It can be deduced from this formula that the minimum radius is porportional to the reciprocal of the gravity.

Conclusions. A study of the literature indicates that due to the large surface area of fine minerals, large amounts of collector are required for collector coverage. If large amounts of collecter are used the surfaces of the mineral would be covered, even though excessive amounts of collector may be necessary for the flotation of these fines. The literature review also points to the decreasing flotation rates for finer particles, indicating that a method must be found for increasing these rates for the successful flotation of the fines. Bubble and mineral non-attachment has been postulated as being one of the contributing causes for the non-flotation of the slimes. Gaudin has advanced the theory that the surfaces of the mineral and the gangue become alike due to their mutual coating from ions given off by minerals in the pulp causing non-flotation of the fines. The conclusions drawn by the writer from the literature review are that some method must be found for attaching bubbles to the fines, as well as a method for separating the floated mineral from the floated gangue.

EXPERIMENTAL

Purpose of Investigation

The experimental work was designed to investigate the effect of increasing gravity on the froth flotation of finely ground ores.

Plan of Experimentation

The first stage of the experimentation was to investigate the effect of gravity on the collection and flotation of finely ground ores in the minus ten micron range. To do this, an International Centrifuge, size 2, model V, manufactured by the International Equipment Company of Boston, Massachusetts was modified so that it could be operated under centrifugal force. The centrifuge was modified by using a stainless steel perforated basket, catalog number 5-327 supplied by the Fisher Scientific Company of Silver Spring, Maryland. The holes in the basket were filled by using number 5-40 machine screws and nuts and painting the inside of the basket with Plicote Laboratory paint, number 13-312, also supplied by the Fisher Scientific Company. A drain chamber of stainless steel, number 5-418-5, manufactured by the Fisher Scientific Company was first installed in the centrifuge. Two blocks, 2.5 inches high, 1.3 inches wide, and 3/8 inches thick with four 1/8-inch holes, were then installed in the perforated basket referred to in Figure 1 as the rotating eleven-inch flotation cell. This cell was next installed in the centrifuge. A froth pick-up box with a vacuum line was then constructed and mounted onto the cell so that it could be moved in and out independently of the cell.

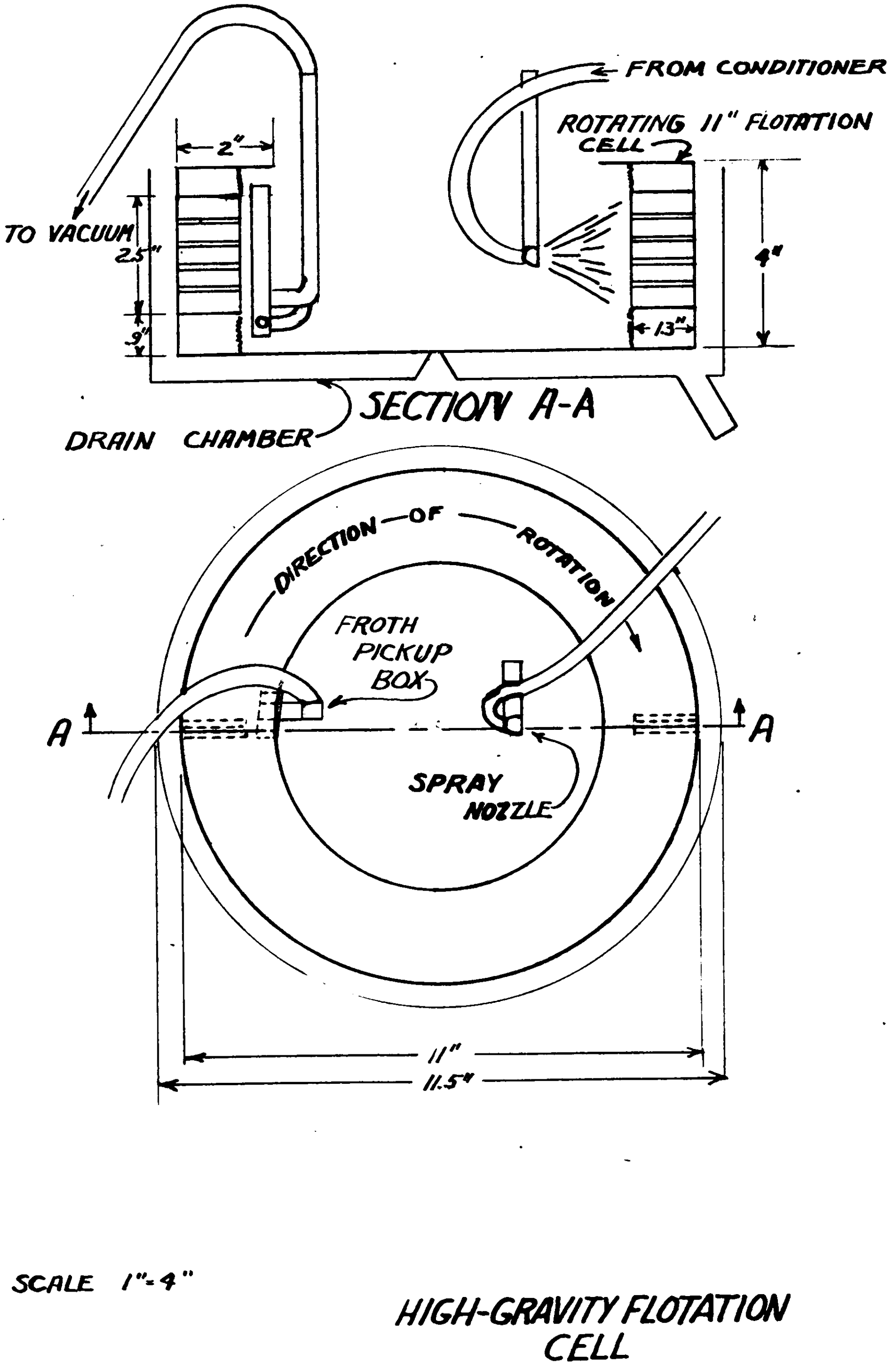

FIGURE 1. HIGH-GRAVITY FLOTATION CELL

A spray nozzle manufactured by Spray Engineering Company, 138 Cambridge Street, Burlington, Massachusetts, number 062 F, 55 degrees was then installed and mounted so that it could be moved independently of the flotation cell.

In operation the cell was to have rotated and held the slurry to the outside of the cell. The spray would have then impinged on the vertical wall of the slurry, and froth should have appeared on the surface. The unfloated solids in the slurry were to have gone through the blocks attached to the stainless steel basket by way of the four holes in each block. The solids, after having gone through the blocks, would have discharged into the drain chamber where they would have been collected as flotation sink. See Figure 1. The froth pick-up box would then have picked up any froth that would have formed on the slurry by means of a vacuum line that transferred the froth to a bottle where the froth would have been collected.

The silica used in this experiment was obtained from the Department of Ceramic Engineering at the Virginia Polytechnic Institute, Blacksburg, Virginia. It was obtained as ground silica--99 per cent minus 200 mesh Tyler and 60 per cent minus ten microns. The amount of collector of the cationic type necessary to float 100 per cent of the material was determined from tests in a conventional cell as being ten pounds per ton of 3037 Aeromine collector and 0.28 pounds per ton of pine oil as a frother in a circuit of pH 7.4 using distilled water to give a slurry density of ten per cent solids by weight.

The Aeromine and pine oil were added to the conditioning tank and conditioned for thirty minutes. The slurry was then pumped into the rotating cell through a spray nozzle to induce a froth on the air-water interface. Spraying the slurry entrained enough air so that a froth could be obtained if the spray had been directed at a fixed surface. Air was not injected into the slurry due to the speed of the centrifuge. When an attempt was made to put a pipe into the slurry, the high speed of the slurry threw the material in all directions. The test gave only qualitative results, as the cell could not be operated on a quanitative basis. At ten revolutions per minute, the cell would give a very small amount of froth, but due to the slope of the slurry in the cell at this speed, a sample of the froth could not be obtained. When the speed was increased so that the air-water interface was vertical, there was no froth at all, and the material sank to the outside of the cell. It was concluded that the application of increased gravity to the collection of minerals has an adverse effect on froth flotation.

High Gravity Separations. Due to the results obtained from the operation of the high-gravity flotation cell, it was theorized that a separation between the wanted mineral and unwanted mineral could be obtained in the froth column by subjecting the froth, from the flotation operation, to artificially induced high gravity. The high gravity could be achieved by placing the froth in a centrifugal field where the settling rates of the fine material could be increased.

One of the mechanisms effecting the poor results obtained with floating fine ores may possibly be due to the very low settling velocity of fine particles of non-floated minerals that become intrapped between the bubbles of the froth column. Various metallic and non-metallic minerals were caused to float by using large amounts of collector. This caused a relatively large amount of the raw ore to float with the idea of first causing the complete flotation of all the material; then separating the wetted minerals from the non-wetted minerals in a centrifuge. The general procedure used was to weigh the finely ground material to be floated on an analytical balance. This material was then added to 300 milliliters of water, together with the various reagents necessary for conditioning the slurry. The slurry was prepared in a Waring Blender, number 700 A, manufactured by the Winstead Hardware Manufacturing Company of Winstead, Connecticut.

