Arab Gulf J. Scient. Res., **10**(3), pp. 57-79 (1992)

Upgrading of Abu-Tartur Phosphate Ore

Galal Abd El-Azim Ibrahim[1], Shawki Abd El-Wahab Mabrouk[1], Elsayed Mohamed Abd El-Rasoul[2] and Mohamed Husain Abu Ali[1]

[1]Mining and Metallurgical Department, [2]Mechanical Department, Faculty of Engineering, Assiut University, Assiut, Egypt

ABSTRACT. Aiming to increase P_2O_5%, to decrease Fe_2O_3, MgO, and to remove the impurities such as pyrite, sulphates, carbonates, organic matter and clay material from Abu-Tartur phosphate ores, a series of calcination and flotation experiments for the deslimed ground and subjected to attrition ore was made. Two variants were tried to treat the ore.

1. The first variant: The ore was treated by crushing and screening to be divided into three parts:
   a) Part one contains the large fraction (+2mm.) and was neglected.
   b) Part two contains the medium fraction (−2 + 0.2 mm.) and was wet subjected to attrition, deslimed by hydrocyclone to eliminate sizes −0.08 mm., and then was calcined at 900°C for 30 min, before quenching.
   c) Part three contains the fine fraction (−0.2 mm.) and was deslimed by the hydrocyclone (to eliminate size −0.08 mm.) calcined and quenched under the same previous conditions.
2. The second variant: The same steps such as variant I were applied with respect to parts a & b the only difference is that part c in this variant was treated by flotation instead of calcination. The concentrate obtained from the first variant represented 58.96% of the ore, in which P_2O_5 = 32.41% and MgO= 0.94%, while the concentrate of variant 2 represented 58.64% of the ore, and contains 32.32% P_2O_5 and 0.42% MgO. The rate of water consumption per ton treated ore was calculated for the two variants. It was found to be 1.2, and 1.0 cubic meters respectively.

The two size fractions +2 & −0.08 mm represented 8.06% & 10.33% of the ore by weight respectively and are relatively low grade with respect to P_2O_5 (20%) & high MgO content (2.8%) and are considered as tail to be rejected.

The reserves of Abu-Tartur ores (at 50 km. west of El-Kharga the principal town of the New Valley Governorate) represents more than 30% of all the phosphate ore reserves of Egypt. More attention is now paid to this ore because of its better grade and its existance in a relatively limited area and in a regular seam of thickness 1.5-4m. The underground water table is relatively low in this region. McCune (1980) showed that the ore can also be directly used as a fertilizer in its fine ground form. Yosseif (1979) by the minerallogical investigations for thin and polished sections showed that the ore is composed of collophane pellets and piodeterial phosphate grains cemented by clayey or calcereous cement and carbonate minerals most probably dolomite with few calcite, and material. Other than the phosphate grains and clayey and carbonate cement, pyrite, quartz, gypsum, and iron oxides are recorded. Sofremine-Alusuiss (1982) shows that for beneficiation, the impurities whose removal is aimed at are iron, aluminium, magnesium, calcium carbonate, chlorine, silicon oxide, & organic matter. There are also florine, sodium, and potassium in the phosphated material. Iron in the form of sulphides or oxides is mainly within the phosphatic elements and can therefore be separated in a partial way through the usual physical process, while Aluminium is mainly found in clay which can be removed by deslimining after pulping the ore and subjecting it to attrition. Magnesium is linked to montmorillonite soluble sulphates and as it is a highly undesirable impurity should be removed from fractions below 200 microns.

Many workers had investigated the beneficiation and upgrading of this ore. Among them are (Abu-Gharib and El-Shafi 1979, Abu-Gharib *et al.* 1980, Sofremine-Alusuiss 1977, 1980, Gock 1984, Galal Abd El-Azim *et al.* 1990 and Mohamed Hussien Abu Ali 1987). Wassef 1977 by dry processes were able to raise P_2O_5% from 26.3% to 28.3% at a recovery of 75.5%, while they were by washing & desliming able to have a concentrate contain 29% at a recovery of 90%. They by using a combined washing & flotation technique raise P_2O_5% to 30% at recovery of 80%. Abu Gharib and El-Shafi 1979 by agitation of fine (−0.5 mm) at speed 1400 R.P.M. at solid/liquid ratio 1:1 for 10 min. and deslimizing (size −0.05 mm) by decantation separate 81.72% of the ore as concentrate having 29.98% P_2O_5 at recovery 89.74%. By washing-flotation processes they raise P_2O_5 up to 30% at a recovery of 88.62%, which by roasting of the washed phosphate at 900°C for about 30 min., quenching and removing the formed calcium & magnesium hydroxides raise P_2O_5% to 33.41% at recovery of 85.97% with a weight yield 70.28% and MgO% of 0.63%. Abu Gharib *et al.* 1980 by attritioning, roasting, magnetic separating and flotation raise P_2O_5 to 31.78% at recovery of 50.4%. Sofremine - Alusuiss 1977, 1980 by hand picking, blending, pulping, scrubbing, screening, attritioning, desliming, wet classification and magnetic separation of the fine fraction was able to concentrate Abu-Tartur ore into 58% of the ore having 31.5% P_2O_5 at a recovery of 73% & MgO = 0.5%. Gock (1984) by crushing thermic

pretreatment in a rotary kiln at a temperature of about 400°C, pneumatic classification, two stages of dry high voltage magnitic separations in between a second thermic pretreatment was able to raise P_2O_5% up to 32.2%. A 4 variants flowsheets for the concentration of Abu-Tartur phosphate ore using crushing, screening, attritioning, desliming, classifications and tabling in different arrangements were investigated by Galal *et al.* 1990 and Mohamed Hussien Abu Ali 1987, P_2O_5% and recoveries in the concentrates of the 4 variants were 28.5% at 54.5%, 28.04%, at 81.4%, 31.1% at 75% and 30.77% at a recovery of 79.59%.

Experimental Work

The following steps were followed for the preparation of the ore:

i) The ore was hand picked, and primary crushed, by a Denver Jaw crusher with a discharging opening (set) 10 mm.

ii) Further crushing of the ore using a roll crusher having a distance between the rolls (set) 2 mm. Then screening of the ore on a 2 mm. sieve, the over screen was recrushed by the roll crusher with the same set.

iii) A representative sample was taken from the crushed ore for chemical and screen analysis. The gravimetric method was used in the chemical analysis with an accuracy ± 0.3. Part of the samples were chemically analyzed in the laboratory of the fertilizer factory of Mancabad-Assiut while the other part of the samples were analyzed in the laboratory of the Mining and Metallurgical Department, Faculty of Engineering, Assiut University by the same gravimetric method. The ore chemical analysis is shown in Table (1).

iv) The ore was fed to a double deck vibrating screen (2,0.2 mm) to divide it into 3 fractions. A representative sample was taken for chemical analysis from each fraction. The results of the chemical analysis were as follows:

1. Part one contains the large fraction (+2 mm), represent 7.6% of the ore, having the following chemical analyses, P_2O_5 = 8.46%, MgO = 3.26%, Fe_2O_3 = 3.28%, Al_2O_3 = 4.54%, and I.R. = 19.84%, and this fraction was rejected.
2. Part two contains the medium fraction (−2+0.2 mm) have a reasonably higher P_2O_5% and a moderate content of the other oxides.
3. Part three contains the fine fraction (−0.2 mm), has medium to poor P_2O_5% while MgO and Al_2O_3 increased towards the finer fractions.

