RI 8681

Bureau of Mines Report of Investigations/1982

Beneficiation of a Phosphate Ore Produced by Borehole Mining

By B. E. Davis, T. O. Llewellyn,
and G. V. Sullivan

UNITED STATES DEPARTMENT OF THE INTERIOR

Report of Investigations 8681

Beneficiation of a Phosphate Ore Produced by Borehole Mining

By B. E. Davis, T. O. Llewellyn,
and G. V. Sullivan

UNITED STATES DEPARTMENT OF THE INTERIOR
James G. Watt, Secretary

BUREAU OF MINES
Robert C. Horton, Director

This publication has been cataloged as follows:

Davis, B. E. (Broderick E.)

Beneficiation of a phosphate ore produced by borehole mining.

(Report of investigations ; 8681)

Includes bibliographical references.

Supt. of Docs. no.: I 28.23:8681.

1. Flotation. 2. Phosphates. 3. Phosphate mines and mining. I. Llewellyn, T. O. II. Sullivan, G. V. III. Title. IV. Series: Report of investigations (United States. Bureau of Mines) ; 8681.

TN23.U43 [TN523] 622s [622'.364] 82-600096 AACR2

CONTENTS

TABLES

BENEFICIATION OF A PHOSPHATE ORE PRODUCED BY BOREHOLE MINING

By B. E. Davis,[1] T. O. Llewellyn,[2] and G. V. Sullivan[3]

ABSTRACT

As part of its program to increase the domestic availability of critical minerals, the Bureau of Mines conducted characterization and beneficiation studies on a St. Johns County, Fla., phosphate ore that was mined by borehole mining. Results revealed that the ore from this test mine was mostly sand-size carbonate-fluorapatite and quartz and contained 24.6 percent P_2O_5. Concentration by various flotation techniques produced phosphate concentrates that ranged, in percent, from 30.0 to 31.4 P_2O_5, 0.73 to 0.80 MgO, and 2.8 to 7.2 acid insoluble. Recovery of the phosphate from the ore ranged from 89.4 to 96.3 percent.

[1]Minerals engineer.
[2]Supervisory (group) metallurgist.
[3]Supervisory (research) metallurgist.
The authors are with the Tuscaloosa Research Center, Bureau of Mines, Tuscaloosa, Ala.

INTRODUCTION

Phosphate is an essential fertilizer material, and an adequate supply is critical. As world population increases, so will the demand for food and, therefore, for fertilizer produced from phosphate. The demand for phosphate rock in the United States is predicted to increase by 47 percent by the year 2000.[4]

Presently, 80 percent of domestic phosphate production is from the central and north Florida area.[5] This phosphate is mined by stripping up to 50 feet of overburden, slurrying the phosphate matrix, and pumping it via pipeline to the concentration plant. At the concentrating plant a pebble product (plus 14-mesh) is produced by sizing. The remaining material is deslimed at 150 mesh, leaving minus 14- plus 150-mesh flotation feed. A phosphate concentrate is produced by flotation of this feed. The flotation scheme involves a fatty acid-fuel oil rougher float of the phosphate, a deoiling step, and amine flotation of the remaining quartz mineral. Generally, the feed is 10 to 15 percent P_2O_5 and is upgraded to 29 to 32 percent P_2O_5, with about 80 percent recovery of the phosphate.[6]

The Bone Valley deposits in central Florida are reported to contain enough phosphate to meet demands for the next two decades.[7] As this rich deposit is depleted, other sources must be evaluated. One such source is the phosphate contained in the Hawthorn Formation along part of the Florida east coast, referred to as the "East Coast" deposit. This deposit is described as being much deeper and containing little or no pebble compared to the Bone Valley deposit.[8] Because of the depth of the East Coast deposit, the present phosphate mining techniques are not feasible.

In pursuit of its goal of increasing the domestic availability of mineral resources and reserves, the Bureau of Mines recently awarded a contract to Flow Technology Co., Phoenix, Ariz., to evaluate borehole mining on one of these deep phosphate deposits. The deposit tested is located in St. Johns County, Fla.