The jar on the blender had a 1000-milliliter capacity. The reason that the Waring Blender was chosen for this investigation, instead of a conventional flotation cell, was due to its ability to control conditioning impeller speeds by controlling the voltage applied to the blender; combined with its ability to float small samples of ten to twenty grams. The blender also could be easily cleaned. After flotation, the froth was transferred to a 250-milliliter pyrex bottle from the 1000-milliliter jar of the Waring Blender with an apparatus as shown in Figure 2. After the transfer, the bottle was placed in a centrifuge and centrifuged at a given speed

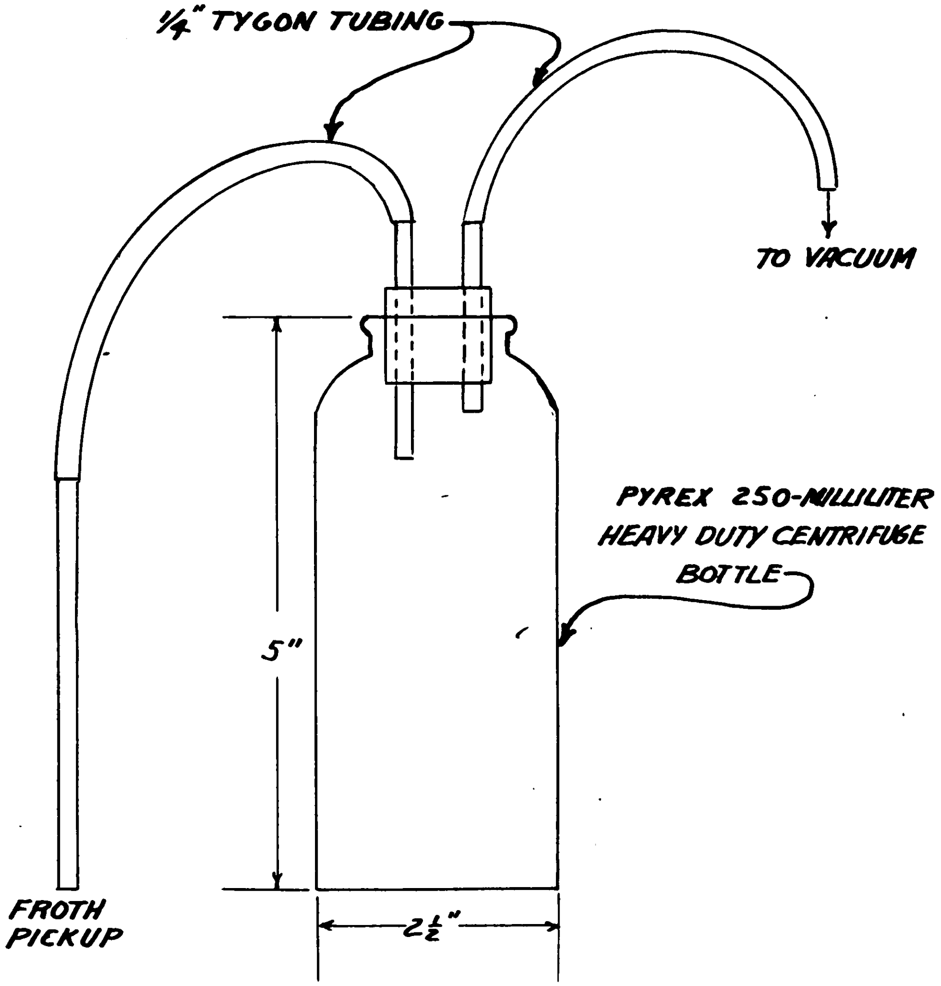

FIGURE 2. FROTH PICK-UP BOTTLE

for a specified time. After the centrifuging, the bottle was removed and the resultant froth was transferred to another 250-milliliter pyrex bottle. The material that had sunk to the bottom of the bottle was labeled as high-gravity sink. The material that had stayed on top of the water in the bottle was labeled as high-gravity float: material that did not float and was left in the Waring Blender jar was titled non-float. The various fractions were either dried in the 250-milliliter pyrex bottles or transferred to 250-milliliter beakers and dried at 202 degrees Fahrenheit in a steam heated drier.

The material in the beakers or bottles were weighed and analyzed. The tests conducted and discussed above were made on a synthetic mixture of copper sulfide and silica (covellite and silica); a zinc ore from Austinville, Virginia; an iron-bearing rock from the Clinton Formation; a semi-anthracite coal obtained from the Plank Mine near Blacksburg, Virginia, and a zinc-bearing material from the Great Gossan Lead Deposit of Carroll County, Virginia.

Test on Copper Sulfide and Silica. For the test, a chemically pure copper sulfide was obtained from the Fisher Scientific Company of Silver Spring, Maryland and ground in a hand mortar to 100 per cent minus 44 microns. Minus ten micron fractions were then obtained by decantation of the silica and the copper sulfide. These fractions were obtained by utilizing Stokes' Law. A two per cent, by weight, mixture of solids and water was placed in a 1000-milliliter graduate. A specified point was marked on the graduate. The time required for a ten micron particle to settle from the top of the mixture in the graduate to the specified point was then calculated by the use of Stokes' Law. After the calculated time for a ten micron particle to settle had elapsed, the material between the top of the slurry and the specified point was decanted. This material was then dried and used for the flotation tests.

The silica used in this test was obtained from the Department of Ceramic Engineering, the Virginia Polytechnic Institute of Blacksburg, Virginia. It was sixty per cent minus ten microns as received, so the minus ten micron fraction was prepared as before by decantation.
One gram of the copper sulfide was mixed with 19.00 grams of the silica to give a synthetic ore containing 3.25 per cent copper. The mixture was placed in the Waring Blender and conditioned for ten minutes at an impeller speed of 11700 revolutions per minute which would correspond to an imput voltage of 70.

After conditioning, the copper sulfide was floated by the addition of pine oil as a frother. The impeller speed was increased to 19000 revolutions per minute, corresponding to a voltage of 120. This high speed gave good mixing of air into the slurry. An excellent froth appeared and was transferred to a 250-milliliter bottle by the froth pick-up bottle apparatus as shown in Figure 2. The bottle was then centrifuged for one minute, and the material that would not go through the froth water interface was collected as the high-gravity float. The high-gravity sink from the centrifuge was then transferred

TABLE I

Data for Synthetic Mixture of Silica and Copper Sulfide

Amount of synthetic ore (gm)	20.00	20.00	20.00	20.00
Amount of water (ml)	300	300	300	300
Amount of collector (lb/ton) [a]	2.00	2.00	2.00	2.00
Amount of frother (lb/ton) [b]	2.47	2.47	2.47	2.47
Amount of NH_4OH	As required	As required	As required	As required
Conditioning time (min)	10	10	10	10
pH	11.0	10.7	11.0	11.0
Conditioning voltage for speed control [c]	70	70	80	70
Flotation time (min)	1	1	1	1
Number of times floated [d]	2	2	2	2
Flotation voltage for speed control	120	120	120	120
Speed of centrifuge (rpm)	500	1000	1500	1850
$G_c$ravity	42.6	165.1	383.0	580.0
Time in centrifuge (min)	1	1	1	1
(1) Grams of non-float	16.92	16.95	16.20	16.00
(2) Grams of high-gravity sink	2.23	2.35	2.95	2.50
(3) Grams of high-gravity float	0.60	0.46	0.50	0.19
(1) % of copper in non-float	0.79	0.85	0.79	0.89
(2) % of copper in high-gravity sink	5.65	6.95	8.30	11.50
(3) % of copper in high-gravity float	57.20	55.50	55.10	57.50
(1) Wt. % of copper in non-float	22.20	25.6	20.5	26.4
(2) Wt. % of copper in high-gravity sink	20.80	29.0	37.4	53.1
(3) Wt. % of copper in high-gravity float	57.00	45.4	42.1	20.5
% of copper in float (calculated)	17.95	14.97	15.09	18.51
% of copper in the ore	3.25			

[a] Sodium Aerofloat:

[b] General Naval Stores, Number 5 pine oil

[c] 70 volts equals 11700 rpm: 80 volts equals 13000 rpm: 120 volts equals 19000 rpm.

[d] Second float--15 seconds

[e] Average--16.63

Note: Numbers found in parentheses refer to Figure 3.

