

Selective separation of galena and sphalerite from pyrite-rich lead-zinc ores: A case study of the Kooshk mine, Central Iran

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ABSTRACT

In this research, the selective flotation of galena and sphalerite minerals was evaluated from a pyrite-rich lead-zinc sample. An optimum mixture of selective collectors against pyrite and also organic/inorganic pyrite depressants was obtained in the flotation of galena and sphalerite by an experimental mixture design. The effect of different collectors and pH values were investigated in the obtained optimum flotation condition. The results indicated that the mixture of Aero 404 Promoter collector with Potassium Ethyl xanthate (PEX) has the highest Pb recovery and separation efficiency. The mixture of auxiliary collectors of Aero 3477, Aerophine 3418 and Florrea 2255 with Potassium Amyl Xanthate (PAX) improved Zn recovery in comparison with the test in which PAX was added alone in the Zn rougher flotation stage. The highest Zn separation efficiency was achieved by Aerophine 3418 when it was mixed with PAX for which the lowest pyrite particles were floated. Zn separation efficiency was markedly improved from 47.47% to 65.31% from pH 9 to 12. It can be concluded that increasing pH with lime, up to 12, successfully depressed pyrite without any significant negative effect on Zn recovery.

Keywords : Flotation, Pyrite, Mixture design, Auxiliary Collector, Kooshk

1. Introduction

Lead and zinc ores are the most abundant metallic ores on Earth's crust and they are found all over the world, such as North America, South America, Balkan Peninsula, Europe, Russia, Australia, Africa and Asia. The sedimentary lead-zinc deposits are important sources of lead and zinc and amount to about 50% of the world lead-zinc production. Some sedimentary lead-zinc deposits are located in black shale, as their host are gangue minerals. Examples of these deposits would be Red-Dog (USA), Cirque (Canada), Mount Isa (Australia), Elura (Australia) and Kooshk (Iran). The presence of organic matters and carbonaceous pyrite in these deposits influences the selectivity between galena, sphalerite and pyrite species in the flotation process [1-3].

The Kooshk mine is one of the largest zinc and lead deposits in Iran. The ore contains of sphalerite and galena as the valuable minerals, and pyrite, dolomite, calcite, are the main gangue minerals. In addition, carbonates are present among the gangue minerals [1]. The presence of high amounts of pyrite and carbonaceous matters are detrimental to the concentration of lead and zinc flotation in the Kooshk plant. Therefore, pre-flotation, which is a flotation stage prior to the galena flotation, is employed to remove part of the organic carbon. Following the pre-flotation stage, lead and zinc sulfides are floated in separate circuits and during the flotation process pyrite is floated to the concentrates. Therefore, the presence of carbonaceous matters and pyrite are two main issues in the Kooshk mineral processing plant, affecting the flotation selectivity.

Pyrite is readily floated due to its natural hydrophobicity or the collector-induced hydrophobicity. Pyrite floatability can also be enhanced by the activation of copper or lead ions either dissolved from complex sulfide minerals due to galvanic interactions or are present as

contaminants in the processing water [3-8]. Increasing the pulp pH using lime is one of the most commonly method to depress pyrite that will inhibit the copper activation and subsequent collector adsorption. In addition, to remove pyrite during the flotation of galena and sphalerite, sodium cyanide is commonly used as a selective pyrite depressant [3, 9-11]. However, sodium cyanide consumption has been forbidden in the Kooshk plants due to environmental concerns. Consequently, selective separation of lead-zinc sulfide minerals from pyrite requires appropriate selective depressant and collectors.