Table 1. Abu-Tartur phosphate ore chemical analysis. (All chemical analysis ore on wet basis)

Compound	Moisture content	P_2O_5	MgO	Fe_2O_3	Al_2O_3	CaO	I.R	Cl	Sulphate
Assay %	2.52	24.0	1.85	2.7	1.58	38.5	15.28	0.14	2.10

A– Concentration of the medium fraction (–2+0.2 mm):

A series of experiments were made to determine the optimum time through which the ore was subjected to wet attrition and the desliming conditions. Samples were taken from the deslimed fraction and were calcined in a muffle furnace using porcelain dishes. Calcined samples were quenched using a definite quantity of water. The quenched sample was filtered, dried at 110°C, weighed, analyzed for P_2O_5, MgO, Fe_2O_3 and I.R. The calcination was performed under different conditions such as:

1. The temperature was raised from 500°C step wise by 50°C up to 1000°C.
2. The calcination time was changed from 10-50 min. by increments of 10 min.
3. The quantity of water used for quenching was changed from 25 up to 125% of the weight of the sample by steps of 25%.

B– Concentration of the deslimed fine fraction (–0.2+0.08 mm):

This fraction was tested for concentration by two different methods,

a) By calcination

The fine deslimed phosphate ore fraction (–0.2 + 0.08 mm) was calcined by using the muffle furnace as mentioned in the calcination of the medium deslimed ore under the obtained optimum following conditions:

i) Calcination time	30	min.
ii) Calcination temperature	900	°C
iii) Water/ore % for quenching	75	%

b) By Flotation

The deslimed fine fraction ore (–0.2+0.08 mm) was fed into a 3 liter flotation machine working at 1425 R.P.M. and at constant air rate introduced in it, in batch

process in order to decrease the magnesium oxide and to raise P_2O_5%. The effect of the pH, solid/liquid ratio, conditioning time, flotation time, collector, frother and depressant dosage on the phosphate concentration was studied through several sets of experiments. One parameter was changed in each set while the other parameters were kept constant. Sodium hydroxide, oleic acid, denatured alcohol, and a mixture of Aluminium sulphate and sodium potassium tartrate at a ratio 1:2 were used as pH regulator, collector, frother and depressant for the phosphate materials respectively. The variation was as follows:

pH	6, 7, 8, 9, 10
collector dosage kg/t	0.75, 1.00, 1.25, 1.50, 1.75
frother dosage kg/t	0.50, 0.60, 0.70, 0.90, 1.10
depressant dosage kg/t	(0.25+0.50), (0.35+0.70), (0.4+0.8) (0.5+1.0), (0.60+1.20)
solid/liquid %	12, 16, 20, 25, 30
conditioning time minutes	1, 2, 4, 6, 8
flotation time minutes	1, 3, 5, 8, 10

The different steps performed in each experiment were as follows:

a) The pulp solid/liquid % was adjusted and the agitation begun without introducing air into the cell.

b) The pH was adjusted by sodium hydroxide and conditioning was continued for 5 min.

c) The depressant was added in the proper dosage and the conditioning was continued for 5 min.

d) The pH was controlled, and the collector dosage was added and the pulp was adjusted for 5 min.

e) The frother was added in the needed dosage and conditioning was continued for 50 sec.

f) The air was allowed into the cell and the mineralised froth was skimmed for 8 min.

g) Both the skimmed froth and the depressed parts were filtered separately, dried, weighed, and were chemically analyzed for P_2O_5, MgO, CaO, and I.R.

The water consumption of different processes was calculated in order to compute the rate of water consumption per one ton of the treated ore for the different suggested flow sheets (variants).

Results and Discussion

1. Calcination of the medium fraction (−2+0.2 mm):

Since the desliming of the ore subjected to attrition by the hydrocyclone was found to be insufficient for the needed grade a series of calcination experiments were made on the deslimed ore. It was found that 6 min. is an optimum time for wet attritioning. A series of experiments were made to show the effect of different calcination temperature, time, and water/ore % for quenching.

a) Effect of calcination temperature:

Calcination experiments were performed at different temperatures (500 to 1000°C every 50°C) at constant time (30 min) and water/ore % for quenching (100%), the results are shown in Fig. (1). From this figure it can be seen that as the temperature of calcination increases the MgO % content and the recovery of P_2O_5 and the yield decreases while, the P_2O_5 % increases. It can be observed also that the best calcination temperature is 900°C. Table (2,a) gives the % and recoveries of the different elements for this temperature.

b) Effect of time of calcination:

Several experiments were made to study the effect of calcination time. In these experiments both temperature and water/ore ratio for quenching were kept constant at 900°C and 100% respectively. Time of calcination was varied from 10 to 50 minutes in steps of 10 min. results are shown in Fig. (2). From these results it can be seen that as the time of calcination increases, the P_2O_5% and recovery increases while MgO% recovery and weight yield decreases. It is noticed also that the best results were obtained at a time of 30 min. Table (2,b) gives the % and recovery of the different elements of this experiment.

Table 2. The result for the optimum calcination parameters used

Symbol	wt. yield %	P_2O_5		MgO		F_2O_3 %	CaO %	I.R.+Sio_2 %
		%	Rec.%	%	Rec.%			
a	84.22	32.31	97.10	0.24	44.22	3.83	42.31	13.40
b	83.30	32.75	97.88	0.23	42.57	3.75	43.81	13.76
c	82.65	32.16	95.37	0.23	42.24	3.42	42.90	13.37

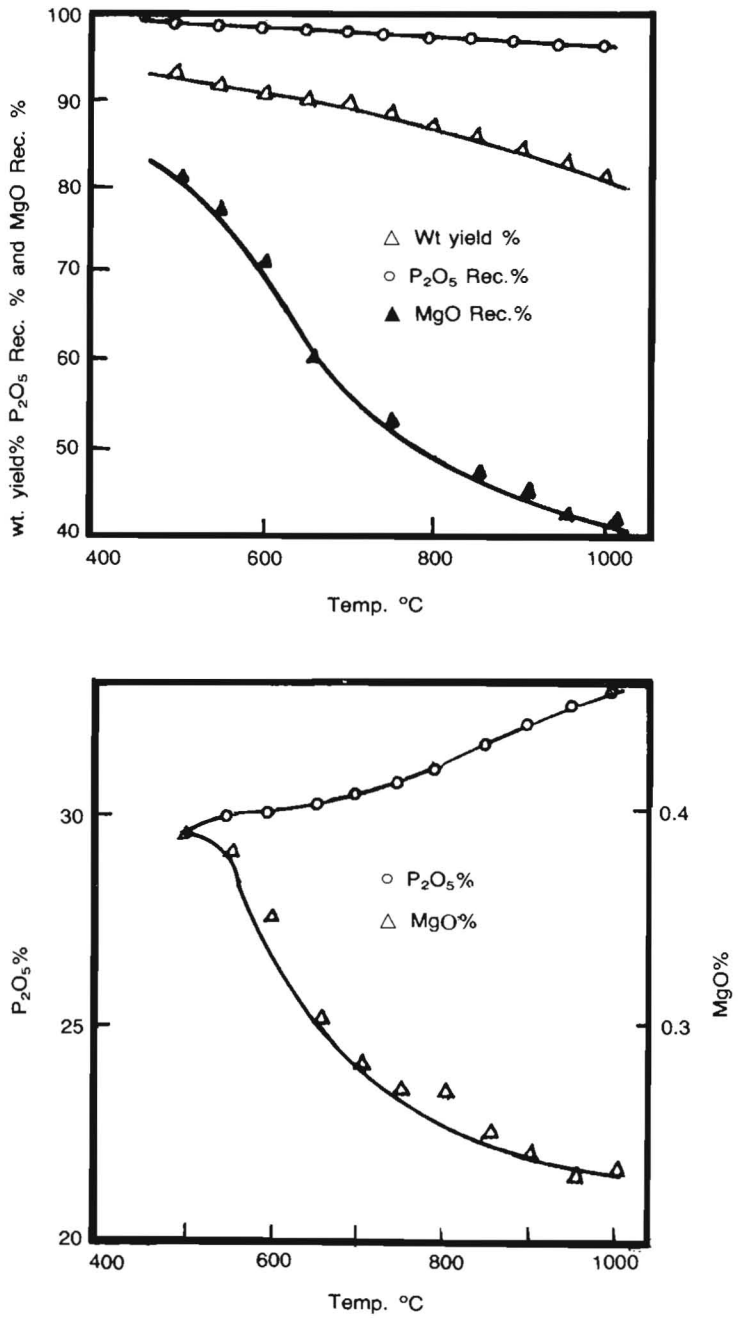


Fig. 1. The effect of the degree of temperature on the wt. %, P_2O_5%, MgO and Rec. % (calcination process).