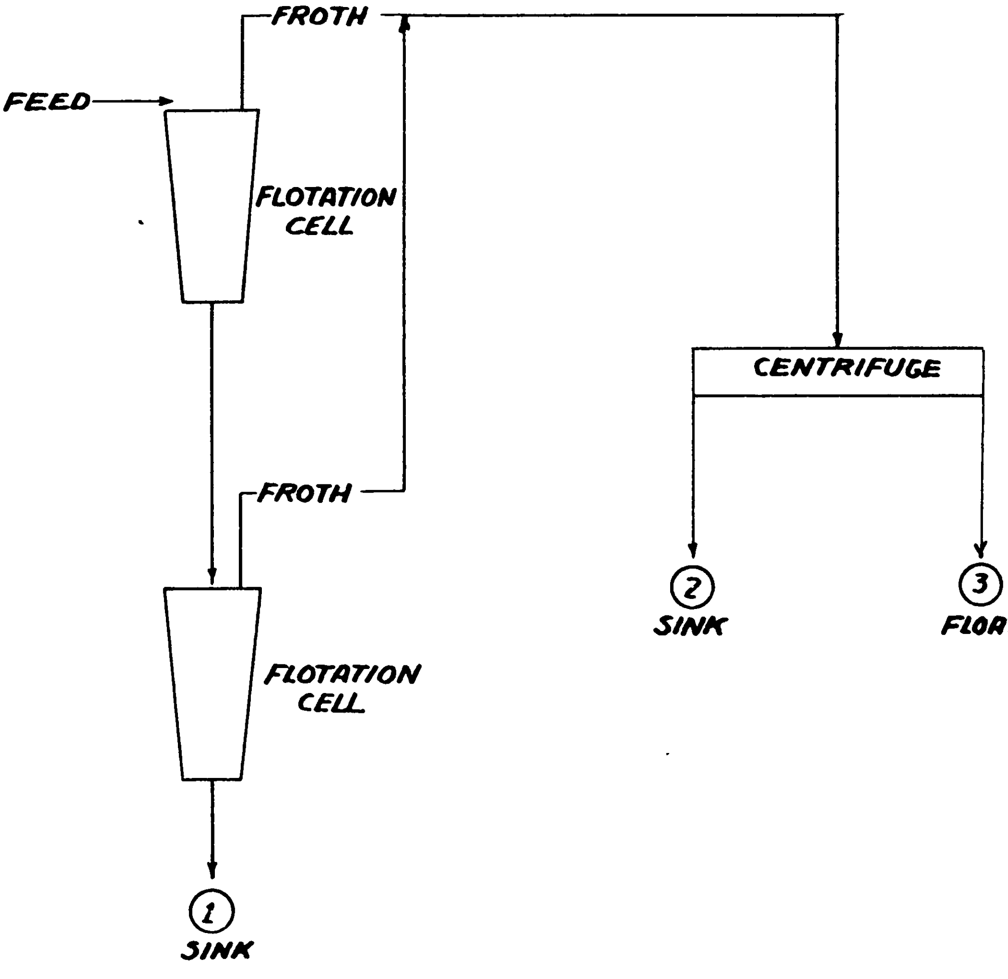

FIGURE 3. FLOW SHEET OF HIGH-GRAVITY FLOTATION

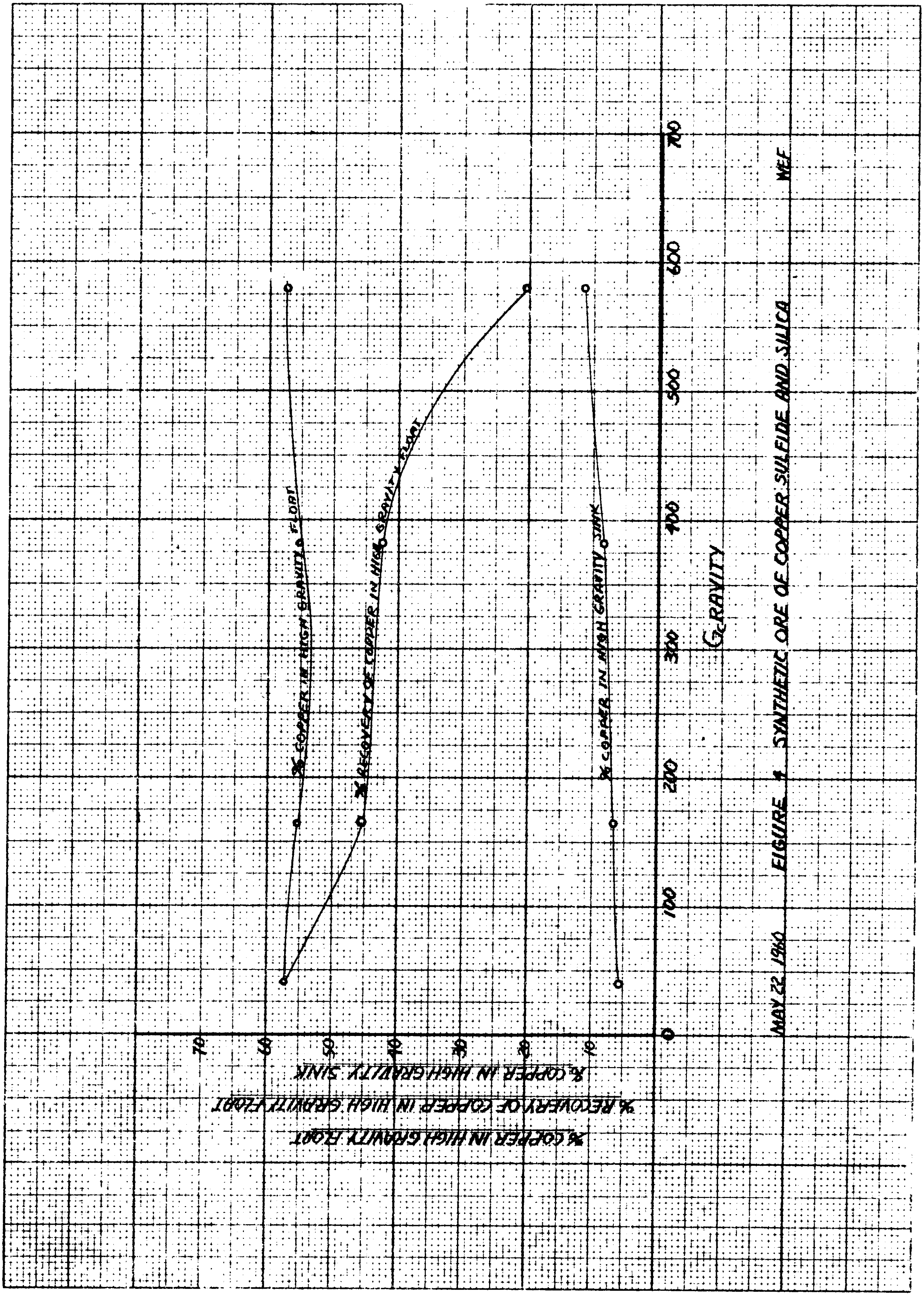

FIGURE 4 SYNTHETIC ORE OF COPPER SULFIDE AND SILICA

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to a beaker and all of the products were dried at 202 degrees Fahrenheit in the drier. After drying, the samples were weighed and analyzed for copper content. See Methods of Analyses, Figure 3 and 4, and Table I for the results of these tests. Figure 4 is a graph of gravity versus per cent of copper in high-gravity float; the recovery of copper in high-gravity float and the per cent copper in high-gravity sink. The numbers of the various products as shown on Figure 3 also appear in Table I.

Test on Austinville Zinc Ore. Tests were made on a natural zinc ore from Austinville, Virginia. The material was crushed in a jaw crusher to minus one-half inch; then repassed ten times through a hammer mill, type U. P. R., number 9 x 9 manufactured by the American Pulverisor Company of Saint Louis, Missouri until the particle size was 100 per cent minus 100 mesh Tyler. The ore was then passed through an one-inch Reductionizer. The Reductionizer is an air attrition mill that uses air to grind the material on itself in a turbulent air stream. It is manufactured by the Reduction Engineering Corporation, 184 Blanchard Street, Newark, New Jersey. After five passes through the mill, the ore as shown by a size analysis would not pass the minus ten micron average size specification that had been set.

The material was then rerun through the Reductionizer at a pressure of fifty pounds per square inch gage to separate the fines from the coarser material and obtain a final product. The Reductionizer had a one-inch cyclone separater with a bag. The cyclone and the bag were attached at the outlet of the mill to obtain the product of the Reductionizer. The fines were collected in the bag and utilized for the flotation tests of the Austinville zinc ore. All materials were ground in the Reductionizer at a pressure of 105 pounds per square inch gage. The method of size analyses used was described by Krumbein and Pettijohn.(17) A sample in between six and seven grams was weighed on an analytical balance including 0.1000 grams of Aero Depressant 615, manufactured by the American Cyanamid Company, New York, New York. The sample containing the Aero Depressant was mixed to form a suspension of 550 milliliters using distilled water in an Andreasen sedimentation pipette obtained from the Fisher Scientific Company of Silver Spring, Maryland. The suspension was sampled at the initial density by taking a ten milliliter sample. Sedimentation times were calculated for ten, 5.8, and 0.58 microns. The ten milliliter samples in tared beakers were placed in the drier until dry, removed and cooled, and weighed on an analytical balance. The sample was one-hundred per cent finer than ten microns and 43.2 per cent finer than 0.58 microns. See Table II.

TABLE II

Austinville Zinc Ore Size Consist

Size (in microns)	Per Cent
+10	0.0
-10 + 5.8	16.4
-5.8 + 0.58	40.4
-0.58	43.2
	100.0

Samples were taken of the zinc ore with a glass tube sampler and weighed to 20.00 grams on an analytical balance. Solutions were made of Aerofloat 211 and pine oil, manufactured by the American Cyanamid Company. A 20.00 gram sample of zinc ore was placed in the Waring Blender and conditioned with four pounds per ton of Aerofloat 211. Sodium carbonate and Hydrochloric acid were used to adjust the pH to 9.8.(20) The slurry was then conditioned for ten minutes at 11700 revolutions per minute, corresponding to an imput to the blender of 70 volts. The speed in the blender was then raised by increasing the voltage to 120 and adding pine oil. The ore was floated twice, and the two floats were combined in one 200-milliliter bottle. The bottle containing the froth from the flotation operation was then centrifuged at a selected speed in the laboratory centrifuge for a pre-set time. The final high-gravity float, sink, and non-float were transferred to 250-milliliter tared beakers, dried and cooled, and weighed. The various products were analyzed for zinc as shown under the Methods of Analyses. See Table III for results and Figure 5. Figure 5 is a graph showing gravity versus per cent zinc in high-gravity float, and per cent zinc in high-gravity sink. The numbers of the various products as shown on Figure 3 also appear on Table III.

