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Column Flotation of Non-Slimed Jordanian Siliceous Phosphate

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Abstract

In phosphate flotation, it is a standard practice to remove (deslime) fine particles before conducting flotation. Since phosphate matrix is friable, agitating, scrubbing, or grinding causes a significant loss in valuable phosphate; this has an economic and ecological impact on phosphate industry. The aim of this study is to minimise the loss of phosphate slime by conducting column flotation without desliming. Fractional factorial experimental design was used to evaluate flotation performance of non-slimed siliceous phosphate in flotation column. The effect of gas flow rate, feed size (P_{80}), and sodium silicate dosage were statistically evaluated. The results showed that gas flow rate followed by feed size were the most significant parameters. The general trend in flotation results was poor flotation recovery and concentrate grade. However, concentrate grade and flotation recovery were slightly improved when fine feed was used which may be due to the increase in phosphate particles liberation.

Keywords: Column Flotation, Sedimentary Phosphate Flotation, Phosphate Slime, Jordan.

1. Introduction

Using column flotation in minerals industry has been increased in recent years. Conventional mechanical flotation cells have been replaced by column cells in several mines especially in cleaning circuits. The main advantages of column flotation can be summarised as follows (Yahyaei et al, 2008; Finch and Doby,1991;Tao et al, 2000; Ityokumbul,1992; Oliveira et al,2007; Hacifazlioglu ,and Sutcu,2007; Guimaraes and Peres,1999; Fortes et al,2007; Delvillar et al,1999; Abdel-Khalek et al, 2000).

- Cleaner concentrate which reduces cleaning and re-cleaning circuits.
- Reducing energy consumption because of less mechanical parts compared with mechanical flotation cells.
- Less maintenance costs due to less mechanical parts.
- High capacity to size-ratio compared to mechanical flotation cell, which may reduce capital costs.

Extensive research has been done in order to identify the important operating parameters that affect column flotation performance in order to maximise flotation recovery and concentrate grade. To mention some, Patile et al (1996) used factorial experimental design to study the effect of gas flow rate, wash water, and froth height on column flotation of Indian siliceous phosphate. Using Sodium Oleate as phosphate collector and sodium silicate as silica depressant, the authors obtained concentrate grade containing 31% P_2O_5 with more than 94 % recovery.

El- Shall et al (2003) studied the use of column flotation to upgrade Florida coarse phosphate (850-425 μm) feed. The authors used different types of frothers to investigate the effect of collector –frother interaction on flotation recovery and concentrate grade. They obtained a concentrate assaying 31% P_2O_5 with more than 96% recovery.

Ityokumbul et al (2003) evaluated the effect of amine dosage and airflow rate on phosphate rougher flotation in a pilot- plant column cell. The authors mentioned that collector dosage used in column flotation was less than that used in conventional mechanical cell. In addition, the authors claimed that flotation water didn't significantly affect flotation efficiency if its pH, and turbidity were kept under a certain level.

Recently, Fortes et al (2007) used bench- scale column flotation to separate siliceous gangue from Brazilian phosphate ore. Using alkali amine as silica collector and cornstarch as phosphate depressor, the authors obtained concentrate with less than 8% SiO_2 and more than 90 % P_2O_5 recovery.

In most of phosphate flotation reported in the literature, flotation feed is usually deslimed i.e. fine fraction, less than 100 μm , is removed. This is due to the deteriorating effect of slimes on flotation recovery and concentrate grade. Phosphate particles are friable; so agitating, scrubbing or grinding followed by desliming may cause an economic loss where considerable amount of phosphate is lost. This work is an attempt to evaluate the possibility of separating valuable phosphate from siliceous gangue by column flotation without desliming. Flotation performance was statistically evaluated by laboratory scale column flotation using different feed size (P_{80}), depressant dosage, and gas flow rate.

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2. Experimental set up

Phosphate feed used in this work was obtained from Eshydia mine in south of Jordan. The feed (-1mm) was dried in 70c oven for 24 hours then separated into two parts: feed 1 is (-1000+355 μm) while feed 2 is (-355 μm). No further desliming was conducted on both feeds. Chemical analysis of flotation feed, concentrate and tailing were conducted by Induced coupled Plasma-spectroscopy (ICP). Tables 1 and 2 show the chemical and physical analysis of these feeds.

Table 1 Chemical analysis of flotation feed.

Content	%wt	
	Feed1	Feed 2
P_2O_5	23.6	21.2
SiO_2	34.4	41.7
CaO	35.6	31.8
Fe_2O_3	0.66	0.49
Al_2O_3	0.65	0.37
MgO	0.2	0.14
LOI	3.56	2.94

Table 2 Size distribution of flotation feeds (feed 1 and 2).

