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A study on the effect of active pyrite on flotation of porphyry copper ores

Nahid Molaei ¹, Fatemeh Sadat Hoseinian ², Bahram Rezai ²

¹ Department of Mining Engineering, Research and Science Campus, Islamic Azad University, Poonak, Hesarak Tehran, Iran

² Department of Mining and Metallurgical Engineering, Amirkabir University of Technology, Tehran, Iran

Corresponding author: f_hoseinian@aut.ac.ir (Fatemeh Sadat Hoseinian)

Abstract: Active pyrite is one of the most undesirable phenomena in the flotation of porphyry copper ores. Misreported pyrite into copper concentrates decreased the grade and recovery of copper. In this study, the effective parameters on the flotation process including grinding condition and chemical parameters were evaluated in order to decrease the active pyrite recovery by flotation. Firstly, optical microscopic and grinding studies were carried out to determine the optimal particle size and grinding time. The results showed that 43 minutes of grinding is necessary to achieve the optimum liberation degree of 53 μm for flotation. Then, the flotation effective parameters such as pH (7.3, 10, 10.5, 11, 11.5 and 12), collector type (Nascol, Aero 407, Aero 3477 and X231), collector concentration (12 and 25 mg/dm^3), depressant concentration (0 and 25 mg/dm^3) and frother concentration (0 and 25 mg/dm^3) were investigated in a Denver-type laboratory flotation cell with a constant capacity of 2.5 dm^3 . The results showed that the optimal conditions for chalcopyrite flotation were pH of 11.5, Aero 407 as a collector with concentration of 25 mg/dm^3 , Dowfroth 250 (DF250) as a frother with concentration of 25 mg/dm^3 and Na_2SO_3 as a depressant with concentration of 25 mg/dm^3 . The type of collector had greater effect on the chalcopyrite flotation than the other parameters. The recovery and grade of chalcopyrite and pyrite were obtained as 79.95%, 49% and 5.3%, 7.98% using the Aero 407, respectively. Under the optimum conditions, the grade of final concentrate increased from 0.94% to 21.3% with three cleaner stages.

Keywords: Active pyrite, Chalcopyrite, Collector, Flotation, Optimization

1. Introduction

Chalcopyrite is often associated with pyrite which is the most undesirable mineral with the same floatability (Chandraprabha et al., 2004). The selective separation of chalcopyrite from pyrite is very difficult due to several interactions that occur at the mineral surfaces during the processes of grinding and conditioning. Flotation of copper sulphides is a traditional technique for processing the sulphide copper. The presence of pyrite in copper concentrates (especially the pyrite particles that are surrounded with the chalcopyrite) is the most important problem of copper ore processing that leads to a decrease in the quality and grade of concentrate. It also creates an increase in the cost of the financial and ecological in the next steps such as smelting process (Gu et al., 2010). Table 1 shows some recent studies of copper ore flotation to separate chalcopyrite from pyrite with the evaluation of the effect of different important parameters on the process efficiency.

Researchers have investigated (Martin et al., 1989, Leppinen, 1990, Boulton et al., 2001, Peng et al., 2003, He et al., 2006, Peng and Grano, 2010, Owusu et al., 2011, Chen et al., 2011, Owusu et al., 2013, Chen et al., 2013, Agorhom et al., 2014, Wang et al., 2015, Zhong et al., 2015) efficient parameters which have been useful for separating pyrite from chalcopyrite. Some researchers focused on the type

Table 1. The effect of various parameters on the selective separation of chalcopyrite from pyrite