<u>Test on Clinton Iron Formation</u>. Tests were conducted on a low-grade iron-bearing rock from the Clinton iron formation. The material was reduced in a laboratory jaw crusher to minus one-half inch. It was crushed to minus ten mesh Tyler in a laboratory roll, number 1139,

TABLE III

Data for Austinville Zinc Ore

	Test 1	Test 2	Test 3	Test 4
Amount of Austinville zinc ore (gm)	20.00	20.00	20.00	20.00
Amount of water (ml)	300	300	300	300
Amount of collector (lb/ton)[a]	4.00	4.00	4.00	4.00
Amount of frother (lb/ton)[b]	2.47	2.47	2.47	2.47
$NaCO_2$ and HCL	AS REQUIRED		AS REQUIRED	
$CuSO_4$ (lb/ton)	4.27	4.27	4.27	4.27
Conditioning time (min)	10	10	10	10
pH	10.0	9.5	9.5	9.8
Conditioning volts for speed control	80	80	80	80
Flotation time (min)	2	2	2	2
Number of times floated[c]	2	2	2	2
Flotation voltage for speed control[d]	120	120	120	120
Speed of centrifuge (rpm)	500	1000	1500	2000
Gravity	42.4	169.4	382.0	678.0
Time in centrifuge (min)	1	1	1	1
(1) Grams of non-float	5.90	7.77	15.27	9.36
(2) Grams of high-gravity sink	7.89	8.88	3.68	9.02
(3) Grams of high-gravity float	6.10	3.31	0.41	1.24
(1) % of zinc in non-float	0.62	2.18	2.64	1.70
(2) % of zinc in high-gravity sink	4.66	5.59	16.15	4.70
(3) % of zinc in high-gravity float	12.45	20.70	32.50	27.30
(1) Wt. % of zinc in non-float	3.10	12.60	35.50	17.25
(2) Wt. % of zinc in high-gravity sink	31.70	36.70	52.60	45.90
(3) Wt. % of zinc in high-gravity float	65.20	50.70	11.80	36.75
% of zinc in float (calculated)[e]	8.07	9.72	17.81	8.40
% of zinc in heads (calculated)[f]	5.80	6.75	5.85	4.70
% of zinc in heads	5.55			

[a] Aerofloat 211

[b] General Naval Stores, Number 5 pine oil

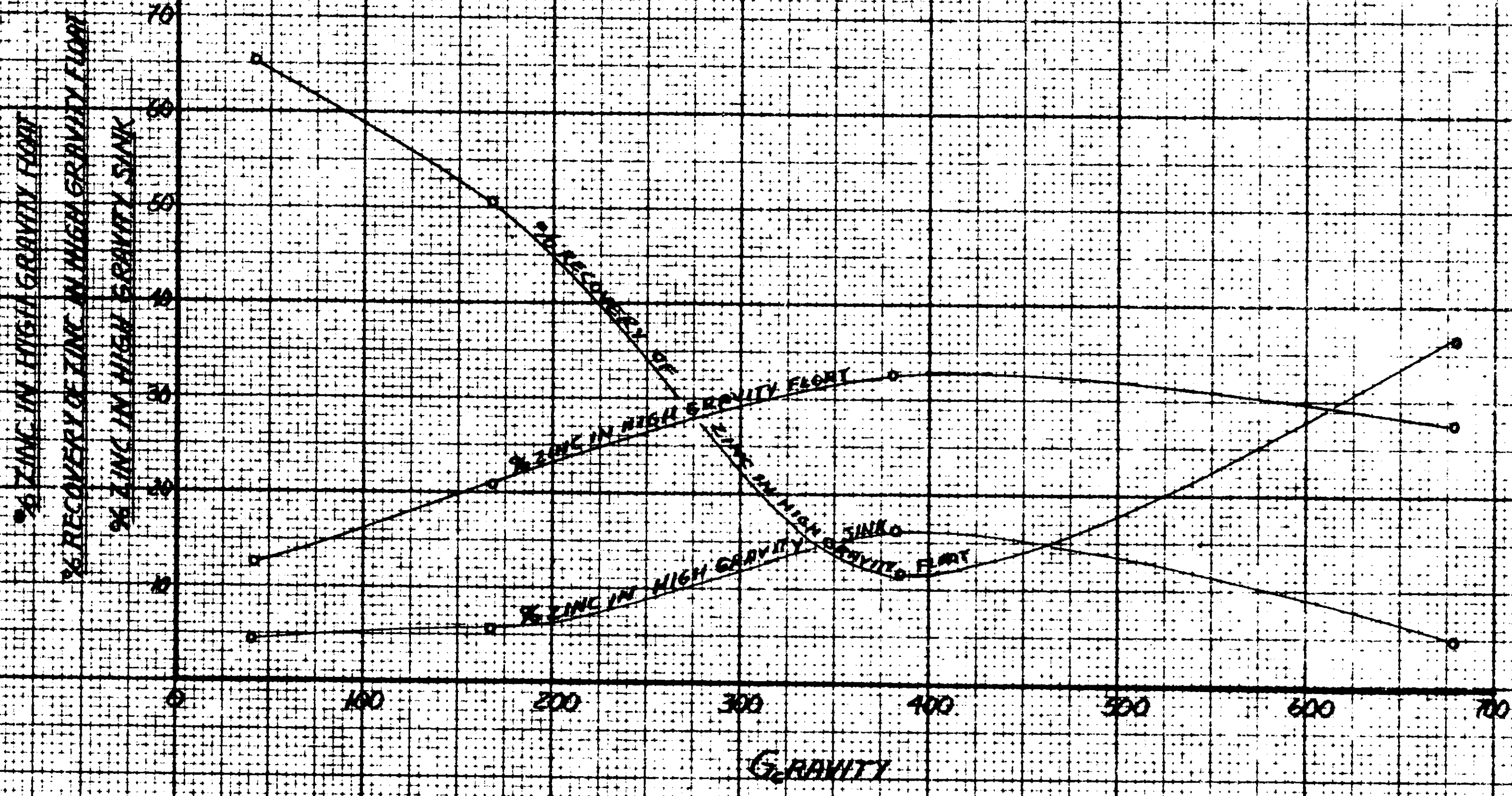
[c] Second float--15 seconds

[d] 70 volts equals 11700 rpm: 80 volts equals 13000 rpm: 120 volts equals 19000 rpm.

[e] Average--11.01

[f] Average--5.78

Note: Numbers found in parentheses refer to Figure 3.



MAY 22 1960 FIGURE 5 AUSTINVILLE ZINC ORE WEF manufactured by the Sturtevent Manufacturing Company of Boston, Massachusetts. The crushed material was then placed in an eight-inch ball mill with alundum balls and ground for thirty-six hours to minus 325 mesh Tyler. This operation was done dry. The flotation of this material was conducted as described under the Austinville zinc ore except that different reagents were used, and the flow sheet was more complex. For results see Figures 6 and 7 and Tables V and VI. The iron analyses were run using the method shown in the Methods of Analyses.

TABLE IV

Clinton Iron Formation Size Consist

Size (in microns)	Per Cent
+ 20	23.4
- 20 + 10	20.1
- 10 + 5	14.0
- 5	42.5
	100.0

Test on Semi-anthracite Coal. Tests were conducted on a semi-anthracite coal obtained from the Plank Mine which is operating in the Big Seam five miles from Blacksburg, Virginia. The coal was reduced to approximately 75 per cent minus ten microns. See Table VII. This was done by crushing the coal to minus one-half inch in the laboratory jaw crusher. It was then repassed through the hammer mill until it was minus 100 mesh Tyler. It was passed through the Reductionizer five times and sampled to ascertain its fineness by a sedimentation analyses.(17) Due to the coal's light specific gravity, it was reasoned

TABLE V

Data for Clinton Iron Ore (1)

Amount of Clinton Iron Ore (gm)	20.0
Amount of water (ml)	300
Amount of Collector (lb/ton)[a]	12.5
Amount of frother (lb/ton)[b]	4.94
HCL	As required
NH_4OH	As required
Conditioning time (min)	10
pH[c]	1.8 - 10.2
Conditioning volts for speed control	70
Flotation time (min)	1
Number of times floated[d]	5
Flotation voltage for speed control	120
Speed in centrifuge (rpm)	1000
$G_c$ravity	169.4
Time in centrifuge (min)	2
(1) Grams of non-float	2.96
(2) Grams of high-gravity sink	10.95
(3) Grams of final high-gravity sink	4.96
(4) Grams of final non-float	1.58
(5) Grams of final high-gravity float	0.43
(1) % of iron in non-float	7.50
(2) % of iron in high-gravity sink	16.50
(3) % of iron in final high-gravity sink	27.30
(4) % of iron in final high-gravity float	33.70
(5) % of iron in non-float	11.30
(1) Wt. % of iron in non-float	5.57
(2) Wt. % of iron in high-gravity float	45.83
(3) Wt. % of iron in high-gravity sink	34.17
(4) Wt. % of iron in final high-gravity float	13.16
(5) Wt. % of iron in final non-float	1.27
% of iron in heads	19.75

[a] 801: American Cyanamid Company

[b] General Naval Stores, Number 5 pine oil

[c] 1.8 -- conditioning: 10.2 -- flotation

[d] One minute each time

Note: Numbers found in parentheses refer to Figure 6.