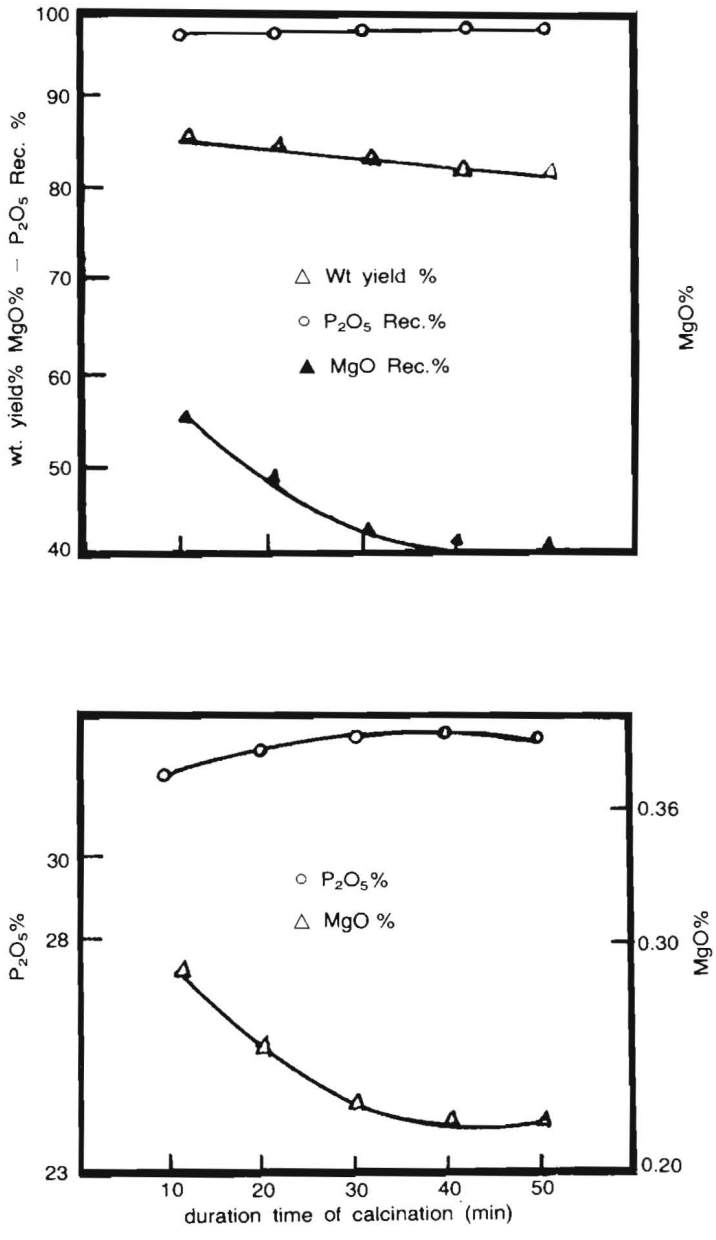


Fig. 2. The effect of the duration time of calcination on wt. %, P_2O_5%, MgO and Rec. % (calcination process).

c) Effect of water/ore quenching:

Several experiments were made in order to study the effect of water/ore ratio used for quenching the calcined ore. In those experiments, temperature and time of calcination were kept constant at 900°C and 30 min. respectively. The amount of water used for quenching was varied from 25-125%. The results are given in Fig. (3), it is clear that the increase of water/ore % for quenching increases P_2O_5% and decreases each of the P_2O_5 recovery, MgO% and recovery, and the % weight yield. It can be shown also that 75% water/ore % is enough for quenching the calcined ore. Table (2,c) shows the % and recoveries of the different elements.

2. Calcination of the deslimed fine fraction (−0.2+0.08 mm):

The deslimed fine fraction (−0.2+0.08 mm) was calcined at the optimum conditions obtained from the calcination of the deslimed medium fraction, (calcination temperature 900°C, calcination time 30 min., and water/ore % for quenching 75%) by the same method. The obtained concentrate contained 31.44%. P_2O_5, MgO = 1.24%, Fe_2O_3 = 2.33%, CaO = 44.28% and I.R. = 11.29%.

3. Flotation of the deslimed fine fraction (−0.2+0.08):

In order to optimize the flotation process, the different parameters governing the flotation were changed as mentioned before. The effect of these parameters are:

a) The results obtained from the changing of the pH are shown in Fig. (4). From this figure it can be seen that pH = 8 is the best value for dolomite flotation. Table (3,a) gives the results of this experiment.

b) The results obtained during changing of the collector dossage are represented in Fig. (5). From this figure, it can be seen that the optimum results were obtained at collector dossage 1.5 kg/t. Table (3,b) illustrate the results of this experiment.

c) The frother dossage results are represented in Fig. (6), it shows that the optimum concentrate is at 0.6 kg/t, (Table 3,c).

d) The optimum depressant dossage from Fig. (7) is 1.2 kg/t see table (3,d).

e) The solid/liquid effects are shown in Fig. (8) that shows best results at 20% solid/liquid %, which is shown in Table (3,e).

f) Conditioning time results are given in Fig. (9), showing that the best results are at conditioning time 2 min, illustrated in Fig. (3,f).

Table 3. The results of the optimum flotation parameters used

Symbol	wt. yield %	P_2O_5		MgO		CaO		I.R.+Sio_2%		
		%	Rec.%	%	Rec.%	%	Rec.%	%	Rec.%	
a	16.20	19.52	11.17	6.97	56.92	32.77	13.26	16.61	20.68	float
	83.80	30.13	88.83	1.02	43.08	41.44	86.74	12.41	79.32	sink
b	15.60	14.15	7.93	9.36	63.54	29.76	11.92	15.00	17.93	float
	84.40	30.37	92.07	0.98	36.16	40.62	88.08	12.70	82.07	sink
c	14.00	15.84	7.83	9.39	59.75	28.15	10.05	15.88	16.72	float
	86.00	30.34	42.17	1.03	40.25	41.00	89.95	12.88	83.28	sink
d	15.20	17.13	9.26	8.45	59.76	32.76	12.67	14.84	17.24	float
	84.80	30.09	90.74	1.02	40.24	40.67	87.33	12.76	82.76	sink
e	16.30	16.03	9.23	7.80	60.54	33.34	13.76	16.51	20.01	float
	83.70	30.70	90.77	0.99	39.46	40.70	86.24	12.86	79.99	sink
f	14.43	13.73	7.16	8.83	59.82	28.71	10.59	20.77	22.22	float
	85.59	30.03	92.84	1.00	40.18	40.89	85.41	12.26	77.78	sink
g	15.09	16.49	8.78	8.62	60.50	28.88	11.21	17.06	19.01	float
	84.91	30.45	91.22	1.00	39.50	40.66	88.79	12.92	80.99	sink

g) Fig. (10) illustrates the effect of flotation time on the flotation of the carbonate. It was found that 8 min. was enough for the carbonate flotation. Table (3,g) illustrate the results of this experiment.