Row	Subject	Parameter	Date	Result
1	The impacts of using nitrogen were investigated instead of air on the flotation of activate copper and separation of sulphides from ores (Martin et al., 1989).	Nitrogen	1989	Using nitrogen increased the floatability of pyrite.
2	The polyacrylamide polymers were used to separate the copper-activated from pyrite along with isobutyl xanthate (Boulton et al., 2001).	Polyacrylamide	2001	Using the polyacrylamide leads to depressing the pyrite during the flotation.
3	The special mill was used for controlling the pH for separating chalcopyrite from pyrite (Peng et al., 2003).	Chromium grinding medium	2003	Using Chromium grinding medium made the high recovery of chalcopyrite, compared with the mild steel grinding medium.
4	The performances of oxidation potential (Eh) and zinc sulphate were considered in order to separate the chalcopyrite and pyrite at pH=9 (He et al., 2006).	Oxidation potential (Eh) and zinc sulphate	2006	Adding zinc sulphate improved the separation chalcopyrite from pyrite by depressing pyrite.
5	The effect of the grinding media conditions were investigated (Peng and Grano, 2010).	The grinding media	2010	The grinding media led to a decrease in the amount of floating pyrite.
6	The effects of the amount of pyrite on the chalcopyrite recovery with X-ray photoelectron spectroscopy (XPS) were considered (Owusu et al., 2011).	The effect of the pyrite amount	2011	XPS results confirmed that the activation of pyrite can be increased with increasing the pyrite amount that leads to a decrease in the chalcopyrite recovery.
7	Sodium humate was used for depressing pyrite and separate pyrite from chalcopyrite (Chen et al., 2011).	Sodium humate	2011	Adding sodium humate depressed pyrite and separated it from chalcopyrite.
8	The flotation behavior of pyrite and chalcopyrite by using redox potential (Eh), dissolved oxygen (DO) and pH were investigated (Owusu et al., 2013).	Redox potential (Eh), dissolved oxygen (DO) and pH	2013	The recovery and grade of chalcopyrite were improved by using redox potential (Eh) and dissolved oxygen (DO).
9	The impacts of regrinding circumstances on activated copper and pyrite flotation in the cleaner step were investigated (Chen et al., 2013).	Regrinding conditions	2013	Regrinding conditions led to depressing the amount of pyrite by using the mild steel media during regrinding conditions.
10	The effects of diethylenetriamine (DETA) in different situations (under aerated and non-aerated) on Cu-activated pyrite hydrophobised by xanthate were assessed (Agorhom et al., 2014).	Diethylenetriamine (DETA)	2014	Due to the presence of DETA, the depression of pyrite increased due to the increase in oxidation of surface copper.
11	Using sodium glycerine-xanthate (SGX), the floatability of pyrite decreased during flotation of chalcopyrite and using sodium butyl xanthate (SBX) as a collector were investigated (Wang et al., 2015).	Sodium glycerine-xanthate (SGX) and butyl xanthate (SBX)	2015	By using sodium glycerine-xanthate (SGX), the floatability of pyrite decreased during flotation of chalcopyrite and using sodium butyl xanthate (SBX) as a collector led to an increase in the floatability of chalcopyrite.
12	The flotation behavior of chalcopyrite, pyrite, galena and sphalerite was considered by using sodium diisobutyl dithiophosphate (DTPiNa) (Zhong et al., 2015).	Sodium diisobutyl dithiophosphate (DTPiNa).	2015	The recovery of chalcopyrite increased significantly compared with the recovery of pyrite

of grinding, regrinding (Chen et al., 2013) and the type of mill and crusher (Peng et al., 2003, Li and Gao, 2017). They showed that the grinding condition, size distribution, liberation degree and pulp aeration have a positive impact on the separation of chalcopyrite. The chromium grinding media improved the recovery of chalcopyrite and the depressing of the activated pyrite by changing the pulp potential. Also, appropriate regrinding conditions can decrease the floatability of pyrite. The chemical conditions of pulp such as the pH, type and concentration of reagents (collectors, depressants and frothers) also are the effective factors on the process efficiency. Hassanzadeh and Hasanzadeh (2016) evaluated the effect of three chemical reagent configuration on selective flotation in low and high pyritic copper sulphide ores. They showed that the collector of isopropyl-n-ethyl thionocarbamate creates the highest copper grade and the lowest iron recovery (Hassanzadeh and Hasanzadeh, 2016). Qi et al. (2017) studied the selective collector of N-isopropoxypropyl-N'-ethoxycarbonyl thiourea to separate the pyrite from chalcopyrite and molybdenite. They indicated that the collector of N-isopropoxypropyl-N'-ethoxycarbonyl thiourea exhibits excellent selectivity towards chalcopyrite against pyrite (Qiu et al., 2017). Li et al. (2012) investigated the depression of pyrite in alkaline medium using NaOH and CaO and its subsequent activation by copper. The results showed that the Cu activation of pyrite after depression using CaO was more difficult than that after depression using NaOH (Li et al., 2012). He et al. (2005) evaluated the effect of pulp oxidation potential on the copper-activated pyrite flotation. They demonstrated that the pulp oxidation potential is one of the important factors controlling the copper-activated pyrite flotation so that it can determine the different proportions of hydrophobic and hydrophilic species on the pyrite surface (He et al., 2005).