TABLE VI

Data for Clinton Iron Ore (2)

Amount of Clinton iron ore (gm)	20.00
Amount of water (ml)	300
Amount of collectors (lb/ton) [a] [b]	5.58 - 19.6
Amount of frother (lb/ton) [c]	2.47
H_2SO_4	11.0
NH_4OH	As required
Conditioning time (min)	10
pH	3.0
Conditioning voltage for speed control [d]	70
Flotation time (min)	1
pH	10.0
Number of times floated [e]	3
Flotation voltage for speed control	120
Speed of centrifuge (rpm)	1000,1500,500
$G_c$ravity	169.4, 382.0 42.5
Time in centrifuge (min)	2
(1) Grams in first non-float	0.86
(2) Grams in first high-gravity sink (calculated)	10.69
(3) Grams in first high-gravity float non-sink	0.28
(4) Grams in second float high-gravity sink	0.68
(5) Grams in second float high-gravity float	4.34
(6) Grams in first high-gravity sink non-float	1.62
(7) Grams in first high-gravity sink, float, high-gravity sink	9.62
(8) Grams in first high-gravity sink, float, high-gravity float	1.07
(1) % of iron in first non-float	10.10
(2) % of iron in first high-gravity sink (calculated)	24.20
(3) % of iron in first high-gravity float, non-float	11.10
(4) % of iron in second float, high-gravity sink	18.20
(5) % of iron in second float, high-gravity float	30.10
(6) % of iron in first high-gravity sink, non-float	8.30
(7) % of iron in first high-gravity sink, float, high-gravity sink	26.90
(8) % of iron in first high-gravity sink, float high-gravity float	25.50
(1) Wt. % of iron in first non-float	1.87
(2) Wt. % of iron in first high-gravity sink (calculated)	63.78
(3) Wt. % of iron in first high-gravity float non-sink	0.67
(4) Wt. % of iron in second float, high-gravity sink	2.67
(5) Wt. % of iron in second float, high-gravity float	28.21
(6) Wt. % of iron in second float, high-gravity non-float	2.88
(7) Wt. % of iron in first high-gravity sink, float, high-gravity sink	57.83
(8) Wt. % of iron in first high-gravity sink, float, high-gravity float	5.85
% of iron in heads	22.20

[a] American Cyanamid Company, 801

[b] Number 2 fuel oil

[c] General Naval Stores pine oil

[d] 70 volts equals 11700 rpm: 120 volts equals 19000 rpm.

[e] One minute each time

Note: Numbers in parentheses refer to Figure 7.

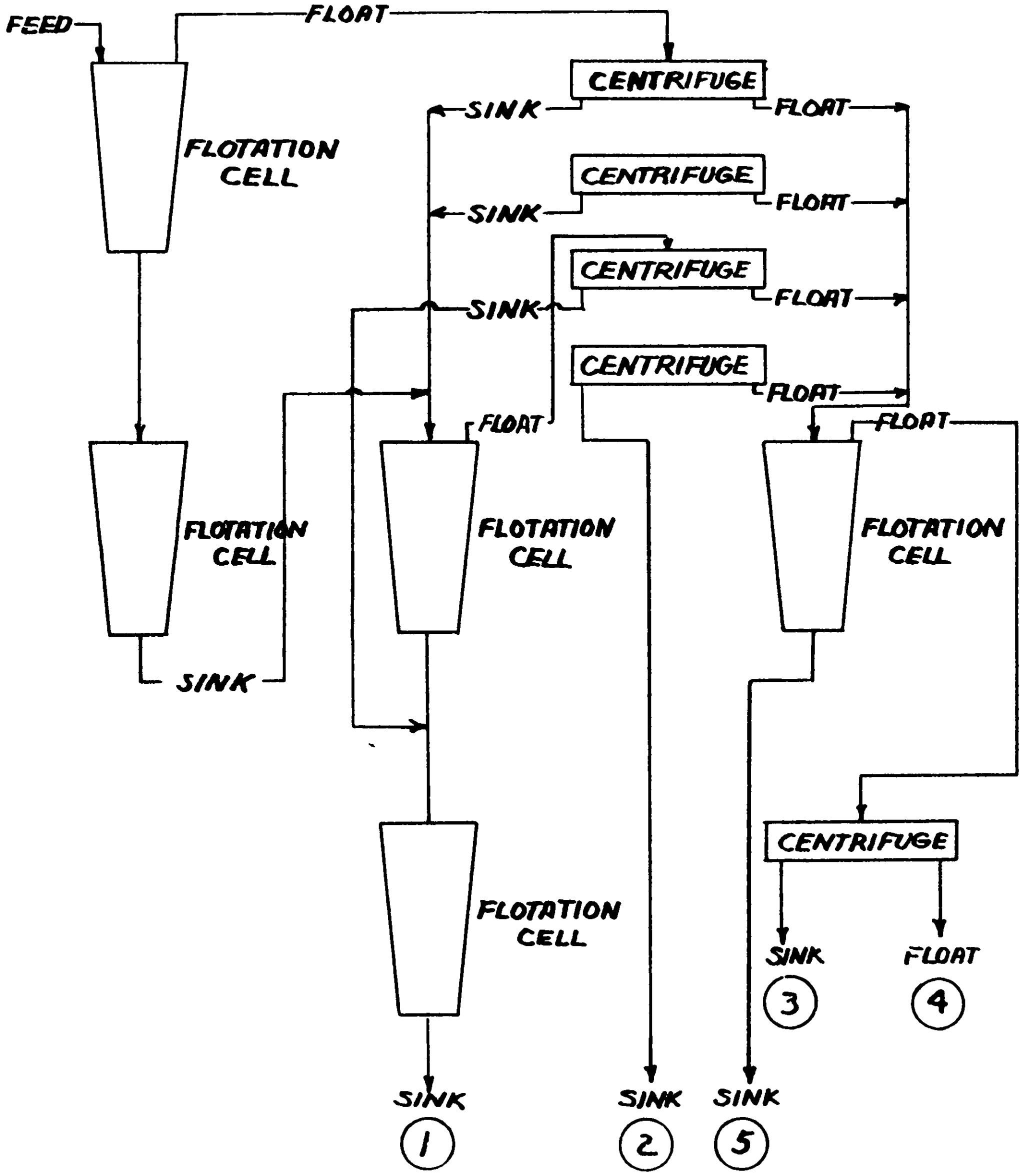


FIGURE 6. FLOW SHEET OF HIGH-GRAVITY IRON ORE SEPARATION. SEE TABLE V.

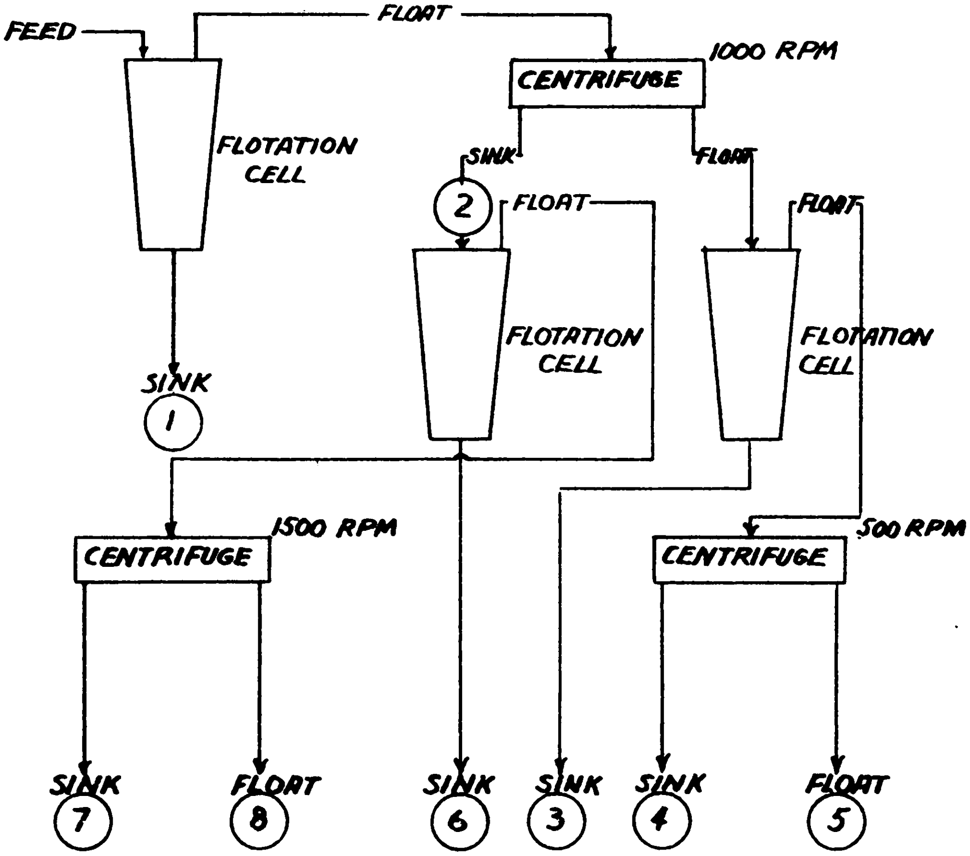


FIGURE 7. FLOW SHEET OF HIGH GRAVITY IRON ORE FLOTATION. SEE TABLE V1.

that the 75 per cent minus ten micron coal would correspond to a 100 per cent minus ten micron sulfide ore. Ten gram samples were used, and the material was floated as described for the zinc ore. See Table VIII and Figures 3 and 8. Figure 8 is a graph showing gravity versus per cent ash in high-gravity float and recovery of coal in high-gravity float.

TABLE VII

Plank Mine Coal Size Consist

Size (in microns)	Per Cent
+ 10	24.4
- 10 + 5	26.7
- 5	48.9
	100.0

The reagents used for the flotation of this coal were suggested by Mr. M. P. Corriveau of the Clinchfield Coal Company of Dante, Virginia. The samples of non-float, high-gravity coal, and float were dried in the laboratory drier at 202 degrees Fahrenheit, weighed, and analyzed for ash content according to the method approved by the American Society for Testing Materials. See Methods of Analyses.

Test on Great Gossan Lead Ore. The Gossan Lead ore is a low-grade deposit of pyrrhotite found in Carroll County, Virginia. Associated with the pyrrhotite are small percentages of zinc, lead, and copper sulfides. The material available for study was minus one-half inch in size and was reduced by the laboratory rolls to about ten mesh Tyler.