Upper size (micron)	% Undersize	
	Feed 1	Feed 2
1000	100.00	*
710	82.61	*
500	50.35	*
355	*	100.00
255	*	57.37
180	*	31.40
125	*	16.22
90	*	5.71
63	*	2.73
45	*	1.05

Flotation tests were conducted by column flotation as shown in Figure 1. For each test, 3kg of the feed was conditioned with the required amount of collector in 45 L tank at 50 % solids for 10 minutes. The pulp was then diluted to 10 % solids and frother was added. Slurry was then pumped into the column until the pulp level reached the required height (froth height 1 m).

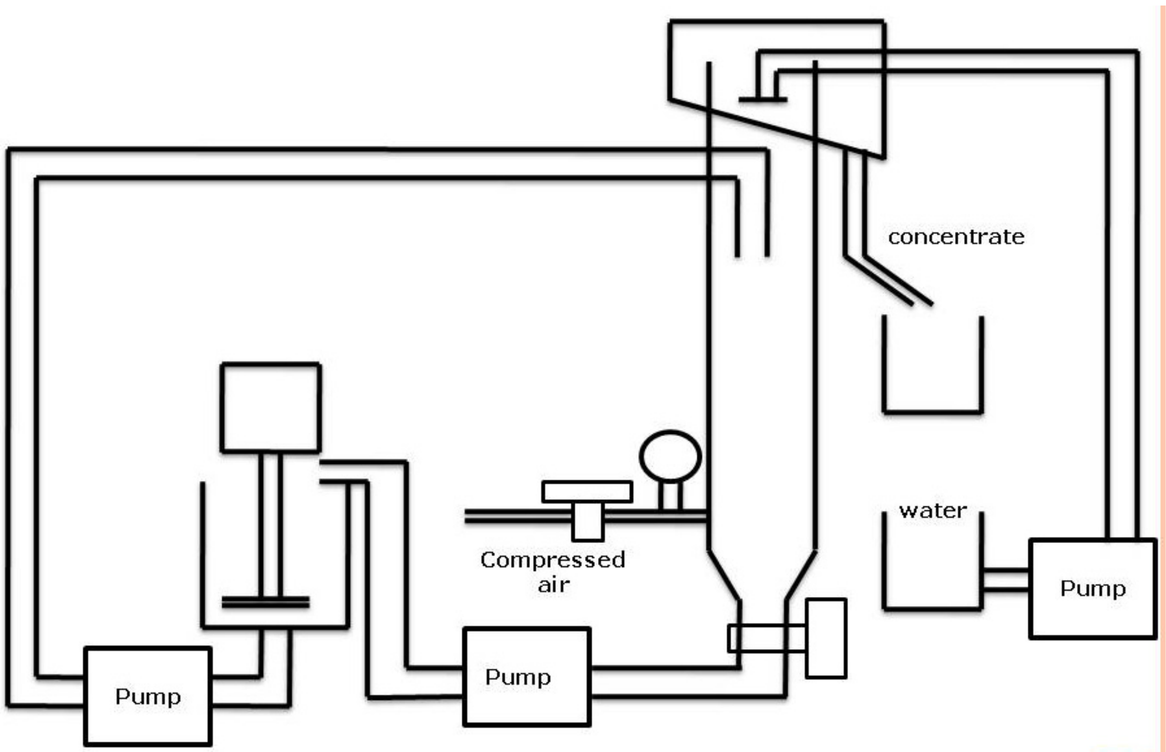

Figure 1. Schematic diagram of flotation column

Slurry pumping rates were calibrated to keep pulp level at 2m. Air was then introduced to the cell for 10 minutes. Concentrate was collected and the slurry left in the column and tanks were collected as flotation tailing. Flotation and conditioning parameters shown in Table 3 were held constant throughout the test unless otherwise is stated. Chemicals used in flotation tests were oleic acid as phosphate collector, sodium silicate as silica depressor, Dowfroth200 as frother, and sodium hydroxide as pH modifier. All these chemicals were used in their reagent grade.

Table 3 Constant parameters of conditioning and flotation

Conditioning		Flotation	
Parameter	value	Parameter	value
Solids %	50 %	Solids %	10
Time	10 minutes	Flotation time	10 minutes
pH	9.5	Wash water	0.2 L/min (0.019cm/s)
Impeller speed	750 RPM	Collector dosage/(g/t)	1200 (4.2×10^{-3} M)
Solids charge	3000 g	Froth height	1000 mm
Water type	Tap water	Frother dosage (MIBC)	80 PPM

Two level fractional factorial experimental design (2^3) with one replicate was used to evaluate flotation performance on different set of parameters. These parameters and their values are shown in Table 4. Flotation recovery, concentrate grade, and amount of water reported to concentrate were measured for each test.