This study investigated the effect of the most important parameters such as grinding condition (particle size and grinding time) and chemical parameters (pH, type and concentration of collector, frother concentration and depressant concentration) on the chalcopyrite flotation in the presence of activated pyrite. In this regard, optical microscopic and grinding studies were initially performed to determine the optimal particle size and grinding time. Then, the optimal chemical conditions of flotation were determined to achieve the maximum grade and recovery of chalcopyrite and the effect of activated pyrite on the chalcopyrite flotation efficiency was investigated.

2. Materials and methods

The experiments were performed on copper ore obtained from Sarcheshmeh Copper mine, Kerman, Iran. The representative ore samples were used to measure particle size distribution by mechanically shaken Tyler sieves, mineralogical characterization using optical microscopic studies, determining species of ore mineralogy by X-ray diffraction and chemical analysis by atomic absorption spectroscopy. Ball mill with 50 percent solid was used for grinding. The results of chemical and mineral composition are shown in Table 2 and Table 3, respectively. The results showed that the main mineral in the sample ore is quartz.

Nascol from Nasco, Aero 407 and Aero 3477 from Sitec and X231 from Hoechst were used as collectors with concentration of 25 mg/dm³. Dowfroth 250 (DF250) from Dow Chemical was used as the frother. Calcium hydroxide Ca(OH)₂ (Merck) was used for adjusting the pH of the pulp. A Denver-type laboratory flotation cell with a constant capacity of 2.5 dm³ and impeller speed at 1500 rpm was used. In each flotation experiment, 550 g of the sample was used. Before performing flotation tests, the pulp was conditioned for 9 minutes: for preparation of the pulp with a certain depressant concentration, it was agitated for 4 minutes; then the collector and frother were added to the pulp with a desired concentration and agitated for 4 and 1 minute, respectively. The flotation experiments were performed for 4 minutes. At the end of the flotation experiment, the concentrate and tail were separately weighed after filtration and drying to calculate the recovery. Chemical analysis of the concentrate and tail fractions were performed by atomic absorption spectroscopy. The recovery (*R*) was calculated by Eq. (1).

$$R = \frac{C.c}{F.f} \times 100 \quad (1)$$

where *R*, *C*, *F*, *c* and *f* are the recovery, concentrate weight, feed weight, concentrate grade and feed grade, respectively.

Table 2. Chemical composition of the feed

Elements	Mass (%)
Cu	0.94
Fe	4.98
Na	3.8
K	3.55
Si	25.45
Al	1.25
other	29.79

Table 3. Mineral composition of the feed

Mineral	Chemical composition
Pyrite	FeS ₂
Chalcopyrite	CuFeS ₂
Bornite	Cu ₃ FeS ₄
Chalcocite	Cu ₂ S
Molybdenite	MoS ₂
Galene	PbS
Feldspar	(K,Na,Ca)Al(Si,Al) ₃ O ₈
Arsenopyrite	FeAsS
Quartz	SiO ₂

3. Results and discussion

3.1 Microscopic studies

Microscopic studies are the most important methods for the selection of separation methods. They can be applied for liberation degree to confirm the XRD studies and determine some ore properties such as grain zone, shape, texture and intergrowth of minerals. In this regard, the fractions of -25+150 and -177+1000 μm were used for polished and thin sections, respectively (polish sections: -177+150, -150+100, -100+75, -75+53, -53+38, -38+25, -25 μm and thin sections: +1000, -1000+600, -600+300, -300+177 μm). Thin sections of copper ore were prepared and applied for petrographic studies by Quantitative Evaluation of Minerals by Microscope (QEMSCAN) to identify the type of minerals and assess the abundance of each mineral phase in a multiple mixture.

The sections of the mineral were thoroughly polished. The reflecting power was determined in yellow light. The microscopic studies showed that the chalcopyrite is the most predominate copper mineral in ore. Fig. 1.a and Fig. 1.b showed that chalcopyrite has been interlocked by pyrite. Fig. 1.c shows that the separated particles obtain in a fraction of -53+38 μm . The color of chalcopyrite is yellow in the polished section of copper ores. Usually, pyrite has been seen in the form of yellow-veined and crack filling. The results show that there is a locked particle of chalcopyrite and pyrite and some pyrite particles are coated with chalcopyrite. Also, pyrite exists along the particle boundaries and fractures of the chalcopyrite. The locked particles of pyrite and chalcopyrite are unlocked with gangue such as quartz and feldspar. The fine chalcopyrite was smaller than 10 μm and was interlock with gangue (notably sericite). The chalcopyrite particles were very fine and can be observed from the texture that chalcopyrite particles segregated from pyrites in this section. It seems that if the sample was ground to less than 53 μm , chalcopyrite can be liberated from these impurities; however, this has to be confirmed by further liberation studies. Agheli et al. (2017) also indicated that there are a significant amount of locked chalcopyrite-pyrite particles in concentrate streams of Sarcheshmeh copper flotation plant for various size fractions (Agheli et al., 2017).