Two flow-sheets 1 and 2 are given here to illustrate the different experiments performed within the two suggested variants for the concentration of Abu-Tartur phosphate ore. The consumed water per ton was computed for each variant. It was found that for variant I the rate of consumption is 1.2 m^3/t, while for variant II the rate was 1.0 m^3/t of treated ore.

Conclusion

1. The two size fractions of +2 mm and −0.08 mm. represented 8.06 and 10.33% of the ore by weight respectively and are relatively low grade with respect to P_2O_5 (20%) and high MgO content (2.8%). Therefore these two fractions are considered as tail and can be rejected.

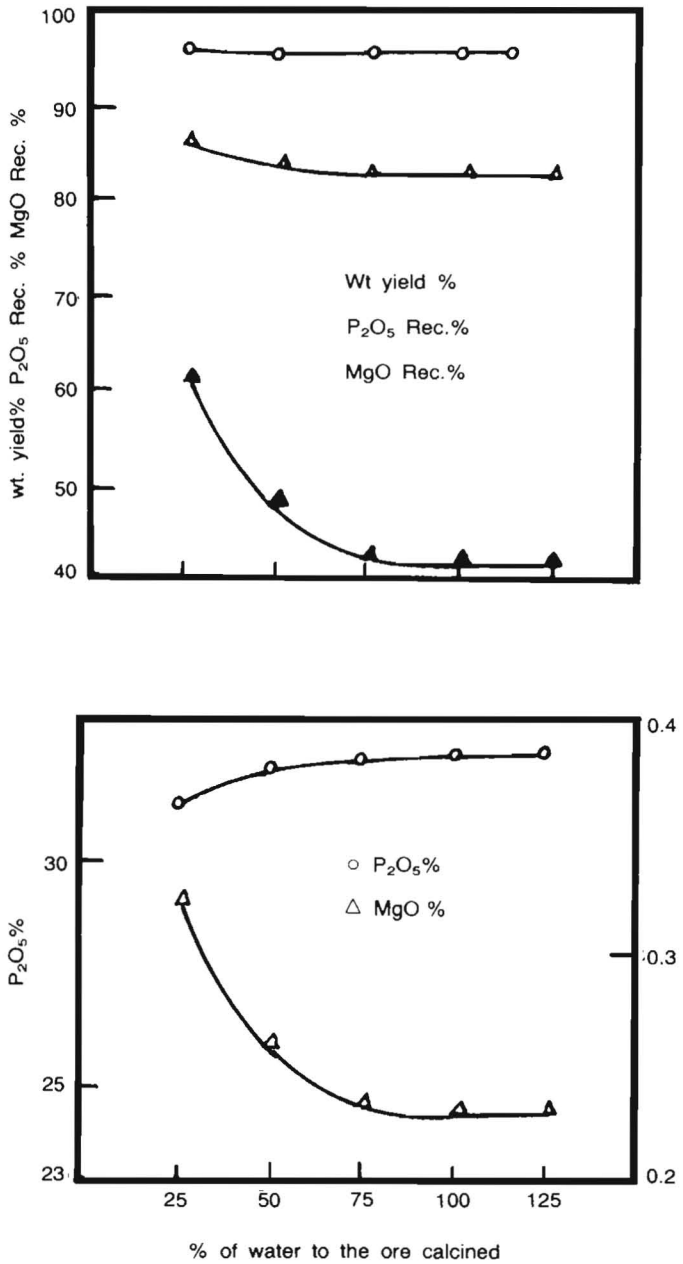


Fig. 3. The effect of the quantity of water for quenching on the wt. %, P_2O_5%, MgO and Rec. % (calcination process).

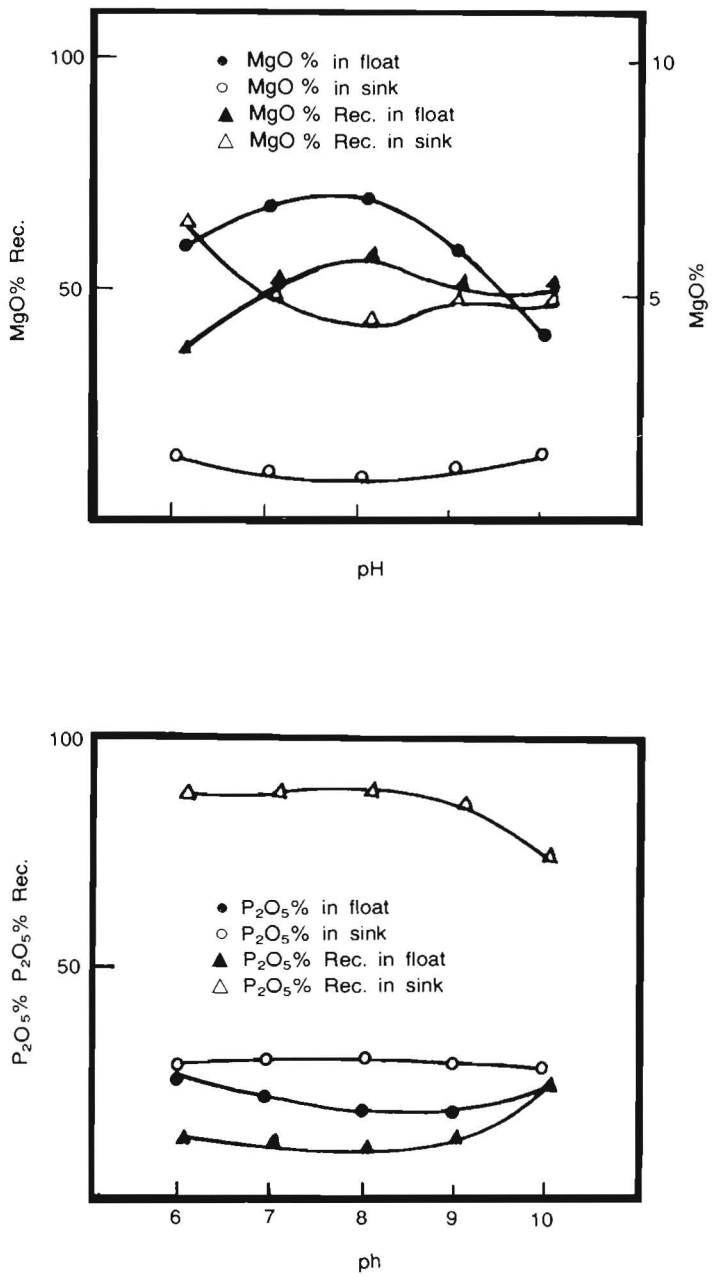


Fig. 4. Effect of pH on dolomite flotation.