In borehole mining, minerals are extracted through a borehole drilled from the surface through the mineral deposit, eliminating open pit and conventional underground mines. A high-pressure water jet, placed at the bottom of the borehole, is used to fragment and slurry the mineral-bearing ore. The slurry is pumped to the surface, the mineral recovered, and the water recycled into the mining process.

The Bureau of Mines conducted characterization and beneficiation studies of a grab sample of the St. Johns County ore mined by the borehole technique. The sample had been deslimed at the mining site and generally represented a flotation feed product. The studies included chemical, size, and petrographic analyses; heavy-liquid separation; and flotation. This report summarizes the results of the studies.

---

[4]Stowasser, W. F. Phosphate Rock. Ch. in Mineral Facts and Problems. BuMines Bull. 671, 1980, p. 681.

[5]Work cited in footnote 4.

[6]Zellars, M. E., and J. M. Williams. Evaluation of the Phosphate Deposits of Florida Using the Minerals Availability System. Final Report. (Contract J0377000). BuMines Open File Rept. 112-78, 1978, p. 57; available for reference at the Office of the Director, Division of Minerals Availability, Bureau of Mines, Washington, D. C.; Bureau of Mines facilities in Denver, Colo., and Pittsburgh, Pa.; and the National Library of Natural Resources, U.S. Department of the Interior, Washington, D.C.; and from National Technical Information Service, Springfield, Va., PB 286 648/AS.

[7]U.S. Comptroller General (General Accounting Office). Report to the Congress. Phosphates: A Case Study of a Valuable, Depleting Mineral in America. GAO Rept. EMD-80-21, Nov. 30, 1979, p. 1.

[8]Page 8 of work cited in footnote 6.

CHARACTERIZATION STUDIES

Chemical Analysis

Approximately 100 pounds of the ore were obtained. The ore contained about 20 percent moisture and was air dried to facilitate thorough mixing. A sample was taken using the coning and quartering technique. The head sample analyzed, in percent, 24.6 P_2O_5, 39.3 CaO, 0.71 MgO, 7.6 CO_2, 21.6 insolubles, and 0.46 Al_2O_3.

Sieve Analysis

Phosphate presently strip mined in central Florida contains a pebble-size product (plus 14-mesh), which does not require concentration. The ore produced by the borehole mining test did not contain any plus 14-mesh material. Most of the ore (85.8 percent) was in the minus 35- plus 150-mesh size range and contained only a minimal amount of clay, because the ore was dewatered by decantation prior to shipment. Table 1 presents a summary of the results of a sieve analysis.

TABLE 1. - Sieve analysis

Screen size fraction, mesh	Wt-pct	Analysis, pct			Distribution, pct		
		P_2O_5	MgO	Insolubles	P_2O_5	MgO	Insolubles
Plus 20....................	1.0	9.5	0.70	66.2	0.4	1.0	3.0
Minus 20 plus 28...........	3.7	9.1	.33	70.2	1.3	1.8	11.9
Minus 28 plus 35...........	5.3	19.2	.64	35.4	4.1	4.9	8.6
Minus 35 plus 48...........	13.6	25.6	.68	17.1	14.0	13.2	10.7
Minus 48 plus 65...........	31.6	26.9	.66	16.8	34.2	30.2	24.4
Minus 65 plus 100..........	26.4	27.9	.67	13.3	29.6	25.6	16.2
Minus 100 plus 150.........	14.2	25.6	.58	20.4	14.7	12.0	13.4
Minus 150 plus 200.........	2.8	12.2	.30	57.6	1.4	1.2	7.4
Minus 200 plus 400.........	.6	6.4	.19	79.6	.2	.2	2.3
Minus 400..................	.8	3.7	8.08	53.0	.1	9.9	2.1
Composite..............	100.0	24.8	.69	21.7	100.0	100.0	100.0

Petrographic Analysis

Petrographic studies of the ore revealed that the plus 400-mesh material was composed of carbonate-fluorapatite and quartz. The carbonate-fluorapatite was described as brown rounded pellets. The quartz varied in shape from rounded to subangular to angular. The minus 400-mesh material contained the MgO-bearing minerals sepiolite and dolomite, plus clay-dolomite aggregates along with some carbonate-fluorapatite and quartz. The phosphate mineral was liberated in all size fractions. Table 2 presents the petrographic data of the ore.