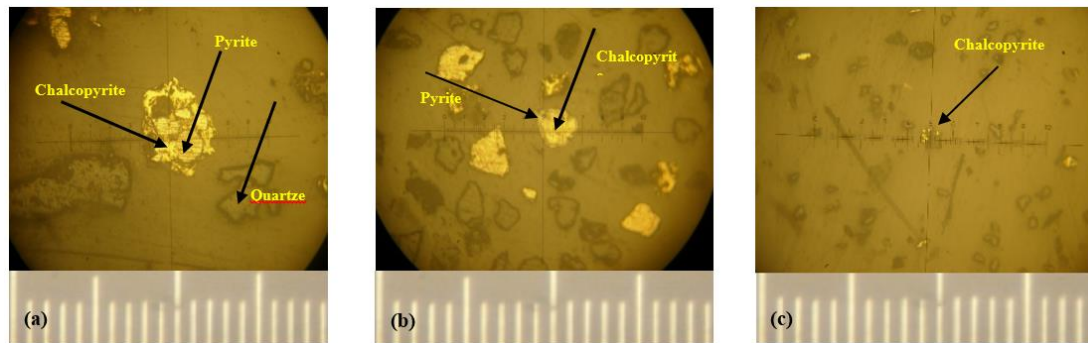


Fig. 1. The samples of polish sections (a): $-150+100\ \mu\text{m}$, (b): $-100+75\ \mu\text{m}$, (c): $-53+38\ \mu\text{m}$ (The color of chalcopyrite is yellow and the color of pyrite is yellow-veined and crack filling)

3.2 Grinding studies

In order to determine the efficiency of comminution equipment and the ore optimum liberation size for flotation tests, the sieve analysis was carried out. The d_{80} of feed was $103.71\ \mu\text{m}$. Thus, the ore grinding for reduction of the particle size is necessary. The flotation process is highly sensitive to fine particle size. Therefore, it is important to achieve an appropriate particle size in the flotation. In order to determine the optimum grinding size, a ball mill was used for grinding of crushed materials at various times including 30, 40, 50 and 60 minutes. The results are shown in Fig. 2. The particle size decreased with increasing the grinding time. The d_{80} of the sample decreased from 103.71 to $35.42\ \mu\text{m}$ during 60 minutes of grinding. There was a reverse polynomial relationship between d_{80} and grinding time ($R^2 = 0.96$). According to the mineralogical study and grinding study results, the best grinding time to achieve the optimum liberation degree of $53\ \mu\text{m}$ was determined in 43 minutes (Fig. 2).

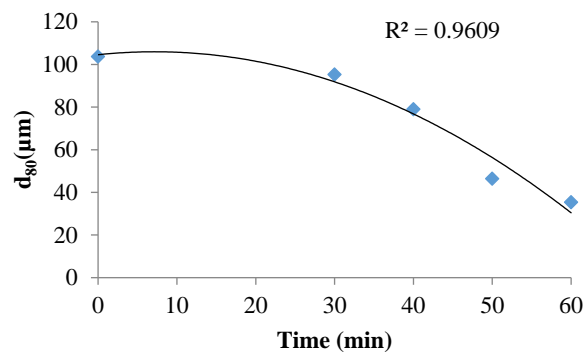


Fig. 2. The effect of grinding time on d_{80}

3.3 Flotation experiments

3.3.1 pH effect

In order to investigate the effect of the activated pyrite in the chalcopyrite flotation, the experiments were performed in the alkalinity conditions. To evaluate the effect of pH on the selective separation of chalcopyrite from pyrite, flotation experiments were performed at various pH values including 7.3, 10, 10.5, 11, 11.5 and 12 and the constant conditions of $25\ \text{mg}/\text{dm}^3$ X231 and $25\ \text{mg}/\text{dm}^3$ DF250. The results are shown in Fig. 3.