Different reagents are usually applied in the selective separation and depression of iron sulfide minerals; they include inorganic depressants such as cyanide, lime, oxygen conditioning, sulfite, etc., as well as organic reagents such starch, dextrin, chitosan, carboxymethyl cellulose, polyacrylamides, wood extracted biopolymers, a combination of these reagents [4, 13-18]. Inorganic depressants, though effective, are found to be detrimental to the environment and cause additional cost during the handling processes. Organic depressants, with rich sources, a biodegradable nature and a relatively inexpensive price are attracting great attention and possess the potential to function as alternative depressants [8, 19-22]. It is suggested to combine the use of polysaccharides with other metal selective chelating agents, organic or inorganic, to improve the selectivity of polymers towards the target mineral by selectively removing the contaminating metal ions [14]. Besides using pyrite depressants, applying some specific collectors would improve the selective separation of lead-zinc minerals from pyrite. Despite being a strong collector, xanthate cannot provide the desired selectivity in a complex system involving pyrite. In this respect, other collectors such as dithiophosphate (DTP), dithiophosphinate (DTPI) or thionocarbamate (TC) type collectors are frequently used in the flotation of sulfide minerals, alone or in combination with xanthates. They satisfy the selectivity against iron sulfides and have a strong collectivity power similar to that of xanthate [3, 15]. Dithiophosphinate

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and thionocarbamate are mainly recommended to float lead, zinc, copper and other precious metals from pyrite-rich minerals. The increased selectivity against pyrite is due to the more pronounced affinity of dithiophosphinate and thionocarbamate towards metallic species which constitute the mineral matrix than towards iron species [23-24].

In this research, the selective flotation of galena and sphalerite was evaluated in a pyrite-rich sample of the Kooshk mine. Organic and inorganic pyrite depressants and also specific collectors were used in the appropriate combination using the mixture design method. After achieving an optimum flotation test condition, the effect of auxiliary collector types and pH were evaluated on the flotation of galena, pyrite and sphalerite.

2. Materials and Methods

2.1. Characteristics of pyrite-rich lead-zinc sample

A representative pyrite-rich lead-zinc sample was obtained from the Kooshk mineral processing plant in Yazd, Central Iran. The sample was collected from a cyclone overflow (froth flotation feed) by a systematic sampling program. Sampling increments were collected in equal time intervals of 1 h. The gross sample was obtained by mixing the increments collected in equal time intervals during the working shifts, more than 200 kg. Particle size distribution of the initial sample specified that D_{80} of the representative sample was 104 μm . The chemical analysis (Atomic adsorption spectroscopy) of this sample contained 7.64% Zn, 1.47% Pb and 23.08% Fe. For a complete ore characterization, an optical mineralogy was carried out ((U-LH100, Japan)), using polished and thin sections, as well as semi-quantitative x-ray diffraction techniques (SQXRD). XRD and mineralogical studies revealed that pyrite, sphalerite, and galena were the major sulfide phases and calcite, dolomite, were the major gangue phases.

2.2. Experimental procedures

2.2.1. Flotation experiments

Flotation experiments were carried out in a 2.5 L Denver laboratory flotation cell using a 1000 g ore sample. Pre-flotation was employed to remove some of the carbonaceous matter through adding diesel oil (120 g/t) and methyl isobutyl carbonyl (MIBC 50 g/t) as collector and frother, respectively. Potassium Ethyl Xanthate (PEX) and Na- Di-alkyl Dithiophosphate Florrea 2701 were used as the galena flotation collectors in the Pb rougher stage. Potassium Amyl Xanthate (PAX) and Dialkyl dithiophosphinates Aerophine 3418 were used as the sphalerite flotation collectors in the Zn rougher stage. Dextrin, FeSO_4 , KMNO_4 were applied as the pyrite depressants and ZnSO_4 as the sphalerite depressant. During the galena and sphalerite flotation, pH was adjusted with a lime milk to 8.5 and 10.5, respectively. A copper sulfate (400 g/t) was totally added to the sphalerite conditioning stage for sphalerite activation. These reagents were industrial grades prepared from the industrial lead and zinc flotation plants. Fig. 1 describes the baseline of the flotation experiments.

Forty experiments were designed based on the combined design containing two mixture types in the Pb flotation and one mixture in the Zn flotation stage. The selected control factors and their levels, including blend 1: pyrite and sphalerite depressants; blend 2: Pb flotation collectors (PEX+Florrea 2701) and blend 3: Zn flotation collectors (PAX+ Aerophine 3814) are presented in Table 1. The proposed conditions of mixture experiments are presented in Table 2.