TABLE 2. - Petrographic grain count--percent of grains at each size fraction

Screen size fraction, mesh	Carbonate-fluorapatite	Quartz	Clay-dolomite aggregates	Clay
Plus 28.........................	30	60	0	10
Minus 28 plus 35................	67	30	0	3
Minus 35 plus 48................	85	15	0	0
Minus 48 plus 65................	88	12	0	0
Minus 65 plus 100...............	90	10	0	0
Minus 100 plus 150..............	85	15	Trace	0
Minus 150 plus 200..............	43	55	1	1
Minus 200 plus 400..............	19	55	1	25
Minus 400[1]......................	Moderate	Minor	Moderate	Major

[1]The grains in this fraction could not be accurately counted because of the clay content.

Heavy-Liquid Separation

To obtain an indication of results that could be expected from physical concentration, heavy-liquid separation studies were performed on the sample. The ore was ground to pass 35 mesh and sized at minus 35 plus 150 mesh and minus 150 plus 400 mesh, then separated at specific gravities of 2.68 and 2.85. These separations resulted in three products--float 2.68, sink 2.68-float 2.85, and sink 2.85--on each of the two size fractions. As expected, the float 2.68 products were composed mostly of quartz. In the minus 35- plus 150-mesh fraction, the sink 2.68-float 2.85 contained 29.6 percent P_2O_5 and accounted for 42.1 percent of the P_2O_5 in this size fraction. The sink 2.85 contained 30.7 percent P_2O_5 and accounted for 49.3 percent of the P_2O_5 in the minus 35- plus 150-mesh fraction. By combining these two products, a concentrate containing 30.2 percent P_2O_5 with a 91.4 percent recovery could be produced by physical means. Table 3 presents complete heavy-liquid separation data.

TABLE 3. - Heavy-liquid separation data

Product	Wt-pct	P_2O_5	CaO	MgO	CO_2	Insolubles	Al_2O_3
ANALYSIS, PCT							
Minus 35 plus 150 mesh:							
Float 2.68	24.4	8.80	15.10	0.19	3.60	71.60	0.12
Sink 2.68-float 2.85	35.5	29.60	46.90	.81	11.00	3.60	.45
Sink 2.85	40.1	30.70	48.50	.68	10.60	2.70	.41
Composite	100.0	24.98	39.80	.61	9.04	19.79	.33
Minus 150 plus 400 mesh:							
Float 2.68	53.0	2.10	4.30	0.09	1.20	91.60	0.01
Sink 2.68-float 2.85	34.8	36.40	47.20	.74	10.80	4.10	.41
Sink 2.85	12.2	14.20	22.70	.27	4.70	53.40	.17
Composite	100.0	15.50	21.46	.34	4.96	56.52	.17
Composite:							
Minus 35 plus 150 mesh	89.7	24.50	39.80	0.61	9.00	19.80	0.33
Minus 150 plus 400 mesh	5.9	15.50	21.50	.34	5.00	56.50	.17
Minus 400 mesh	4.4	18.60	32.50	2.76	1.80	28.60	1.80
Composite	100.0	23.71	38.39	.69	8.44	22.37	.39
DISTRIBUTION, PCT							
Minus 35 plus 150 mesh:							
Float 2.68	24.4	8.6	9.2	7.6	9.7	88.1	13.3
Sink 2.68-float 2.85	35.5	42.1	41.9	47.4	43.2	6.4	73.0
Sink 2.85	40.1	49.3	48.9	45.0	47.1	5.5	13.7
Composite	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Minus 150 plus 400 mesh:							
Float 2.68	53.0	7.2	10.6	14.1	12.8	86.0	3.1
Sink 2.68-float 2.85	34.8	81.6	76.5	76.1	75.7	2.5	84.6
Sink 2.85	12.2	11.2	12.9	9.8	11.5	11.5	12.3
Composite	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Composite:							
Minus 35 plus 150 mesh	89.7	92.6	93.0	79.4	95.6	79.4	76.8
Minus 150 plus 400 mesh	5.9	3.9	3.3	2.9	3.5	15.0	2.6
Minus 400 mesh	4.4	3.5	3.7	17.8	.9	5.6	20.6
Composite	100.0	100.0	100.0	100.0	100.0	100.0	100.0