Fig. 3 shows a similar pattern between a pH of 7.3 to 11 for recovery of chalcopyrite and pyrite. In this range of pH, the recovery of chalcopyrite and pyrite slightly changed which has a low increase at a pH of 11 to 11.5. However, the recovery of pyrite was less than chalcopyrite in all ranges of pH. Iron hydroxides on the pyrite surface are soluble in the acidic condition. With increasing the pH, the solubility of iron hydroxides is reduced. Thus, pyrite is more floatable in the acidic conditions than the alkalinity (Davenport et al., 2002). According to the results, the pH of 11.5 was used for all of the experiments in this study. In a pH of 12, the recovery of chalcopyrite and pyrite was reduced from

80.7 and 78.23% to 76 and 68%, respectively. The floatability of chalcopyrite and pyrite decreased due to the formation of a hydrogen bond. As seen in the figure, increasing the pH did not significantly reduce the flotation recovery of pyrite. Pyrite in this condition acted similar to chalcopyrite that led to an increase of iron in the concentrate. Pyrite coated by chalcopyrite has the same results with chalcopyrite in the flotation process. Likewise, pyrite can be activated by Cu-ions that leads to increase its recovery in the alkalinity condition (Ejtemaei and Nguyen, 2017, Guo and Peng, 2017). The mild steel medium that used in the grinding stage can be also caused to form a copper(I) sulphide phase by the reduction of copper(II) to copper(I) on the pyrite surface, which resulted in the pyrite activation (Peng et al., 2003). It alters the surface property of pyrites that acts like chalcopyrite during flotation. Also, the interactions of activated pyrite and pyrite surfaces with collector molecules can increase the floatability of pyrite.

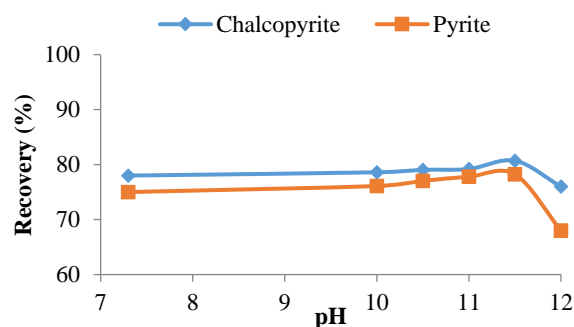


Fig. 3. The effect of pH on the recovery of chalcopyrite and pyrite

3.3.2 Effect of collector concentration

In order to investigate the effect of collector concentration on the chalcopyrite and pyrite recovery, the experiments were carried out with 25 mg/dm³ DF250, pH of 11.5 and Nascol as a collector. The results are presented in Table 4.

Table 4. The effect of collector concentration on the recovery of pyrite and chalcopyrite

Collector concentration (mg/dm ³)	Chalcopyrite grade (%)	Chalcopyrite recovery (%)	Pyrite grade (%)	Pyrite recovery (%)
25	1.93	67.12	8.70	56.66
12	1.94	44.08	6.94	31.32

Flotation recovery and grade of chalcopyrite and pyrite at various collector concentrations are shown in Fig. 4 and Fig. 5, respectively. Recovery of both minerals increased with increasing the collector concentration. The recoveries of chalcopyrite and pyrite were obtained as 67.12% and 56.66% in 25 mg/dm³ of collector concentration, respectively. The results showed that the chalcopyrite recovery was more than that of pyrite. The pyrite recovery increased with increasing collector concentration more than the chalcopyrite recovery. Activated pyrite has the same performance with chalcopyrite, therefore, it floats with increasing the collector concentration. Activated pyrite used the collector in the pulp that led to reduction of chalcopyrite recovery due to the absence of enough collector ions for the recovery of chalcopyrite in the pulp. The grade of pyrite increased from 6.94% to 8.7% in 25 mg/dm³ of collector concentration. In contrast, the grade of chalcopyrite nearly remained steady with increasing the collector concentration. The results show that the activated pyrite led to a decrease in the chalcopyrite flotation efficiency and increased the recovery and grade of the pyrite.

3.3.3 Effect of depressant

Depressant was used for decreasing the undesired mineral recovery in flotation. Hassanzadeh and Hasanzadeh (2017) applied some depressants including sodium sulphide and sodium metabisulfite with the purpose of depression pyrite from chalcopyrite in a high pyritic copper sulphide ore. They