TABLE VIII

Semi-anthracite Coal

	Test 1	Test 2	Test 3	Test 4	Test 5	Test 6	Test 7	Test 8	Test 9	Test 10
Amount of coal (gm)	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00
Amount of water (ml)	300	300	300	300	300	300	300	300	300	300
Amount of depressant (lb/ton)	0.054	0.054	0.054	0.054	0.054	0.054	0.054	0.054	0.054	0.054
Methyl amyl alcohol (lb/ton)	1.6	1.6	1.6	1.6	1.6	3.2	3.2	3.2	3.2	3.2
Conditioning time (min)	1	1	1	1	1	1	1	1	1	1
Conditioning voltage for speed control	70	70	70	70	70	70	70	70	70	70
Flotation time (sec)	30	30	30	30	30	30	30	30	30	30
Flotation voltage for speed control[a]	120	120	120	120	120	120	120	120	120	120
pH	7.4	7.4	7.4	7.4	7.4	7.4	7.4	7.4	7.4	7.4
Number of times floated	2	2	2	2	2	2	2	2	2	2
Speed of centrifuge (rpm)	200	400	800	1000	2000	200	400	800	1600	2000
$G_c$ravity	6.8	27.2	108.9	436.0	681.0	6.8	27.2	108.9	436.0	681.0
Time in centrifuge (min)	1	1	1	1	1	1	1	1	1	1
(1) Grams in non-float	3.38	4.24	4.50	4.61	5.07	2.78	3.00	3.03	3.00	3.02
Grams in float (calculated)	6.53	5.30	5.16	5.30	4.69	6.87	6.67	6.81	6.37	5.78
(2) Grams in high-gravity sink	4.93	4.21	4.13	4.33	3.94	4.39	4.30	4.27	4.13	3.36
(3) Grams in high-gravity float	1.60	1.09	1.09	0.97	0.75	2.48	2.37	2.54	2.24	2.42
(1) % of ash in non-float	36.15	32.38	35.29	34.05	32.73	36.46	37.96	39.10	36.66	37.33
% of ash in float (calculated)[b]	24.15	22.90	23.45	22.95	22.15	23.21	22.95	23.45	22.35	21.90
(2) % of ash in high-gravity sink	25.83	24.46	25.29	24.53	23.51	26.10	25.51	26.24	24.82	24.86
(3) % of ash in high-gravity float	19.02	16.78	16.21	15.76	15.03	18.26	18.41	18.49	17.75	17.84
(1) Wt. % of ash in non-float	28.0	52.1	39.4	39.2	38.1	24.2	27.1	27.2	27.6	30.6
Wt. % of ash in float (calculated)	25.5	46.8	30.0	30.5	38.8	37.9	36.4	36.5	36.4	35.0
(2) Wt. % of ash in high-gravity sink	28.9	29.2	26.1	26.5	35.7	27.2	26.1	25.6	25.8	22.8
(3) Wt. % of ash in high-gravity float	7.6	6.9	4.5	3.8	4.1	10.7	10.4	10.7	10.2	11.6
Calculated heads	28.75	27.07	29.19	28.08	27.67	26.85	27.39	29.63	28.77	30.60
(4) Wt. % of a high-gravity float	16.15	12.88	11.71	9.78	7.68	25.75	24.50	25.82	23.90	27.50

[a] 70 volts equals 11700 rpm: 80 volts equals 13000 rpm: 120 volts equals 19000 rpm.

[b] Average--22.95

Note: Numbers in parentheses refer to Figure 8.

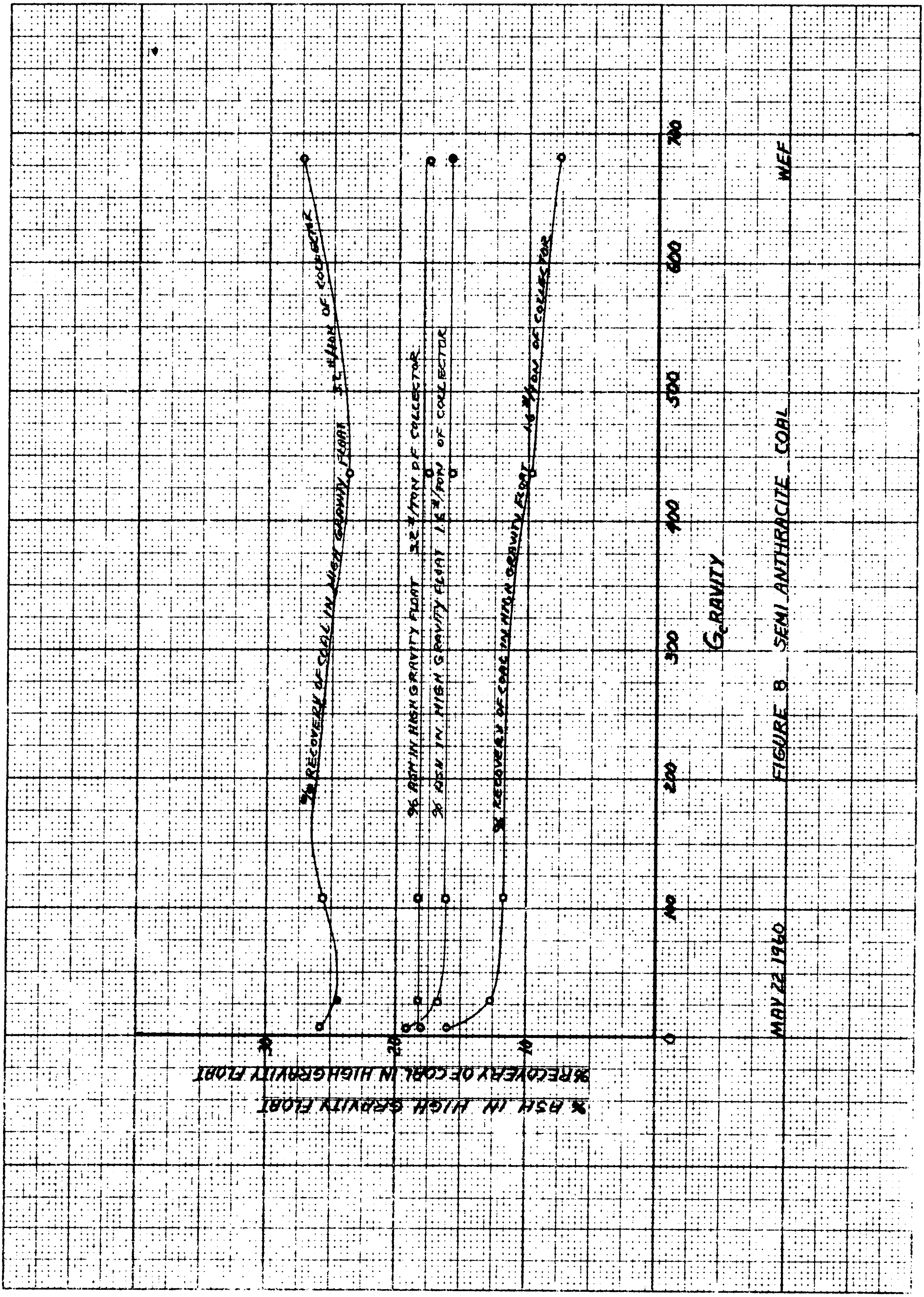


FIGURE 8 SEMI ANTHRACITE COAL

It was then ground to pass 100 mesh Tyler in a McCool six and one-half inch disc pulverizer manufactured by the Mine and Smelter Supply Company of Denver, Colorado. Some difficulty was experienced in grinding this material to pass ten microns. See Table IX for sedimentation analyses.

TABLE IX

Great Gossan Lead Ore Size Consist

Size (in microns)	Per Cent
+ 10	4.4
- 10 + 5	67.4
- 5 + 2	20.4
- 2	7.8
	100.0

Approximately thirty passes were required to achieve minus ten microns. After passing through the Reductionizer, the material had a noticeable oily smell which was interpretated as coming from the compresser. The machine was a two-cylinder, class WL compresser manufactured by the Sullivan Machinery Company of Michigan City, Indiana. The purpose of this phase of the investigation was to depress the zinc and float the pyrrhotite. Reagents and conditions were chosen as suggested by Corriveau(4) to depress the zinc and float the other sulfides. For results see Table X and Figures 3 and 9. Numbers on Figure 3 correspond to the numbers given in Table X. Figure 9 is a graph of gravity versus per cent zinc in high-gravity float, the recovery of zinc in high-gravity float, and the per cent zinc in high-gravity sink.