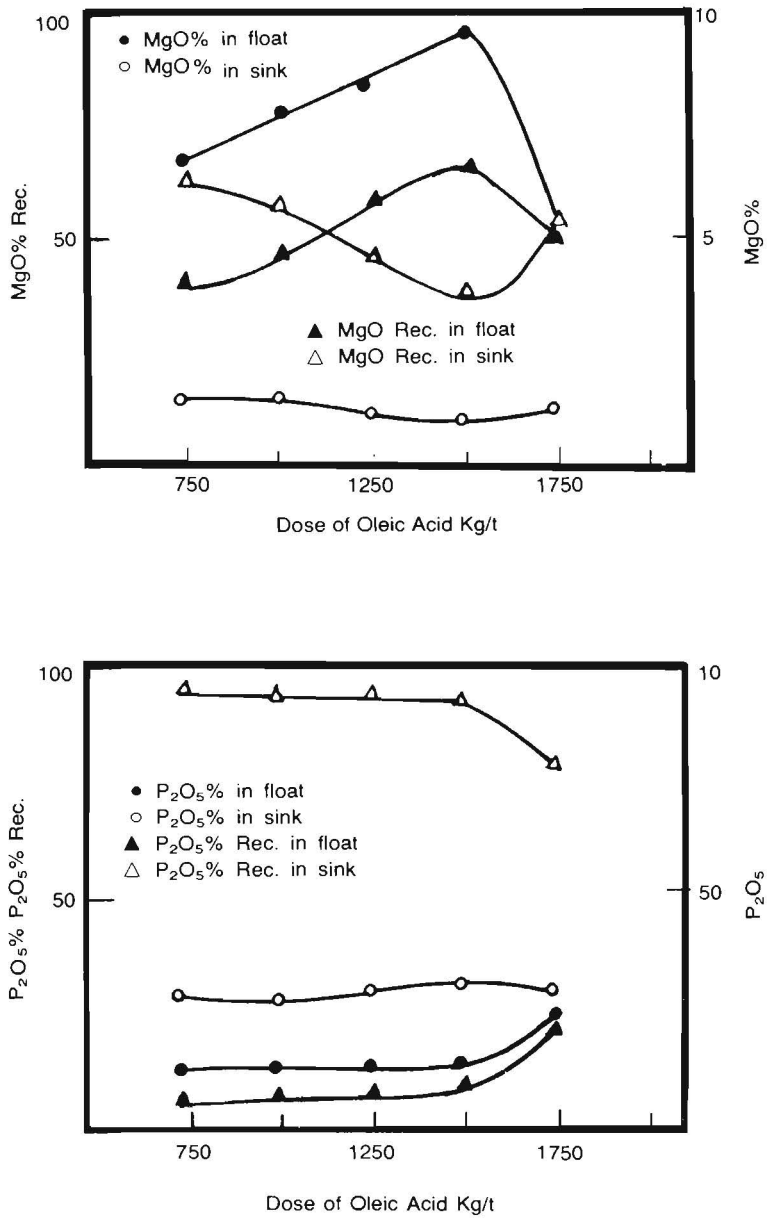


Fig. 5. Effect of dose of collector on dolomite flotation.

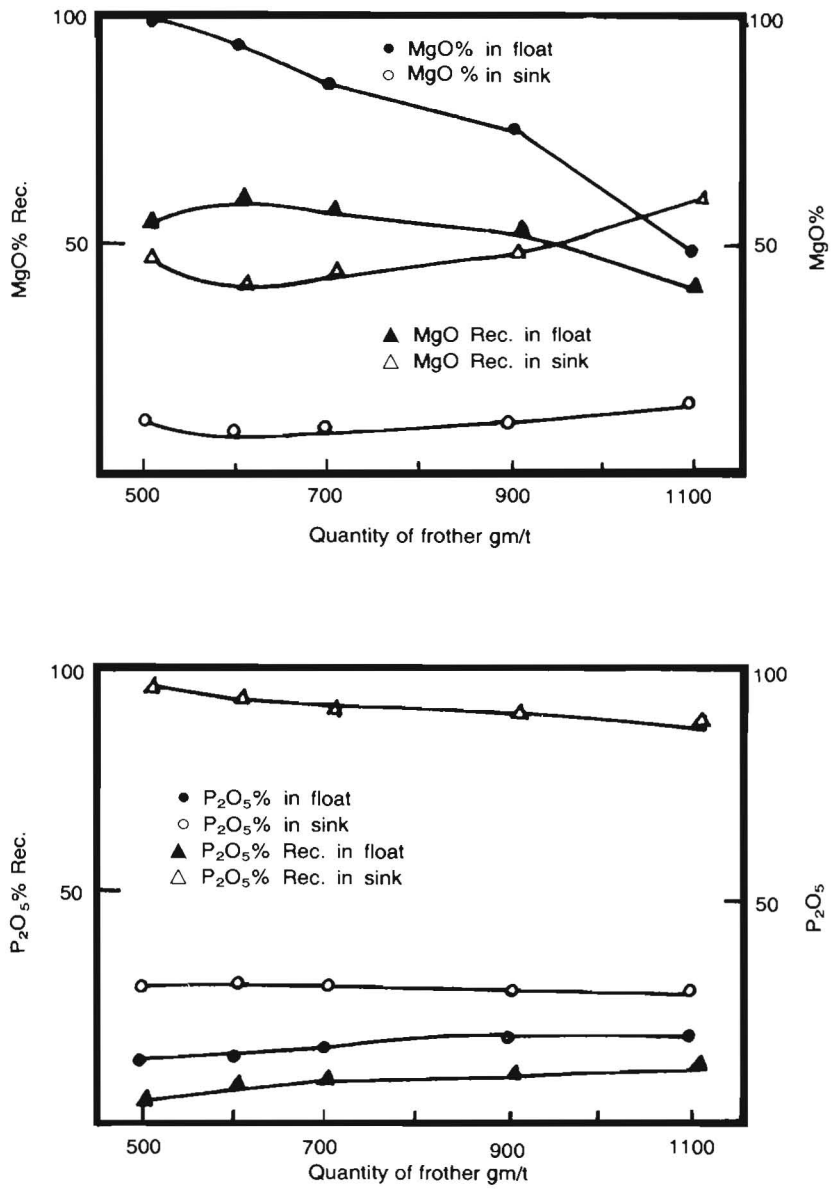


Fig. 6. Effect of dose of frother on dolomite flotation.