A mixture design is a special type of experimental design in which the factors are the ingredients or components of a mixture, and the response is a function of the proportions of each ingredient. In many mixture experiments there are constraints on the component proportions. The general form of the constrained mixture problem is

$$x_1 + x_2 + \dots + x_q = 1$$

$$L_i \leq x_i \leq U_i, \quad i=1, 2, \dots, q$$

Where L_i is the lower bound for the i th component and U_i is the upper

bound for the i th component.

Where $L_i \geq 0$ and $U_i \leq 1$ for $i=1, 2, \dots, q$ [25-26]

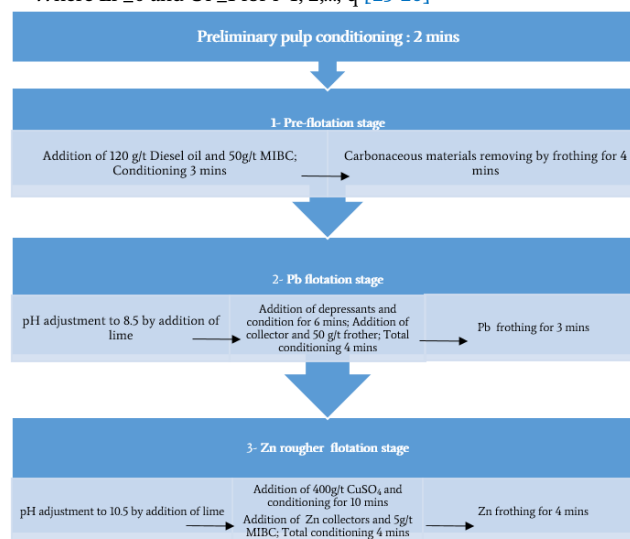


Fig. 1. Baseline for experimental procedure.

Table 1. Mixed parameters and their levels.

Blend	Code	Reagent	Lower bound (L_i)	Upper bound (U_i)
1	A	FeSO_4	300	700
	B	ZnSO_4	0	400
	C	KMNO_4	0	400
	D	Dextrin	0	400
2	E	PEX	100	120
	F	Florrea2701	0	20
3	G	PAX	80	100
	H	Aerophine3418	0	20

Different auxiliary collectors were added in the optimum flotation condition of Pb and Zn. These collectors included Mercaptobenzothiazole and Dithiophosphate (Aero 404 promoter), Na-Di isobutyl Dithiophosphate (Aero 3477 promoter), Na- Di isopropyl Dithiophosphate (Aero Float 211 promoter) and Mono and Dialkyl Dithiocarbamates (Aero 4037 promoter).

3. Results and discussion

3.1. Pb Flotation

Flotation experiments were carried out based on Table 2. Following the pre-flotation stage, the Pb flotation responses were measured and calculated. The obtained data were presented in Tables 3 and 4. The results were compiled into the "Design Expert (DX)" software and appropriate models among several models were chosen and fitted to the Pb-Zn flotation responses. The models were fitted to Pb, Zn and Fe recovery as well as the separation efficiency in the Pb-Zn concentrate. The separation efficiency is the metallurgical efficiency of the separation. In other word, it deals with quality of separation between the gangue and valuable minerals. It can be calculated by [27]:

$$S.E = 100 \frac{C}{F} \left(\frac{c}{f} - \frac{m-c}{m-f} \right)$$

Where c , f and m denote the metal content in concentrate, feed and valuable mineral structure, respectively. C and F refer to the weight of concentrate and feed, respectively.

Based on the analysis of variations and statistical methods, the appropriate models were found to be adequate for the prediction of response variables. The results of variance analysis (ANOVA) of the developed models are shown in Table 6. It illustrates that all fitted models (except Fe recovery) are significant in a 90% confidence level (p -value < 0.1).

Table 2. Experimental plan of flotation tests.