FLOTATION

Flotation studies were undertaken to determine if an acceptable phosphate concentrate could be produced from the St. Johns County ore. Three flotation techniques were employed to concentrate the phosphate. One was the standard double float method currently used by the phosphate industry. The other two techniques were developed by the Bureau of Mines.

Batch flotation tests were conducted with the ore both unground and ground to pass 35 mesh. Grinding was performed at 70 percent solids in a pebble mill. Before flotation, the pulp was scrubbed in an attrition scrubber to break up any remaining clay aggregates and deslimed at 150 mesh.

Standard Double Float Method

The standard double float method employs an anionic flotation step of the phosphate, followed by a cationic flotation step of the remaining quartz. In this method, NaOH was added to the scrubbing step for dispersion and pH control. After desliming, the ore was conditioned at 60 percent solids with 1.5 pounds of tall-oil fatty acid and fuel oil mixture per ton of ore.[9] A rougher phosphate concentrate was floated, leaving a rougher tailing consisting mostly of quartz. The rougher concentrate was deoiled with 5.0 pounds H_2SO_4 per ton. The deoiled pulp was thickened and conditioned with 0.25 pound amine and 0.5 pound kerosine per ton, and a quartz product was floated, with the underflow being the phosphate concentrate. For the unground ore, the concentrate analyzed, in percent, 31.4 P_2O_5, 47.8 CaO, 0.79 MgO, 10.0 CO_2, 3.2 insolubles, and 0.45 Al_2O_3. The attendant recovery of P_2O_5 from the ore was 89.4 percent. For the ore ground to pass 35 mesh, the concentrate contained, in percent, 31.2 P_2O_5, 48.6 CaO, 0.75 MgO, 9.9 CO_2, 2.8 insolubles, and 0.42 Al_2O_3. The recovery of P_2O_5 from the ore was 91.0 percent. Detailed results are presented in tables 4 and 5. The reagent scheme is given in table 6.

TABLE 4. - Standard double float method--ore not ground

Product	Wt-pct	P_2O_5	CaO	MgO	CO_2	Insolubles	Al_2O_3
ANALYSIS, PCT							
Phosphate concentrate........	71.6	31.4	47.8	0.79	10.0	3.2	0.45
Silica tailings..............	6.9	5.9	8.4	.15	2.8	81.9	.12
Rougher tailings.............	16.4	9.5	11.0	.31	3.6	68.8	.21
Slimes.......................	5.1	14.0	21.9	1.69	14.2	47.0	1.08
Composite..............	100.0	25.2	37.7	.71	8.7	21.6	.42
DISTRIBUTION, PCT							
Phosphate concentrate........	71.6	89.4	90.7	79.3	82.6	10.6	76.7
Silica tailings..............	6.9	1.6	1.5	1.5	2.2	26.1	2.0
Rougher tailings.............	16.4	6.2	4.8	7.1	6.8	52.2	8.2
Slimes.......................	5.1	2.8	3.0	12.1	8.4	11.1	13.1
Composite..............	100.0	100.0	100.0	100.0	100.0	100.0	100.0

[9] All reagent usage is expressed in pounds of reagent per ton of ore.