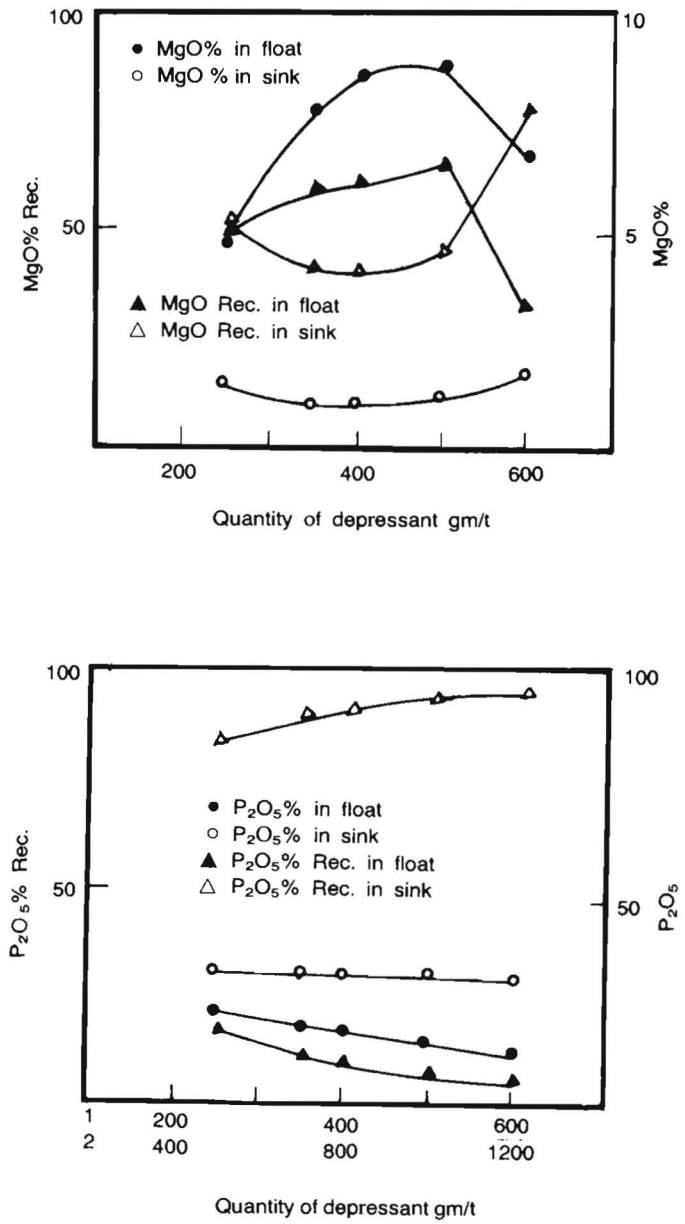


Fig. 7. Effect of dose of depressant on dolomite flotation.

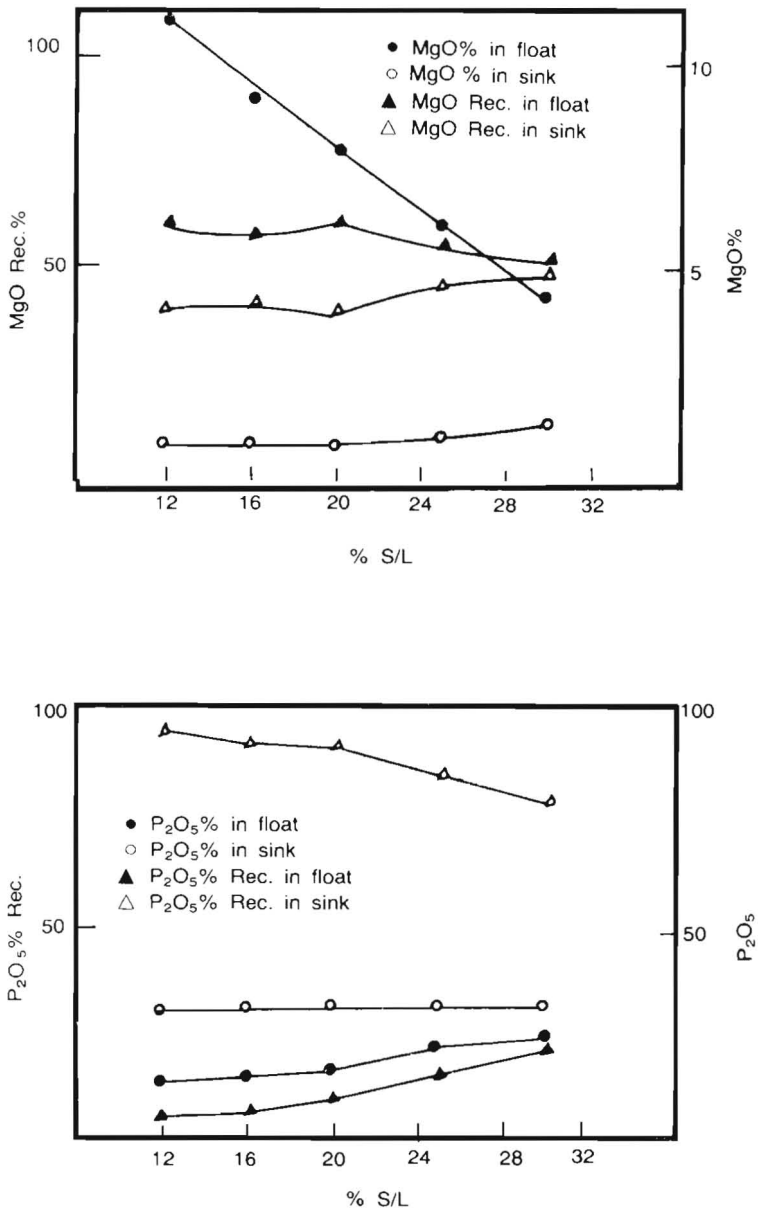
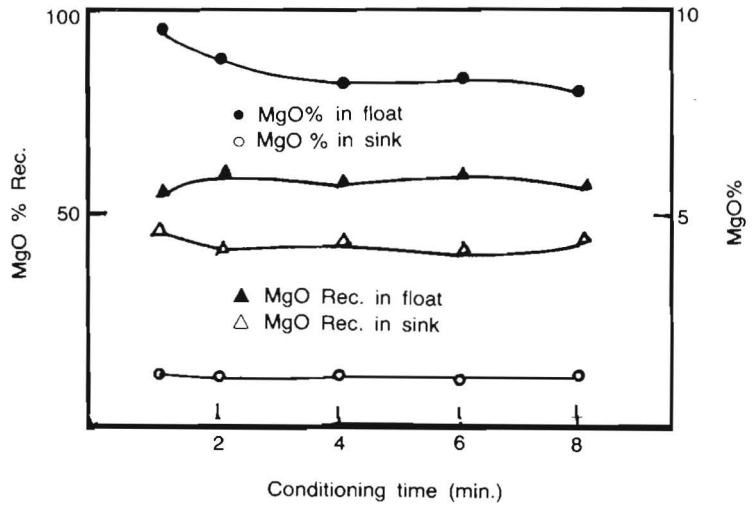


Fig. 8. Effect of S/L% on dolomite flotation.



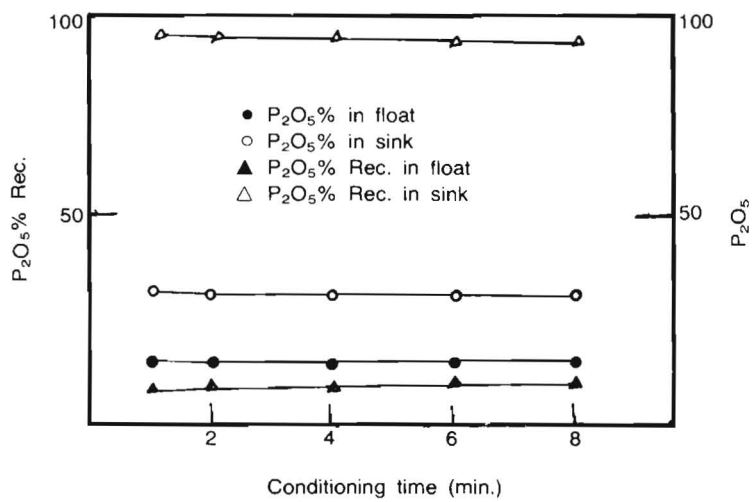
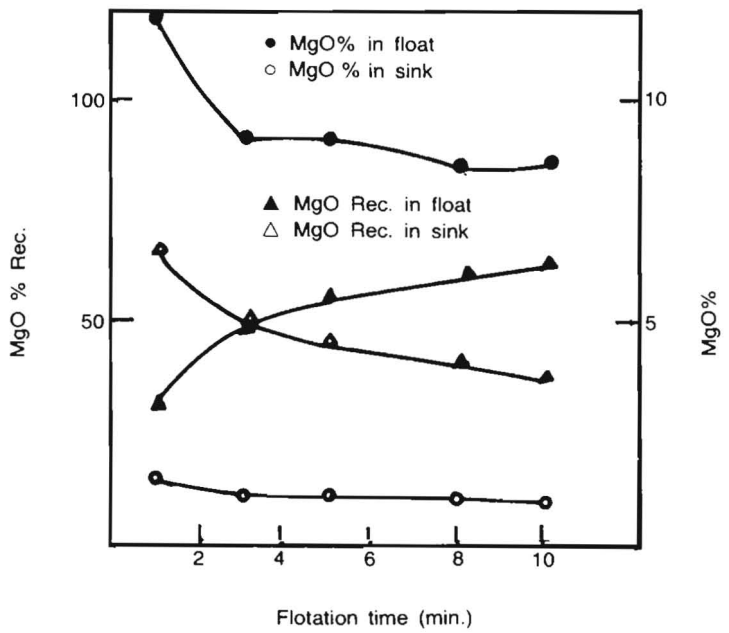


Fig. 9. Effect of conditioning time on dolomite flotation.