Table 4 Variable conditioning and flotation parameters

Parameter	Values	
	Low	High
Feed size(FS) (P80,micron)	300	700
Sodium silicate(S.S) (g/t)	0	100
Air flow rate(J_g)(cm/s)	1	1.5

3. Results and discussion

3.1. The effect of flotation parameters on recovery and concentrate grade

The results of flotation experiments and their replicates are shown in Table 5.The table shows that fractional factorial design was used to evaluate of the studied flotation parameters and their interactions on flotation recovery and grade.

Statistical analysis (analysis of variance, ANOVA) was conducted on previous results in order to statistically evaluate the effect of these parameters as shown in Tables 6 and 7.

Table 5 Flotation experimental results

Exp No.	FS micron	S.S g/t	J_g cm/s	Replicate 1		Replicate 2	
				Recovery %	P_2O5 %	Recovery %	P_2O5 %
1	L	L	L	29.60	28.12	29.35	27.71
2	H	L	L	16.84	28.61	16.71	28.26
3	L	H	L	13.32	27.89	13.46	27.25
4	H	H	L	30.84	26.87	30.29	26.53
5	L	L	H	62.53	28.69	62.97	28.10
6	H	L	H	26.60	24.70	24.96	25.30
7	L	H	H	77.81	27.28	77.92	27.29
8	H	H	H	29.02	24.74	28.93	24.82

Table 6 ANOVA analysis of flotation recovery

Source	Sum Sq.	d.f	Mean Sq.	F	Prob>F
FS	1713.96	1	1713.96	2088.25	0
S.S	68.89	1	68.89	88.93	0
J_g	2754.68	1	2754.68	3356.24	0
FS*S.S	85.66	1	85.66	104.36	0
FS*J_g	2078.45	1	2078.45	2532.34	0
S.S*J_g	112.57	1	112.57	137.16	0
FS*S.S*J_g	409.05	1	409.05	498.38	
Error	6.57	8			
Total	7229.82	15			

Table 7 ANOVA analysis of concentrate grade

Source	Sum Sq.	d.f	Mean Sq.	F	Prob>F
FS	7.3712	1	7.3712	19.11	0.0024
S.S	3.0102	1	3.0102	7.81	0.0234
J_g	4.4944	1	4.4944	11.65	0.0092
FS*S.S	0.2162	1	0.2162	0.56	0.4755
FS*J_g	6.5025	1	6.5025	16.86	0.0034
S.S*J_g	0.2025	1	0.2025	0.53	0.4894
FS*S.S*J_g	0.6241	1	0.6241	1.62	0.2391
Error	3.0858	8	0.385725		
Total	25.507	15			

Table 6 showed that all the flotation parameters (feed size, sodium silicate dosage, and gas flow rate) and their interactions were significant for flotation recovery. However, a comparison between F- values showed that gas flow rate (Jg) has the most significant effect followed by the interaction between feed size and gas flow rate. On the other hand, sodium silicate has the least significant effect; even less than the third interaction between feed size, gas flow rate and sodium silicate. Feed size (P_{80}) was the third significant parameter but its interaction with gas flow was more significant. This emphasizes the role of interactions between parameters in flotation.

Increasing feed size with the same amount of collector reduced the amount of particles reported to concentrate by true flotation since coarser particles require larger area coverage by the collector to be attached strongly enough to the rising air bubbles. On the other hand, fine feed increased the recovery of both gangue and valuable particles due to either entrapment in froth or by entrainment with the rising swarm of bubbles. Increasing gas flow rate encouraged such entrainment but has more effect on flotation of fine particles .These results are in agreement with those obtained by Patil et al (1996) who stated that bubble- particle collection rate is linearly proportional to gas flow rate. On the other hand, the addition of sodium silicate improved the selectivity of the collector by depressing silica and stabilizing the froth. Further increase of sodium silicate increased the bulk precipitation of sodium silicate on both of apatite and silicate surface, which affect the selectivity and apatite recovery (Dho and Iwasaki, 1990).

Table 7 shows that concentrate grade was less sensitive to the change on flotation parameters. Feed size followed by the interaction between gas flow rate and feed size were the most significant parameters. On the other hand, feed size and sodium silicate interactions, sodium silicate and gas flow rate interactions, as well as the third interaction between feed size, sodium silicate and gas flow rate were not significant. This may be due to the non-selectivity in flotation of this type of non-slimed feed where phosphate particles may report to concentrate by other means than true flotation (bubble–particle attachment) such as entrainment. The improvement of concentrate grade when using finer feed and higher gas flow rate supports this claim.