TABLE 5. - Standard double float method--ore ground to minus 35 mesh

Product	Wt-pct	P_2O_5	CaO	MgO	CO_2	Insolubles	Al_2O_3
ANALYSIS, PCT							
Phosphate concentrate	73.0	31.2	48.6	0.75	9.9	2.8	0.42
Silica tailings	8.3	10.4	16.5	.23	4.0	68.5	.14
Rougher tailings	11.8	2.3	3.6	.08	1.5	86.1	.13
Slimes	6.9	16.1	26.9	1.43	15.2	43.3	.77
Composite	100.0	25.0	39.1	.67	8.8	20.9	.39
DISTRIBUTION, PCT							
Phosphate concentrate	73.0	91.0	90.7	81.2	82.3	9.8	79.3
Silica tailings	8.3	3.5	3.5	2.8	3.8	27.2	3.0
Rougher tailings	11.8	1.1	1.1	1.4	2.0	48.7	4.0
Slimes	6.9	4.4	4.7	14.6	11.9	14.3	13.7
Composite	100.0	100.0	100.0	100.0	100.0	100.0	100.0

TABLE 6. - Reagent scheme for standard double float method

Operation	Reagents		Time, min	pH
	Type	Amount, lb/ton		
Scrub	NaOH	0.5	15	[1]8.0
Condition	FA-FO[2]	1.5	2	9.4
Rougher flotation	None	NAp	2	9.4
Deoil	H_2SO_4	5.0	15	6.0
Condition	Armac T[3]	.125	.25	5.8
	Armac C	.125		
	Kerosine	.5		
Sand float	None	NAp	2	5.8

NAp Not applicable.
[1]At end of scrubbing.
[2]Fatty acid-fuel oil mixed by Florida phosphate producer.
[3]Reference to specific trade names does not imply endorsement by the Bureau of Mines.

Anionic Flotation Method

The second method used was a technique developed by the Bureau of Mines for flotation of phosphate from high-MgO ores using only an anionic reagent.[10] The ore was scrubbed in the presence of NaOH for pH control and dispersion. After desliming, the pulp was conditioned at 60 percent solids with 0.64 pound per ton tall-oil fatty acid and 0.96 pound fuel oil per ton. A rougher phosphate concentrate was floated, leaving a quartz tailing as the underflow. The rougher concentrate was cleaned three times in the presence of sodium silicate in the amount of 1.0 pound per ton for gangue depression. For the unground ore, the concentrate contained, in percent, 30.4 P_2O_5, 46.2 CaO, 0.75 MgO, 10.2 CO_2, 5.5 insolubles, and 0.42 Al_2O_3. The P_2O_5 recovery from the ore was 94.9 percent. For the ore ground to pass 35 mesh, the concentrate contained, in percent, 30.2 P_2O_5, 47.4 CaO, 0.73 MgO, 10.2 CO_2, 5.5 insolubles, and 0.38 Al_2O_3. The attendant P_2O_5 recovery was 92.2 percent. Results of this method are featured in tables 7 and 8. The reagent scheme is shown in table 9.

[10]Llewellyn, T. O., B. E. Davis, G. V. Sullivan, and J. P. Hansen. Beneficiation of High-Magnesium Phosphate From Southern Florida. BuMines RI 8609, 1982, 16 pp.

TABLE 7. - Anionic flotation method--ore not ground

Product	Wt-pct	P_2O_5	CaO	MgO	CO_2	Insolubles	Al_2O_3
ANALYSIS, PCT							
Phosphate concentrate........	77.2	30.4	46.2	0.75	10.2	5.5	0.42
Cleaner tailings.............	4.9	10.0	15.8	.31	3.7	68.2	.20
Rougher tailings.............	13.4	1.2	1.8	.05	.8	95.9	.07
Slimes.......................	4.5	13.4	22.5	2.47	13.4	46.6	1.31
Composite...............	100.0	24.7	37.7	.71	8.8	22.5	.40
DISTRIBUTION, PCT							
Phosphate concentrate........	77.2	94.9	94.6	81.4	89.8	18.8	80.6
Cleaner tailings.............	4.9	2.0	2.1	2.1	2.1	14.8	2.4
Rougher tailings.............	13.4	.7	.6	.9	1.2	57.1	2.3
Slimes.......................	4.5	2.4	2.7	15.6	6.9	9.3	14.7
Composite...............	100.0	100.0	100.0	100.0	100.0	100.0	100.0