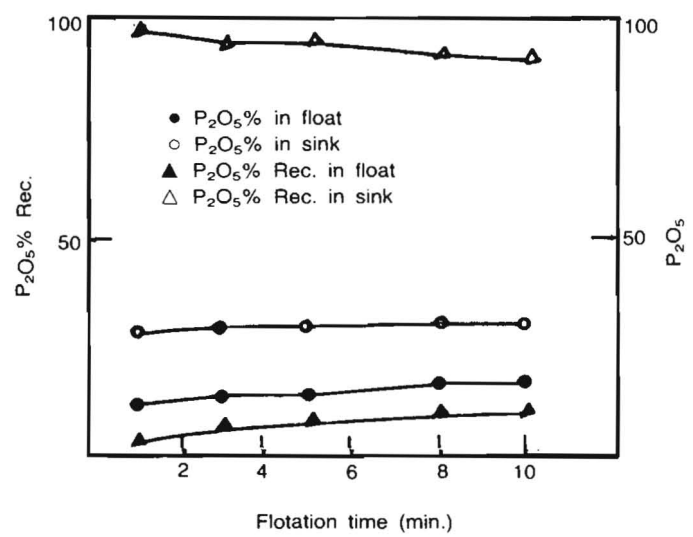
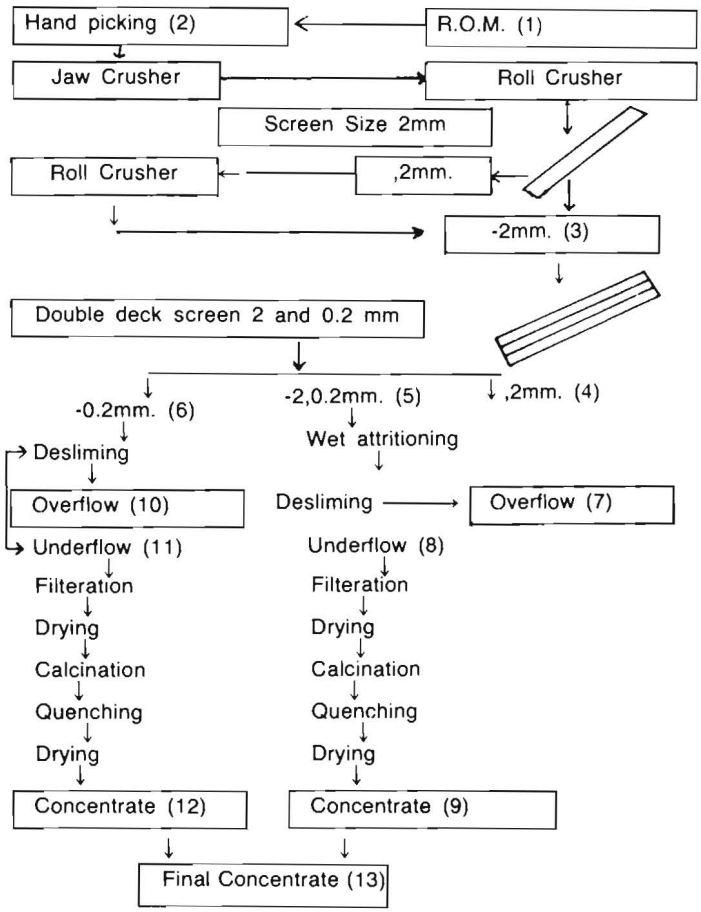


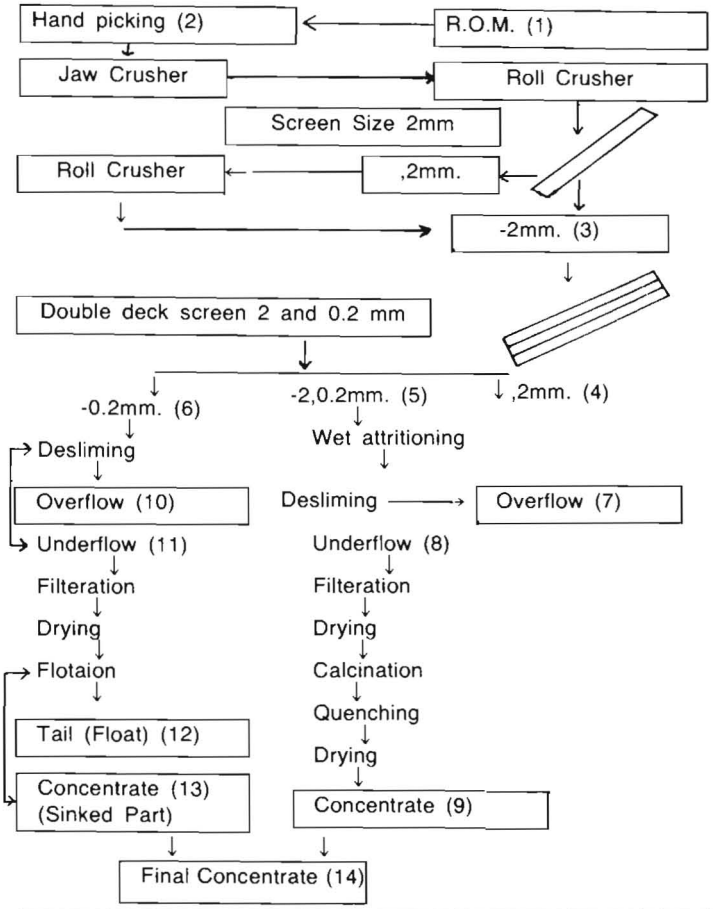
Fig. 10. Effect of flotation time on dolomite flotation.



No.	Wt. %	P_2O_5%	MgO%	Fe_2O_3%	CaO%	I.R.%
1	100.00	24.00	1.85	2.70	38.50	15.30
2	5.40	14.00	3.78	3.16	28.20	18.20
3	94.60	24.48	1.73	2.67	39.02	15.11
4	7.60	8.46	3.36	3.28	31.12	19.84
5	58.50	26.12	0.91	2.63	40.10	14.31
6	28.50	24.47	3.11	2.61	38.13	14.50
7	6.39	11.96	4.39	3.86	29.40	16.07
8	52.11	27.87	0.45	2.40	40.69	13.76
9	43.90	32.75	0.23	3.46	43.81	13.02
10	10.83	16.44	4.48	2.79	35.74	16.92
11	17.67	28.56	2.17	2.43	39.32	13.65
12	15.06	31.44	1.24	2.33	44.28	11.29
13	58.96	32.41	0.49	3.17	43.93	12.58

Flowsheet (1)

Treatment of Abu-Tartur Phosphate Ore Variant (I). Rate of Water consumption $1.2m^3/t$. treated ore.