reported that the selection of an effective depressant is an effective factor on the chalcopyrite recovery and the use of depressant of sodium metabisulfite provides the highest chalcopyrite recovery in a high pyritic copper sulphide ore (Hassanzadeh and Hasanzadeh, 2017). In order to evaluate the effect of adding depressant, the experiments were carried out with and without Na_2SO_3 as a depressant, 25 mg/dm^3 Nascol and 25 mg/dm^3 DF250. The condition time for depressant was considered as 4 minutes. Fig. 6 shows that the recovery of pyrite and chalcopyrite increased in the presence of Na_2SO_3 . The increase in chalcopyrite recovery was more than pyrite recovery. It increased from 69.20 to 78.20 by using the depressant. It is shown that Na_2SO_3 was effective for pyrite depression. Na_2SO_3 reacted with ferric ions/hydroxides of pyrite that led to depressing pyrite, but it had little or no depression ability for the activated pyrite. Thus, the activated pyrite recovery increased by using Na_2SO_3 . Fig. 7 shows that the grade of chalcopyrite was almost steady from 2.11 to 2.36% using the depressant, while the grade of pyrite increased from 10.5 to 11.71%. This indicated that the activated pyrite floated to the concentrate.

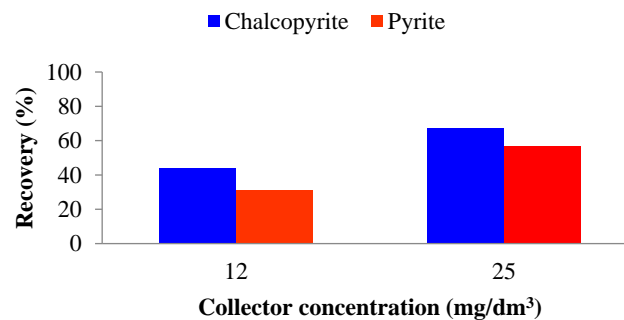


Fig. 4. Chalcopyrite and pyrite recoveries at various collector concentrations

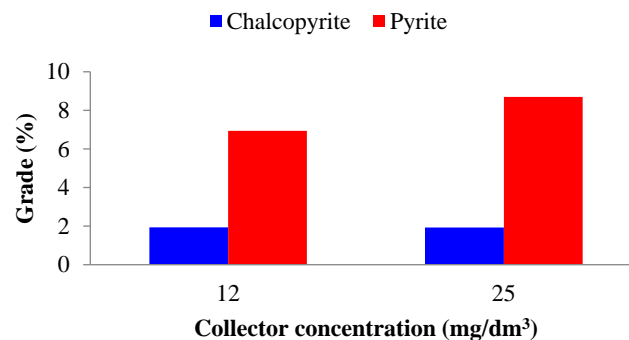


Fig. 5. Chalcopyrite and pyrite grades at various collector concentrations

3.3.4 Effect of frother

The size and stability of bubbles are effective factors in the flotation process. Frothers controlled the bubble size by reducing the surface tension of the liquid phase and stabilized the air/water interface. Adding frother in the pulp provides smaller bubble size and bigger surface area which leads to an increase of flotation efficiency (Hoseinian et al., 2017). If the bubbles do not have enough stability before leaving the cell, the bubbles, which are loaded by minerals, may be collapsed and load minerals will be dropped back to the pulp (Shakir et al., 2010).

Fig. 8 and Fig 9 show that the recovery and grade of chalcopyrite and pyrite increased by adding the frother. The recovery of chalcopyrite and pyrite increased from 29.95 and 23.37% to 67.12 and 56.66% in the presence of frother, respectively. The activated pyrite had a similar performance with chalcopyrite. The recovery and grade of both chalcopyrite and pyrite increased with adding DF250.