TABLE X

Data for the Great Gossan Lead Ore

	Test 1	Test 2	Test 3	Test 4	Test 5
Amount of Gossan Lead Ore (gm)	10.0	10.0	10.0	10.0	10.0
Amount of water (ml)	300	300	300	300	300
Amount of frother (lb/ton)[a]	0.5	0.5	0.5	0.5	0.5
Amount of depressant (lb/ton)[b]	1.0	1.0	1.0	1.0	1.0
Amount of collector (lb/ton)[c]	1.2	1.2	1.2	1.2	1.2
Amount of conditioner (lb/ton)[d]	0.5	0.5	0.5	0.5	0.5
pH	5.8	5.8	5.8	5.8	5.8
Conditioning time (min)	2	2	2	2	2
Conditioning voltage for speed control	70	70	70	70	70
Flotation time (min)	1	1	1	1	1
Flotation voltage for speed control[e]	120	120	120	120	120
Number of times floated	2	2	2	2	2
Speed of centrifuge (rpm)	200	400	800	1600	2000
$G_c$ravity	6.8	27.2	109.8	436.0	681.0
Time in centrifuge (min)	1	1	1	1	1
(1) Grams in non-float	0.36	0.37	0.36	0.47	0.51
(2) Grams in high-gravity sink	0.85	1.97	1.54	2.52	6.28
(3) Grams in high-gravity float	7.93	6.87	7.21	6.30	2.95
(1) % of zinc in non-float	4.11	1.94	2.32	6.90	3.05
(2) % of zinc in high-gravity sink	3.48	3.73	3.77	2.45	3.87
(3) % of zinc in high-gravity float	3.04	3.73	3.27	3.16	4.21
(1) Wt. % of zinc in non-float	5.2	2.1	2.8	11.1	4.1
(2) Wt. % of zinc in high-gravity sink	10.3	21.7	19.3	21.2	63.5
(3) Wt. % of zinc in high-gravity float	84.5	76.2	77.9	67.7	32.4
Calculated % of zinc in float	3.09	3.67	4.49	2.94	3.98

[a] Aerofroth 77: American Cyanamid Company

[b] Aerodepressant 645: American Cyanamid Company

[c] Reagent 325: American Cyanamid Company

[d] NaS_1O_2

[e] 70 volts equals 11700 rpm: 80 volts equals 13000 rpm: 120 volts equals 19000 rpm.

Note: Numbers in parentheses refer to Figure 9.

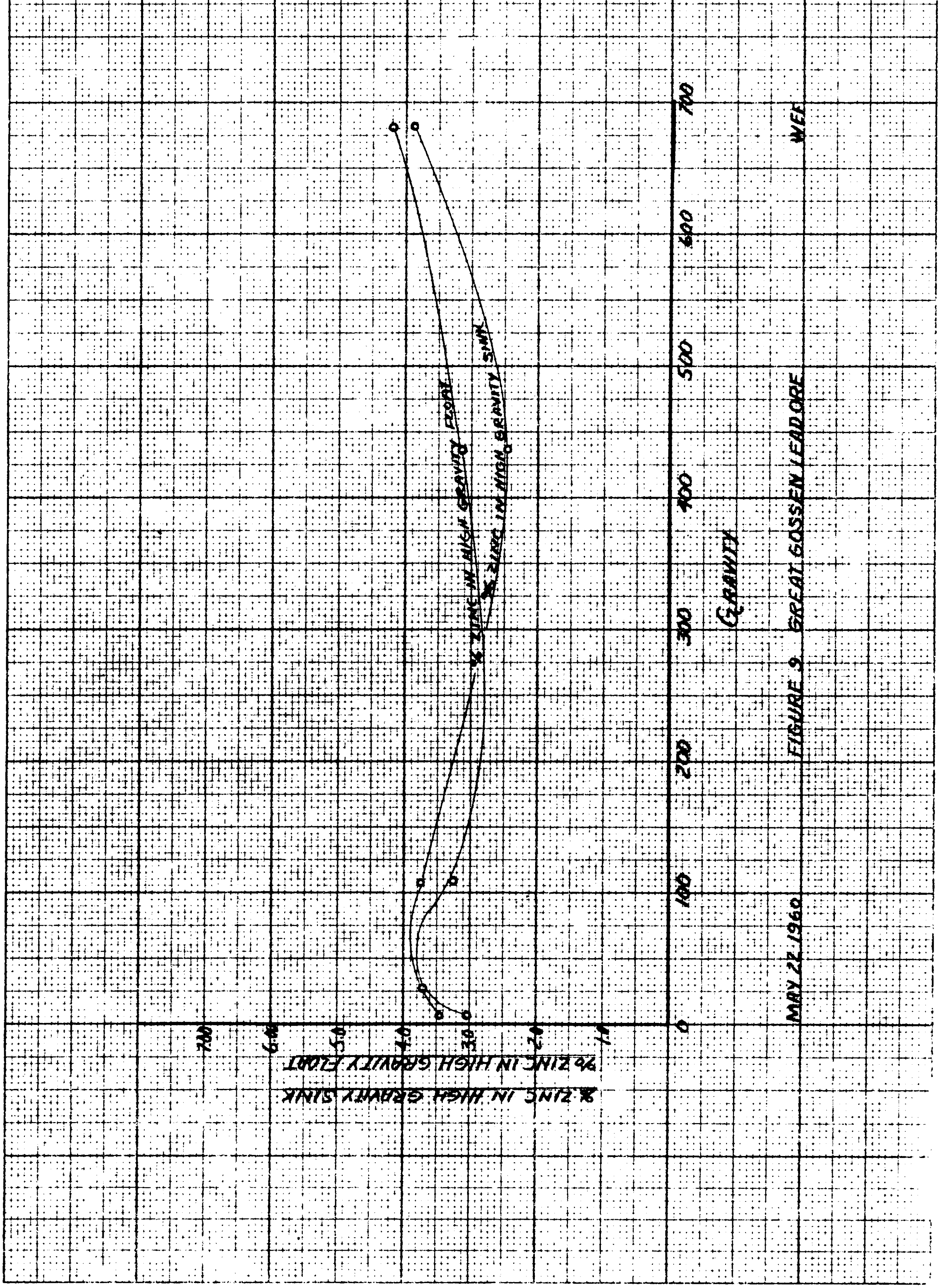


FIGURE 9 GREAT GOSSEN LEAD ORE

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DISCUSSION

Conventional flotation practice has been very successful for the flotation of relatively coarse materials. The method investigated by this thesis would seem to show possibilities for the flotation of fines. The test work that was done during this investigation has indicated that increasing the settling rates of fines has a beneficial effect on the grade of the final product. The test work done on operating a flotation cell in a centrifugal field was a failure in one respect, but it also indicated that the flotation of minerals was hindered by application of increased gravities to the system.

A segment of the study of the flotation of fines was concerned with the collection and separation of the materials by application of centrifugal force to the froth and showed that if conditions were correct for flotation of the material, relatively high upgrading of the flotation froth could be achieved by increasing the settling rates in the froth column. Various materials were tested, such as a synthetic mixture of copper sulfide and silica. Copper sulfide and silica were chosen, because each of the compounds were pure. The results of this test were very outstanding, for they gave good recoveries with an appreciable increase of copper in the final product. These tests, in comparison with the other materials tested, gave the best results achieved during the investigation. The success of these tests may have been due to the copper sulfide and silica not becoming mutually coated with ions given off by their respective surfaces.

Another material tested was a zinc sulfide ore from Austinville, Virginia that gave good results with respect to the grade of zinc in the initial float versus the high-gravity float. Although the grade in the high-gravity float was high, the recovery was low. This may have been due to the mutual coating of sulfides and gangue which would result with a finely ground ore, because the materials were ground together to produce the minus ten micron product. Various combinations of reagents and conditioning variables were not investigated for this particular study. It may be that a more rigerous investigation would materially increase the grade and recovery of this ore by high gravity. The percentage of initial float showed rather wide variations in the zinc content. These wide variations may have been due to very minute changes in the chemistry and electrical effect at the interfaces of these finely ground minerals. It may be noted that the highest grade float gave the highest grade final product with the lowest recovery of zinc. As the amount of applied gravity was increased, the general trend for this zinc ore was the appearance of a higher grade product in the froth that remained on the froth water interface.

The next set of tests conducted were on a low-grade iron ore from the Clinton iron ore formation. These tests were designed to obtain a high-grade product by the use of the high-gravity flotation method. The results of these tests indicated that, although the material can be floated, a high-grade product was not achieved. The test referred to in Table V was designed to duplicate actual flotation practice by retreating the flotation product a number of times so that a higher grade material could be achieved. To do this, the flow sheet became complicated, as can be seen in Figure 6. Both primary floats were centrifuged a number of times, as well as the iron from the primary flotation stage. It was expected that this test would show a high-grade final product with a high recovery of iron. Both the upgrading, from approximately twenty to thirty-four per cent, and the recovery of thirteen per cent could not be considered to be successful. This ore was then tested to determine what would happen to the centrifuge sink if it were refloated and centrifuged again. A test was designed to check the possibility of refloating the centrifuged sink. This test differed from all other tests in this investigation in that the centrifuge speeds were varied so that the sink from the first centrifuge, which had been refloated, would be subjected to a higher centrifugal force. The float from the first centrifuge was then refloated and centrifuged again. For results see Table VI and Figure 7, a flow sheet of the procedure. The data indicates that the material, once having been sunk in the centrifuge, if refloated and centrifuged again will not result in a separation of iron in the float and silica in the sink.

A semi-anthracite coal was utilized to investigate the effect of the high-gravity flotation method. Tests were conducted at two levels of reagent addition and indicated that recoveries could be substantially increased by increasing the amount of collector. Concurrent with the increased recovery was an increased amount of ash, but the amount of ash in the float, although somewhat higher, was not as great as the increase in the recovery of coal. The flotation heads were approximately twenty-eight per cent ash. By a combination of flotation and high-gravity separation, the ash can be reduced to approximately nineteen per cent: a moderate reduction in ash content. Again, the results of this test are indicative of high-gravity flotation having beneficial results on the final product and indicates that if further research were conducted, appreciably better results may be obtained.

The Great Gossan Lead ore is a mineral deposit characterized by its extremely fine intergrowth of zinc minerals with pyrrhotite that requires extremely fine grinding to liberate the zinc from the iron minerals. The test conducted on this ore was designed to depress the zinc minerals, and float the iron minerals. It was expected that the zinc would be found both in the non-float and in the high-gravity sink. The results of the test were disappointing, because they gave no differential separation of zinc from the iron minerals as was expected. It was theorized that this may have been due to the surfaces of the zinc materials becoming activated from the oil ladden air supplied by the compresser. The material required thirty passes through the Reductionizer to grind the ore to minus ten microns.