No.	Wt. %	P_2O_5%	MgO%	Fe_2O_3%	CaO%	I.R.%
1	100.00	24.00	1.85	2.70	38.50	15.30
2	5.40	14.00	3.78	3.16	28.20	18.20
3	94.60	24.48	1.73	2.67	39.02	15.11
4	7.60	8.46	3.36	3.28	31.12	19.84
5	58.50	26.12	0.91	2.63	40.10	14.31
6	28.50	24.47	3.11	2.61	38.13	14.50
7	6.39	11.96	4.39	3.86	29.40	16.07
8	52.11	27.87	0.45	2.40	40.69	13.76
9	43.90	32.75	0.23	3.46	43.81	13.02
10	10.83	16.44	4.48	2.79	35.74	16.92
11	17.67	28.56	2.17	2.43	39.32	13.65
12	2.93	16.28	8.01	3.44	33.60	17.62
13	14.74	31.03	0.98	2.23	40.77	12.81
14	58.64	32.32	0.42	3.15	43.04	12.96

Flowsheet (2)

Treatment of Abu-Tartur Phosphate Ore Variant (II). Rate of Water consumption 1.0m^3/t. treated ore.

2. The following variants can be submitted at the laboratory scale for the beneficiation of Abu-Tartur phosphate ore:

   variant I : Flow sheet no. 1.

   variant II : Flow sheet no. 2.

   The results showed that the concentrate of each of the two variants are near to each other in both the chemical analysis and the recoveries. Due to this we suggest that each of them be used in the concentration of the ore.

3. The rate of water consumption for each variant was calculated taking into account, the water and reclamation by thickening, hydro-cyclone, and flitration. The results obtained showed that the water consumed in variant II is less than that consumed in variant I (1 & 1.2 m^3/t respectively).

References

Abu-Gharib, E.A. and **El-Shafi, S.B.** (1979) *Upgrading of Abu-Tartur phosphates*, Troisieme conference, Arab pour la richesse mineral, Rabat, Maroc, pp. 119-126.

Abu-Gharib, E.A. ,Mostafa, S.N. and **Dardira, A.A.** (1980) *Beneficiation of Abu-Tartur phosphate ore, Annals of the geological survey of Egypt.* **10:** 1069-1085.

Ibrahim., G.A.E., Shwki, A.M., El-Sayed, M.A. and **Mohamed, H.A.** (1990) *Beneficiation of Abu-Tartur phosphate ore*, Bulletin of the Faculty of Eng., Assiut Univ., Assiut, Egypt, **18:** 111-117.

Gock, E. (1984) *Conception for processing the pyrite bearing phosphorite of Abu-Tartur*, Berliner geowiss. Abh. A, 50, Berlin, pp. 381-397.

McCune, D.L. (1980) *Evaluation of Abu-Tartur phosphate materials for manufacture of selected phosphate fertilizer* (IFDC) Muscle shoals, Alabama, May 1980.

Abu-Ali, M.H. (1987) *Beneficiation of Abu-Tartur phosphate ore with special reference to the minimization of water consumption*, M.Sc. Thesis, submitted to Min. and Met. Dept., Faculty of Eng., Assiut Univ., April.

Sofremine-Alusuiss (1977) New valley phosphate project overall project apprisal report, p. 1-50.

Sofremine-Alusuiss (1980) *New valley phosphate project Abu-Tartur*, Sec. III, July.

Sofremine-Alusuiss (1982) *New valley phosphate project Abu-Tartur*, Sec. III, Vol. 1, July.

Sofremine-Alusuiss (1982) *New valley phosphate project Abu-Tartur*, Sec. III, September, p. 10.

Wassef, A.S. (1977) *The laboratory technological tests of phosphorits ore from Abu-Tartur deposits*, Annals of the Geological Survey of Egypt, **7:** 61-68.

Yosseif, A.S. (1979) *Laboratory wet beneficiation tests*, National Research Centre, Cairo, March.

(Received 19/11/1990;
in revised from 12/04/1992)

تركيز خام فوسفات أبو طرطور

جلال عبدالعظيم ابراهيم[1] و شوقي عبدالوهاب مبروك[1]
السيد محمد عبدالرسول[2] و محمد حسين أبو علي[1]

[1]قسم هندسة التعدين والفلزات ـ و[2]قسم الهندسة الميكانيكية ـ كلية الهندسة
جامعة أسيوط ـ أسيوط ـ مصر

بهدف رفع نسبة خامس اكسيد الفوسفور P_2O_5 وتقليل نسبة اكاسيد الحديد والالومنيوم والمغنسيوم مع إزالة بعض الشوائب مثل البيريت والكبريتات والكربونات والمواد العضوية والطينية الموجودة في خام فوسفات أبو طرطور، أجريت مجموعة من تجارب الكلسنة والتعويم بالهواء على الخام المطحون والمعرض لعملية فرك بعد فصل الناعم منه (– ٠,٠٨ مم) وقد تم إستنتاج اختبارين لتركيز الخام.

١ ـ الاختبار الأول وفيه يُعالَج الخام بالطحن والنخل حيث يقسم الخام إلى ثلاثة أجزاء :

(أ) الجزء الأول ويحتوي على الأحجام الكبيرة من الخام (+ ٢ مم) ويهمل لصغر نسبة أكسيد الحديد Fe_2O_3 فيه.

(ب) الجزء الثاني ويحتوي على الأحجام المتوسطة من الخام (– ٢ + ٠,٢ مم) وتجرى عليه عمليات فرك مبلل ثم تُزال منه الحبيبات ذات الأحجام الصغيرة (– ٠,٠٨ مم)، ثم يُكلسن عند درجة ٩٠٠° م لمدة ٣٠ دقيقة ويطفأ بالماء.

(جـ) الجزء الثالث ويحتوي على الأحجام الناعمة (- ٢ , ٠ مم) ويُزال منه الناعم (- ٠٨ , ٠ مم) باستخدام الهيدروسيكلون ثم يُكلسن الباقي تحت نفس الظروف السابقة ويُطفأ بالماء.

٢ ـ الاختبار الثاني وهو شبيه بالاختبار الأول فقط بدلاً من كلسنة الجزء الثالث تجرى عليه عمليات تعويم بالهواء.

وقد أمكن الحصول في الاختبار الأول على ركاز يحتوي على ٥٨,٩٦ ٪ من الخام وفيه نسبة خامس أكسيد الفوسفور ٣٢,٤١ ٪ . ونسبة أكسيد المغنسيوم MgO ٠,٩٤ ٪ ، أما الاختبار الثاني فيحتوي ركازه على ٥٨,٦٤ ٪ من الخام ونسبة اكسيد الفوسفور فيه ٣٢,٣٢ ٪ بينما نسبة أكسيد المغنسيوم فيه ٠,٤٢ ٪ . وقد أجريت حسابات لمعدل إستهلاك المياه في كل اختبار وكانت بالنسبة للاختبار الأول ١,٢ م^{٣}/ طن من الخام المعالج بينما كانت بالنسبة للاختبار الثاني ١,٠٠ م^{٣}/ طن من الخام المعالج.

وقد اعتبرت الأجزاء من الخام + ٢ مم، - ٠٨ , ٠ مم والتي تبلغ نسبتها ٨,٠٦ ٪ ، ١٠,٣٣ ٪ من الخام والتي نسبة خامس أكسيد الفوسفور منخفضة (٢٠ ٪) ونسبة أكسيد المغنسيوم مرتفعة نسبياً (٢,٨ ٪) نفايات.