NO	Pb Flotation				Collectors(g/t)		Zn Flotation Collectors(g/t)	
	Dextrin	KMnO ₄	ZnSO ₄	FeSO ₄	PEX	Florrea 2701	PAX	Aerophine 3418
TF1	0	200	0	500	100	20	80	20
TF2	0	0	200	500	110	10	90	10
TF3	0	0	400	300	120	0	100	0
TF4	0	0	200	500	100	20	80	20
TF5	0	0	400	300	110	10	90	10
TF6	0	0	0	700	100	20	80	20
TF7	100	100	100	400	115	5	85	15
TF8	0	200	200	300	110	10	90	10
TF9	50	50	50	550	115	5	95	5
TF10	0	400	0	300	120	0	100	0
TF11	200	0	200	300	120	0	100	0
TF12	400	0	0	300	100	20	80	20
TF13	200	200	0	300	110	10	90	10
TF14	200	0	0	500	110	10	90	10
TF15	200	200	0	300	120	0	100	0
TF16	50	50	250	350	120	0	100	0
TF17	0	400	0	300	100	20	80	20
TF18	0	0	400	300	100	20	80	20
TF19	100	100	100	400	105	15	95	5
TF20	0	0	0	700	110	10	90	10
TF21	0	200	0	500	120	0	100	0
TF22	200	0	200	300	100	20	80	20
TF23	0	200	200	300	100	20	80	20
TF24	0	400	0	300	110	10	90	10
TF25	200	200	0	300	100	20	80	20
TF26	400	0	0	300	120	0	100	0
TF27	400	0	0	300	110	10	90	10
TF28	400	0	0	300	100	20	80	20
TF29	200	0	200	300	110	10	90	10
TF30	50	50	50	550	105	15	85	15
TF31	200	200	0	300	100	20	80	20
TF32	0	0	200	500	120	0	100	0
TF33	0	0	0	700	100	20	80	20
TF34	200	0	0	500	100	20	80	20
TF35	0	400	0	300	100	20	80	20
TF36	0	0	0	700	120	0	100	0
TF37	0	200	0	500	110	10	90	10
TF38	0	200	200	300	120	0	100	0
TF39	200	0	0	500	120	0	100	0
TF40	0	0	400	300	100	20	80	20

Fig. 2 shows the effect of different depressants mixture and also collectors on the recovery of the Pb flotation stage. It can be concluded that the appropriate mixture of collectors is PEX:113.5 g/t + Florrea 2701:6.5 g/t, in which a higher Pb recovery can be observed. The use of KMnO₄ and Dextrin as the pyrite depressants revealed a negative effect on galena recovery. In other words, these pyrite depressants depressed galena as well.

Fig. 3 presents sphalerite recovery against the mixture of collectors and depressants. It is observed that the combination of PAX with Aerophine 3418 improved sphalerite recovery. In higher amounts of FeSO₄, sphalerite recovery increased and no depression of sphalerite observed with FeSO₄; while, the mixture of ZnSO₄ with FeSO₄ resulted in sphalerite depression.

Fig. 4 obviously shows that the Zn separation efficiency was enhanced when a PAX collector was combined with Aerophine 3418 in higher amounts of FeSO₄.

3.2. Optimization of galena and sphalerite flotation

The objective of the mixture design was to find the desirable blends of reagents. This could be a maximum, minimum, or an area where the response is stable over a range of factors. In this research, a simultaneous optimization technique was used (by DX7 software) for the optimization of multiple responses. Table 7 shows the optimum conditions in the Pb flotation stage. The proposed optimum Pb flotation condition was FeSO₄:700 g/t, Collector mixture of PEX: 113.5 g/t + Florrea 2701:6.5 g/t. In such a condition, the highest Pb recovery and separation efficiency was achieved with the lowest sphalerite and pyrite floatability. However, Pb recovery did not reach higher than about 33%,

possibly due to presence of carbonaceous matters and tarnished surface of galena. Following the Pb flotation stage, the optimum point of Zn flotation condition was extracted from the fitted models. Optimum objectives of Zn flotation were defined the highest Zn recovery and separation efficiency with the lowest pyrite presence in the Zn concentrate, simultaneously (Table 8). In the optimum test condition, 700 g/t FeSO₄ with mixture of PAX:92.5 g/t + Aerophine 3418:7.5 g/t, sphalerite recovery and separation efficiency obtained 73% and 50.60% respectively. In this condition, only 29% of pyrite floated to the concentrate.

The test was validated in the predicted optimum conditions according to Tables 7 and 8. The results are reported in Table 9. The experimental values were found to be close to the predicted models values for Pb and Zn recoveries.