It could be suggested that some precautions be taken in the future to insure that the oil would not contaminate the ore, or that some other method of grinding could be chosen to reduce the size of this material. The results of this test were not conclusive because of the possible pre-activation of the zinc material surface with oil from the compresser.

Some success was achieved by collecting the mineral and then subjecting the resultant froth to artificially induced high gravity by the use of a centrifuge. Some of the tests showed relatively good results, particularly the synthetic copper ore, the Austinville ore, and the semi-anthracite coal. Those tests in which relatively poor results were achieved were made on the iron ore and the Gossan Lead zinc material. It is proposed that the poor results obtained with the above ores was not wholly due to the failure of method, but the choice of reagents or pre-activation of the surface. As can be seen from the curves of the copper sulfide ore in Figure 4, at relatively low gravities the recovery and grade of copper in the float is at a maximum. It would appear from this, and the study of Figure 8 of the semi-anthracite coal, that the amount of reagents control the percentage of wanted mineral and the float to a very large extent.

Contrasting the percentage ash in the high-gravity float of Figure 8, between 3.2 and 1.6 pounds per ton levels, it can be seen that at very low gravities with a larger amount of collector, the amount of gravity applied has very little effect on the curve. For the high-gravity flotation of ash a straight line occurs from the use of the larger amount of collector. It may also be noted that the per cent recovery goes up 250 per cent by doubling the amount of collector. The float normally, of the extremely fine ground Austinville zinc ore, would be approximately nine per cent with a head sample of 5.55 per cent. This could be increased to a maximum of approximately thirty-two per cent by the application of high-gravity flotation. The recovery of zinc at this relatively high grade would be a comparatively low twelve per cent. The recovery could be increased by proper choice of reagents or conditioning if the tests on the semi-anthracite coal are any indication of increasing recovery by increasing the reagents.

The tests indicated that the low settling rates of very fine particles on non-floated minerals are a contributing cause to the poor results achieved with slime flotation. The tests on the iron ore indicated that even with repeated recleaning of the float and the sink, using the high-gravity flotation method, poor results were obtained. The tests on the iron ore indicated that if the reagent choice is incorrect, or conditions are not conducive to flotation, poor results will be obtained. The results on the synthetic ore of copper, the Austinville zinc ore, and the semi-anthracite coal indicate that the flotation of extreme fines can be improved by the application of gravity.

Contrasting the curves of the Austinville zinc ore and the semi-anthracite coal, it can be seen that at very low gravities, the low ash coal separates from the high ash fraction. Higher gravities are required to separate the zinc ore. The explanation of this is that in the case of the Austinville ore, the average size is approximately 0.6 micron, while in the case of the coal, six microns is the average size requiring a higher gravity to separate the gangue from the zinc.

RECOMMENDATIONS

Due to the desire of the author to investigate a variety of ores and non-metallic minerals in connection with high-gravity flotation, he did not investigate any one particular type of material in detail. It was theorized that the method, to be successful, had to be applicable to a variety of minerals. For a better understanding of high-gravity flotation, it would be desirable in the future to concentrate on one particular type of ore by using various amounts of collector, various types of flotation reagents, various sizes, and fully delineate the limitations of high-gravity flotation. It may be that the method has application to flotation in the coarser size range than was investigated in this thesis and that high-gravity flotation may give substantial higher flotation rates in the coarser sizes. Even better results might be achieved by the use of a continuous cell without the repeated handling of the froth product which was necessary for the small scale laboratory manipulations. The manipulations may have caused some loss of values by small amounts of material sticking to the walls of the vessels. The field of study of high gravity in flotation has not been investigated in the past, to the author's knowledge, and would seem to offer excellent opportunities for a variety of investigations to ascertain the engineering feasability in actual flotation practice.

SUMMARY

From a study of the literature concerning the flotation of extreme fines, it appeared that there were various explanations for fines not floating: very little explanation was encountered of how to float these extreme fines. The author postulated that the flotation of fines could be improved by the application of gravity to the collection and separation of ores. Experiments were designed, and equipment was constructed for the experimental investigation of the use of high gravity utilizing centrifugal force in the collection and separation of minerals.

The results of the study of the flotation of extreme fines have indicated that centrifugal force, as applied to the collection of minerals, has a negative effect on their collection. The separation of minerals, after they have once been floated, is enhanced by the application of centrifugal force to the flotation froth.

Successful tests to illustrate the beneficial effects of centrifugal force on the separation of minerals have been done on a synthetic copper ore, a zinc ore, and a semi-anthracite coal. Tests were also conducted on an iron-bearing material and another set of tests were conducted on a zinc ore from the Great Gossan lead deposit which indicated no beneficial results from the application of high gravity in the form of centrifugal force. These results were explained as due to poor choice of reagents or incorrect conditioning.

APPENDIX

Methods of Analyses

Zinc:

1. Weigh 1.000 grams of dried sample.
2. Transfer to 250-milliliter beaker.
3. Add 15 milliliters of concentrated HNO_3.
4. Heat until brown fumes are almost gone.
5. Add one gram of $KClO_3$; reagent grade.
6. Evaporate to dryness.
7. Cool.
8. Add 10 - 15 grams of reagent grade NH_4CL.
9. Add 20 milliliters of strong NH_4OH.
10. Dilute to 75 milliliters with distilled water.
11. Add 20 milliliters of $Br_2(H_2O)$, bromine water.
12. Boil ten minutes.
13. Allow precipitate to settle.
14. Filter through rapid filter paper into 400-milliliter beaker.
15. Wash with hot water.
16. Make filtrate just acid with HCL: use methyl orange.
17. Add 5 - 10 milliliters of HCL in excess.
18. Dilute to 200 milliliters.
19. Add test lead.
20. Boil until all copper is precipitated.
21. Remove from heat.
22. Add 50 milliliters of H_2O.

23. Add 5 milliliters of strong HCL.
24. Titrate with $K_4Fe(CN)_6$ using Uranyl nitrate as an outside indicator.

<u>Copper</u>:

1. Use 1.000 grams of dried sample with material low in copper or 0.1000 grams with concentrate.
2. Add 5 milliliters of strong HCL.
3. Add 7 milliliters of strong HNO_3.
4. Heat to boiling.
5. When action has ceased, add 10 milliliters of 1:1 H_2SO_4 and water.
6. Evaporate until sulfuric acid fumes strongly.
7. Cool.
8. Add 30 milliliters of H_2O.
9. Heat until all soluable salts are in solution.
10. Add 5 milliliters of bromine water.
11. Boil for ten minutes or until bromine is gone.
12. Cool.
13. Add strong NH_4OH until $Fe(OH)_2$ persists.
14. Add 2 - 3 milliliters of 80 per cent acetic acid.
15. Cool to between 60 and 70 degrees Fahrenheit.
16. Add 2 grams of NH_4F.
17. Stir until dissolved.
18. Add 5 milliliters of fresh starch.
19. Titrate with thio sulfate.

Iron:

1. Weigh 1.0000 grams of iron-bearing material into a 250-milliliter beaker.
2. Add 25 milliliters of strong HCL.
3. Boil for one hour with watch glass cover or until residue is iron free.
4. Filter.
5. Heat filtrate to 196 degrees Fahrenheit.
6. Add $SnCL_2$ solution until the filtrate is colorless; then add two drops more.
7. Cool to room temperature.
8. Add saturated $HgCL_2$.
9. Titrate with potassium ferricyanide using barium diphenylamine sulfonate as an internal indicator.

Coal:

These were analysed according to A. S. T. M. (American Society for Testing Materials) specifications, D-271 for ash content.

Sample Calculations

Sedimentation Analyses.

Stokes' Law calculation:

$$V_t = \frac{2 \cdot g\, a^2 \quad (d_1 - d_2)}{9 \quad \eta}$$

V_t = Terminal velocity in centemeters per second

g = Acceleration due to gravity in centemeters per second squared

a = Radius of the particles in centemeters

d_1 = Specific gravity of mineral

d_2 = Specific gravity of fluid

η = Poises

The depth of the suspension is divided by the settling velocity to obtain the sampling time.

Each time the suspension is sampled, the new depth is measured and a new time is calculated.

The initial suspension density is used as a divisor for each sample taken to obtain a percentage which is the amount of material smaller than the calculated size.

$G_c$ravity is given by the formula below and is the relative gravities applied to the system.(33)

$$G_c\text{ravity} = \frac{V_t^{\ 2}}{g\, r}$$

V_t = Tangential velocity

g = Acceleration due to gravity

r = Distance from center of rotation

100 revolutions per minute

r = 0.5 feet

g = 32.2 feet per second ^{2}

D = 2 r

$$V_t = \frac{\pi D \times 100}{60} = \frac{3\cdot 14\cdot 1\cdot 10}{6} = \frac{5.23 \text{ feet}}{\text{second}}$$

$$V_t^{\,2} = 27.4$$

$$G_c\text{ravity} = \frac{27.4}{32.2 \times 0.5} = 1.694$$

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ABSTRACT

The flotation of very fine ores is desirable because of the occurrence of some ores that require fine grinding to achieve liberation. The literature has been concerned with electrical and chemical effects at the interface of finely ground minerals. The writer proposed to investigate the effect of increasing gravity by the use of centrifugal force on the collection and separation of finely ground ores.

The collection of minerals appeared to be hindered by increased gravity, but the separation was enhanced by increased gravity in the tests where conditions were conducive to froth flotation.