TABLE 8. - Anionic flotation method--ore ground to minus 35 mesh

Product	Wt-pct	P_2O_5	CaO	MgO	CO_2	Insolubles	Al_2O_3
ANALYSIS, PCT							
Phosphate concentrate........	75.0	30.2	47.4	0.73	10.2	5.5	0.38
Cleaner tailings.............	5.9	7.4	12.2	.22	3.0	76.4	.12
Rougher tailings.............	11.5	3.4	5.3	.09	1.7	89.3	.09
Slimes.......................	7.6	14.3	24.2	1.40	10.1	49.0	.80
Composite...............	100.0	24.6	38.7	.70	8.8	22.6	.40
DISTRIBUTION, PCT							
Phosphate concentrate........	75.9	92.2	91.7	80.9	87.1	18.2	78.4
Cleaner tailings.............	5.9	1.8	1.9	1.9	2.0	19.9	2.0
Rougher tailings.............	11.5	1.6	1.6	1.5	2.2	45.4	2.9
Slimes.......................	7.6	4.4	4.8	15.7	8.7	16.5	16.7
Composite...............	100.0	100.0	100.0	100.0	100.0	100.0	100.0

TABLE 9. - Reagent scheme for anionic flotation method

Operation	Reagents		Time, min	pH
	Type	Amount, lb/ton		
Scrub......................	NaOH..............	0.5	15	[1]7.8
Condition..................	Acintol FA1[2]......	.64	2	9.0
	Fuel oil..........	.96		
Rougher flotation..........	None...............	NAp	2	9.0
Cleaner flotation 1.......	Sodium silicate...	1.0	1	9.0
Cleaner flotation 2.......	.....do...........	1.0	1	9.0
Cleaner flotation 3.......	.....do...........	1.0	1	9.0

NAp Not applicable.

[1]After scrubbing.

[2]Reference to specific trade names does not imply endorsement by the Bureau of Mines.

Cationic Flotation Method

The third method used was another developed by the Bureau of Mines. It involved flotation of only the quartz mineral with a cationic reagent. The ore was scrubbed in the presence of H_2SO_4 for pH control. The pulp was conditioned at 60 percent solids with 0.25 pound amine per ton and 0.5 pound kerosine per ton and a quartz product floated. The underflow was conditioned a second time with the same reagent scheme and a scavenger float performed, leaving the phosphate concentrate as the underflow. For the unground ore, the concentrate contained, in percent, 30.0 P_2O_5, 46.4 CaO, 0.80 MgO, 9.9 CO_2, 7.2 insolubles, and 0.43 Al_2O_3. The P_2O_5 recovery was 96.3 percent. For the ore ground to pass 35 mesh, the concentrate contained, in percent, 30.8 P_2O_5, 48.3 CaO, 0.75 MgO, 10.3 CO_2, 3.9 insolubles, and 0.45 Al_2O_3. The recovery was 93.7 percent of the P_2O_5 from the ore. Results of the cationic flotation method are presented in tables 10 and 11. Table 12 gives the reagent scheme for this method. Table 13 is a summary of the flotation results for the three methods.

TABLE 10. - Cationic flotation method--ore not ground

Product	Wt-pct	P_2O_5	CaO	MgO	CO_2	Insolubles	Al_2O_3
ANALYSIS, PCT							
Phosphate concentrate........	80.3	30.0	46.4	0.80	9.9	7.2	0.43
Tailings[1]....................	14.8	2.0	3.8	.07	1.4	92.6	.08
Slimes.......................	4.9	12.7	21.8	1.95	1.3	53.5	.99
Composite...............	100.0	25.0	38.9	.75	8.2	22.1	.41
DISTRIBUTION, PCT							
Phosphate concentrate........	80.3	96.3	95.7	85.8	96.7	26.2	85.1
Tailings[1]....................	14.8	1.2	1.5	1.4	2.5	61.9	2.9
Slimes.......................	4.9	2.5	2.8	12.8	.8	11.9	12.0
Composite...............	100.0	100.0	100.0	100.0	100.0	100.0	100.0

[1]Combined rougher and scavenger tailings.