3.2. The relation between water recovery and flotation performance

Table 8 shows the results of water flow rate (J_w) reported to concentrate normalised by the cell cross section while Table 9 and 10 show ANOVA analysis of water recovery. Separation efficiency (SE) was calculated by equation 1.

$$SE = (P_2O_5 \text{ recovery} \times silica\ \ removal\ )*100\% \quad (1)$$

Table 8 Flotation experimental results

Exp No.	FS micron	S.S g/t	J_g cm/s	Replicate 1		Replicate 2	
				J_w (cm/s)	SE %	J_w (cm/s)	SE %
1	L	L	L	0.0399	26.48	0.0418	23.27
2	H	L	L	0.0397	15.58	0.0346	16.10
3	L	H	L	0.0626	13.04	0.0625	10.92
4	H	H	L	0.0644	24.95	0.0641	27.78
5	L	L	H	0.1530	48.81	0.1530	47.20
6	H	L	H	0.0762	19.97	0.0746	21.03
7	L	H	H	0.1355	54.78	0.1409	55.22
8	H	H	H	0.1461	22.89	0.1367	21.04

Table 9 ANOVA analysis of separation efficiency (SE)

Source	Sum Sq.	d.f	Mean Sq.	F	Prob>F
FS	761.48	1	761.48	400.78	0
S.S	9.27	1	9.27	4.90	0.0582
J_g	1102.57	1	1102.57	580.3	0
FS*S.S	80.01	1	80.01	42.11	0.0002
FS*J_g	1085.37	1	1085.37	571.25	0
S.S*J_g	29.32	1	29.32	15.43	0.0044
FS*S.S*J_g	209.53	1	209.53	110.28	0
Error	15.2	8	1.9	1	
Total	3292.77	15			

Table 10 ANOVA analysis of water recovery

Source	Sum Sq.	d.f	Mean Sq.	F	Prob>F
FS	0.00146	1	0.0015	155.86	0
S.S	0.0025	1	0.0025	267.02	0
J_g	0.02298	1	0.0230	2454.75	0
FS*S.S	0.00186	1	0.0017	198.41	0
FS*J_g	0.00131	1	0.0013	139.97	0
S.S*J_g	0	1	0	0.15	0.7052
FS*S.S*J_g	0.00142	1	0.0014	151.81	0
Error	0.00007	8	0.00001		
Total	0.03161	15			

As shown in previous tables, gas flow rate, feed size and their interactions were the most significant parameters on separation efficiency and water recovery. Gas flow rate effect was very significant in water recovery, even higher than the effect of feed size. This shows that gas flow rate increases particle entrainment by increasing water recovery. This was more apparent in flotation of the finer feed.

In all previous results, the general trend was low recovery and concentrate grade, which may be due to poor liberation of phosphate particles in this non-slimed feed. So to investigate the effect of feed liberation on flotation performance, the previous +1 mm run of mine feed were wet grinded by rod mill for 10 minutes to (P_{80}=280 μm) (feed 3). Flotation results for this non-slimed feed are given in the following section.

3.3. The effect of feed size on flotation performance

Size distribution and chemical analysis of feed 3 are given in Tables 11 and 12 while flotation parameters and results are given in Table 13.

Table 11 Size distribution of flotation feed (feed 3).

Upper size (micron)	% Undersize
500	100.00
355	90.61
255	75.80
180	48.02
125	29.54
90	12.42
63	6.36
45	3.12

Table 12 Chemical analysis of flotation feed (feed 3)

Content	Feed 3(%)
P_2O_5	27.6
SiO_2	23.6
CaO	41.8
Fe_2O_3	0.6
Al_2O_3	0.65
MgO	0.25
LOI	3.56

Table 13 Flotation experimental results (feed3)

Collector dosage :1200 g/t ,wash water :200 ml/min, gas flow rate :1.5 cm/s						
Exp No.	Frother (ppm)	S.S g/t	P_2O_5 %	P_2O_5 Recovery %	Silica removal %	Separation Efficiency %
1	40	160	29.8	67.4	76.1	51.21
2	80	160	30.4	56.3	83.0	46.71
3	40	320	30.2	53.7	83.6	44.90
4	80	320	30.4	55.9	83.6	46.71

A comparison between the results shown in Table 13 with those in Table 5 showed an improvement in both of concentrate grade and recovery. This indicates an increase in the percentage of particles reported to concentrate by true flotation other than non-selective entrainment. According to Peng and Gu (2005), about 85 % of Florida phosphate particles are liberated at (P_{100}=300 μm). By interpolation of Peng and Gu data, phosphate feed need to be ground to at least (P_{100}=150 μm) to obtain 95% liberation.

4. Conclusions

Based on previous results, the following conclusions can be drawn:

- Factorial experimental design used in this work showed that the interaction between flotation parameters was significant on flotation recovery and concentrate grade. The order of significance was gas flow rate, feed size, and sodium silicate dosage.
- Agitating and scrubbing of flotation feed was very significant on flotation recovery and concentrate grade especially with no grinding because of low phosphate particles liberation. The effect of slimes (fines) generated by such process on flotation performance can be reduced by using column flotation.

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