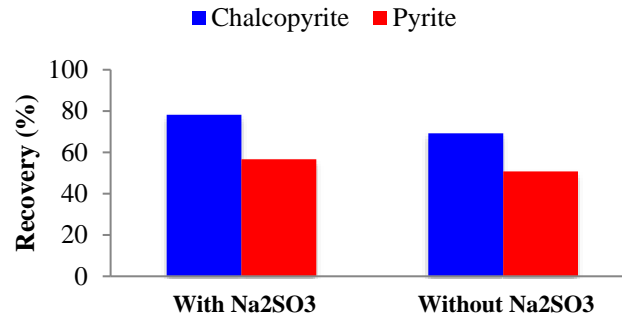


Fig. 6. The effect of Na₂SO₃ on the recovery of chalcopyrite and pyrite

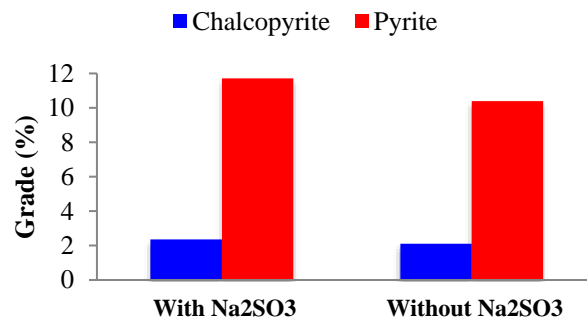


Fig. 7. The effect of Na₂SO₃ on the grade of chalcopyrite and pyrite

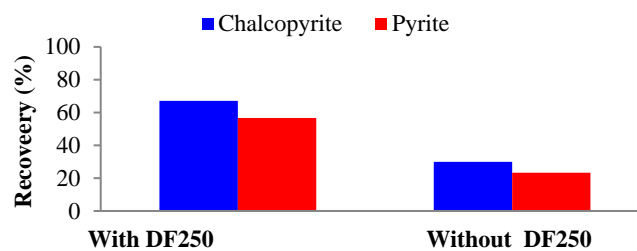


Fig. 8. Chalcopyrite and pyrite recovery at various frother concentrations

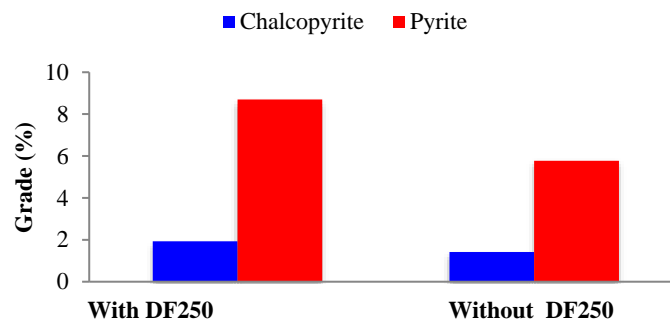


Fig 9. Chalcopyrite and pyrite grade at various frother concentrations

3.3.5 Effect of collector type

One of the effective factors in the flotation process is the nature of the collector and interaction between collector and minerals. In porphyry copper flotation, xanthates are usually used as a primary collector and dithiophosphates as a secondary collector. The flotation experiments were conducted with various collectors such as Nascol, Aero 407, Aero 3477 and X231 in the same conditions. The

anionic collector with oxygen atoms in its charge group can form dipolar bonds and thus the collector absorption during the flotation may be more than the Coulomb absorption. The frother with hydrogen atoms in its structure can form hydrogen bonds with oxygen atoms in the collector charge group. The chemical nature of the specific combination of the frothers and collectors may have appropriate interactions and a positive effect on the recovery of the minerals. Oxygen atoms in the collector structures of Nascol, Aero 407, Aero 3477 and X231 can form hydrogen bonds with the hydrogen atoms in the DF250 frother structure that provide an appropriate stable condition (Hoseinian et al., 2015, Hoseinian et al., 2018).

Table 5. Structures of collectors

Reagent	Formula	Chemical name	Molecular weight(g/gmol)	Supplier
Nascol	$C_8H_{18}NaO_2PS_2 + C_7H_5NS_2$	A mixture of mercaptobenzothiazole and sodium diisobutyl dithiophosphate	431.570851	Nasco
Aero 407	$C_7H_4NNaS_2 + C_8H_{18}NaO_2PS_2$	A mixture of sodium mercaptobenzothiazole and sodium diisobutyl dithiophosphate)	453.550851	Sitec
Aero 3477	$C_8H_{18}NaO_2PS_2$	Sodium-diisobutyldithiophosphate	264.320851	Sitec
X231	$C_6H_{13}NOS$	Ethyl isopropyl thionocarbamate	147.23852	Hoechst

Table 5 shows the collector structures of Nascol, Aero 407, Aero 3477 and X231. All of the four collectors have oxygen atoms in their structures. Fig. 10 and Fig. 11 show the recovery and grade of chalcopyrite and pyrite with different collectors.

Selective reagents can be used for the selective flotation of sulphide minerals. Better selectivity flotation of chalcopyrite against pyrite was achieved by using Aero 407 as a collector. Aero 407 is a mixture of sodium mercaptobenzothiazole and sodium diisobutyl dithiophosphate that acts stronger than the other collectors for decreasing the flotation of activated pyrite. The results show that the recovery and grade of chalcopyrite and pyrite using the Aero 407 were 79.95%, 49% and 5.3%, 7.98%, respectively. In these conditions, the recovery of activated pyrite decreased and Aero 407 selectively floated chalcopyrite. The grade of chalcopyrite using Aero 407 was more than the other collectors, while the grade of activated pyrite with Nascol was more (Fig. 11). The collectors of Aero 3477, X231 and Nascol had less chalcopyrite recovery and also less selective properties to float the chalcopyrite, respectively. Aero 407 collector had a better performance in chalcopyrite selective flotation in the presence of activated pyrite. This means that the highest copper recovery could be achieved by Aero 407 which is the mix of Xanthate and dithiophosphates.