TABLE 11. - Cationic flotation method--ore ground to minus 35 mesh

Product	Wt-pct	P_2O_5	CaO	MgO	CO_2	Insolubles	Al_2O_3
ANALYSIS, PCT							
Phosphate concentrate........	74.7	30.8	48.3	0.75	10.3	3.9	0.45
Tailings[1]....................	17.8	2.8	4.5	.10	1.0	91.5	.10
Slimes.......................	7.5	14.0	23.3	1.48	10.1	49.8	.93
Composite...............	100.0	24.6	38.6	.69	8.6	22.9	.42
DISTRIBUTION, PCT							
Phosphate concentrate........	74.7	93.7	93.4	81.3	89.1	12.7	79.3
Tailings[1]....................	17.8	2.0	2.1	2.6	2.1	71.0	4.2
Slimes.......................	7.5	4.3	4.5	16.1	8.8	16.3	16.5
Composite...............	100.0	100.0	100.0	100.0	100.0	100.0	100.0

[1]Combined rougher and scavenger tailings.

TABLE 12. - Reagent scheme for cationic flotation method

Operation	Reagents		Time, min	pH
	Type	Amount, lb/ton		
Scrub	H_2SO_4	0.5	15	7.0
Condition 1	Armac T[1] Armac C Kerosine	.125 .125 .5	.25	7.4
Float 1	None	NAp	2.0	7.4
Condition 2	Armac T Armac C Kerosine	.125 .125 .5	.25	7.4
Float 2	None	NAp	2.0	7.4

NAp Not applicable.
[1]Reference to specific trade names does not imply endorsement by the Bureau of Mines.

TABLE 13. - Summary of flotation results

Flotation method	Grind, mesh	Concentrate analysis, pct P_2O_5	P_2O_5 recovery, pct
Standard double float	None	31.4	89.4
Do	Minus 35	31.2	91.0
Anionic flotation method	None	30.4	94.9
Do	Minus 35	30.2	92.2
Cationic flotation method	None	30.0	96.3
Do	Minus 35	30.8	93.7

CONCLUSIONS

The Bureau of Mines conducted characterization studies of a phosphate ore mined by borehole mining, which showed that the ore contained mostly phosphate and quartz minerals that were mainly sand sized. Heavy-liquid separation tests showed that a phosphate concentrate could be produced by physical means.

Three flotation techniques were tested on the ore. The standard double float is a two-step process, each step being a single flotation stage. Using the standard double float, a 31.2 percent P_2O_5 concentrate with a recovery of 91 percent of the P_2O_5 was produced with the ore ground to minus 35 mesh, and a 31.4 percent P_2O_5 concentrate with a recovery of 89.4 percent with the unground ore. Two methods developed by the Bureau of Mines were employed to produce a phosphate concentrate. The anionic flotation method is a one-step process involving four stages, a rougher and three cleaner flotation stages. A concentrate containing 30.2 percent P_2O_5

with an attendant recovery of 92.2 percent was produced using the anionic flotation method on the ground ore, and a 30.4 percent P_2O_5 concentrate with a recovery of 94.9 percent using this method on the unground ore. The cationic flotation method is a one-step process using a rougher and scavenger stage. A concentrate containing 30.8 percent P_2O_5 with a recovery of 93.7 percent of the P_2O_5 was produced by the cationic flotation method on the ground ore, and a 30.0 percent P_2O_5 concentrate with an attendant recovery of 96.3 percent with the unground ore.

It is recommended that continuous studies be conducted to obtain engineering data for the three methods described in this report. From the data obtained, process evaluations and comparisons would be reported for the three methods.

☆U.S. GOVERNMENT PRINTING OFFICE: 1982 - 505 - 002/55 INT.-BU.OF MINES,PGH.,PA. 26177