3.3.6 Effect of cleaner stage

Flotation experiments for evaluation of the effect of cleaner stages were carried out using 2 kg of sample in order to increase the grade of concentrate in the optimized conditions (pH of 11.5, DF250

frother= 25 mg/dm³, Aero 407 collector= 25 mg/dm³). Three steps of cleaner were performed at different times around 3, 2 and 1 minutes, respectively. Fig. 12 shows that the chalcopyrite grade increased from 0.94 to 21.13%, with three stages of cleaner. Likewise, the pyrite grade increased with increasing the chalcopyrite grade. This shows that the activated pyrite floated like chalcopyrite in the cleaner stages.

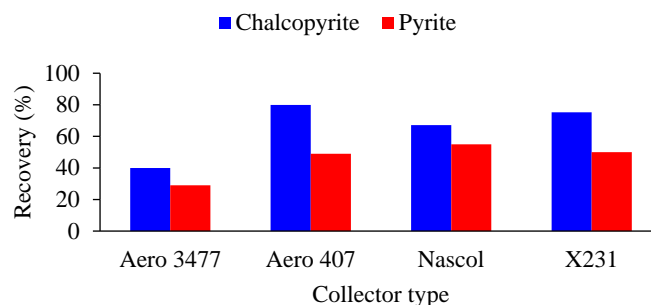


Fig. 10. Flotation recovery of chalcopyrite using various collectors at a pH of 11.5

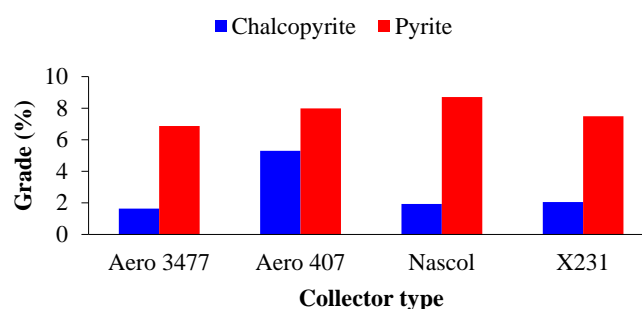


Fig. 11. Flotation grade of chalcopyrite using various collectors at a pH of 11.5

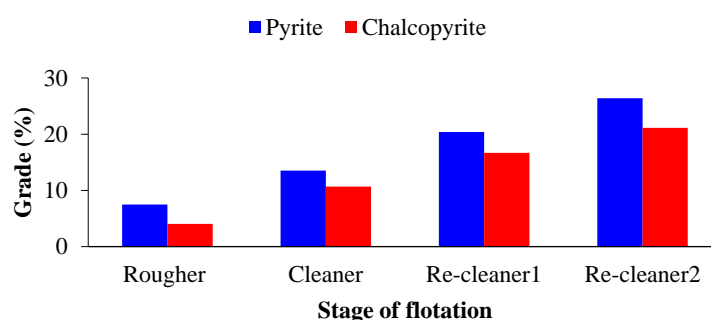


Fig. 12. The grade of chalcopyrite and pyrite in cleaner stages

4. Conclusions

Active pyrites act like chalcopyrite during flotation, which leads to an increase in the iron grade of the concentration. In this study, the effect of reagents and main effective parameters were investigated to achieve high separation between chalcopyrite and activated pyrite. According to the microscopic studies, the optimum liberation degree of 53 μm is necessary for flotation. The grinding studies showed that the best grinding time was 43 minutes to achieve the optimum liberation degree. The effect of flotation reagents was evaluated on the chalcopyrite and activated pyrite flotation. The results showed that in all conditions, activated pyrites act like chalcopyrite. The collector type had a greater effect on the selectivity flotation of chalcopyrite than the other parameters. The optimum condition for

chalcopyrite flotation was 25 mg/dm³ DF250 as frother, 25 mg/dm³ Aero 407 as a collector, pH of 11.5 and 25 mg/dm³ Na₂SO₃ as a depressant. In the optimal condition, the recovery and grade of chalcopyrite and pyrite are 79.95%, 49% and 5.3%, 7.98%, respectively. The grade of chalcopyrite is increased by using the cleaner stages.

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