3.3. Effect of auxiliary collectors

At the optimum Pb flotation condition (FeSO₄:700 g/t, PEX:113.5 g/t, auxiliary collector Florrea 2701:6.5 g/t, obtained from the mixture design in Table 7) different auxiliary collectors were added to pulp instead of Florrea 2701. Fig. 5 presents the results of Pb, Fe, Zn recoveries in the Pb flotation stage. The obtained results indicated that Aero 404 has the significant improvement on the Pb recovery and prevent zinc loss. In addition, this collector was generally more selective than other ones. In other words, the highest separation efficiency in the Pb flotation obtained with Aero 404 in which the separation efficiency increased from 21.32% to 27.65%.

Table 3. Results of Pb flotation stage.

NO	Grade (%)			Recovery (%)			Separation Efficiency (%)
	Pb	Zn	Fe	Pb	Zn	Fe	Pb
TF1	3.64	2.54	36.29	23.98	3.36	18.21	13.79
TF2	4.85	2.59	34.32	27.29	2.84	13.05	19.18
TF3	5.87	2.28	34.73	26.22	2.01	10.81	19.63
TF4	6.02	2.45	35.85	20.11	2.75	7.78	15.51
TF5	6.45	2.34	35.80	24.64	1.83	9.43	18.96
TF6	4.75	2.26	36.07	28.36	2.71	15.12	19.29
TF7	5.20	2.92	30.34	21.45	2.51	9.10	15.04
TF8	4.72	2.84	38.63	17.73	2.20	10.54	11.89
TF9	5.81	3.22	35.46	28.42	2.98	11.80	21.57
TF10	5.70	2.84	40.07	21.47	2.04	10.37	16.11
TF11	4.40	3.25	32.52	20.37	3.05	10.74	13.17
TF12	5.06	3.15	30.11	26.79	3.34	11.47	18.81
TF13	4.69	2.73	35.82	30.98	3.52	15.83	21.25
TF14	5.00	2.54	40.63	33.33	3.74	21.33	22.29
TF15	4.20	2.72	38.22	23.09	2.96	13.75	15.08
TF16	4.80	2.58	35.39	26.12	2.89	14.03	17.93
TF17	4.32	2.90	38.83	25.50	3.86	17.36	16.23
TF18	5.17	2.72	34.04	25.85	2.87	12.19	18.09
TF19	3.39	2.49	35.10	23.71	3.65	17.75	12.44
TF20	6.55	2.84	34.94	32.94	2.78	12.23	24.91
TF21	3.60	2.46	34.39	21.70	3.08	15.21	12.25
TF22	4.91	2.65	33.10	24.46	2.68	11.73	16.84
TF23	6.21	2.57	37.64	25.14	2.18	10.82	19.12
TF24	3.81	2.63	35.61	25.30	3.58	17.32	14.85
TF25	5.51	2.63	38.30	28.21	2.60	13.57	20.79
TF26	5.12	3.02	32.94	21.48	2.85	10.70	14.46
TF27	5.46	3.30	34.86	30.10	3.98	14.46	21.25
TF28	5.00	2.81	30.25	27.58	3.22	12.26	18.93
TF29	4.00	2.63	35.43	28.42	3.57	16.46	18.11
TF30	4.37	2.56	36.12	28.58	3.54	17.22	18.17
TF31	6.12	2.47	36.97	24.98	2.13	12.15	18.33
TF32	3.00	2.37	39.90	23.35	2.76	22.85	10.48
TF33	4.42	2.30	35.87	27.49	2.96	16.13	17.87
TF34	3.86	2.72	31.82	25.96	3.90	14.75	15.01
TF35	4.59	2.78	39.02	24.77	3.56	16.03	16.05
TF36	6.95	2.74	35.10	29.49	2.77	10.07	23.12
TF37	3.50	2.26	38.47	19.71	2.81	16.26	10.58
TF38	4.11	2.67	33.80	21.62	2.94	13.46	13.19
TF39	4.28	2.59	36.17	20.06	2.80	12.66	12.59
TF40	5.00	2.68	34.73	24.24	2.72	12.03	16.78

Table 4. Results of Zn flotation stage.

NO	Grade (%)			Recovery (%)			Separation Efficiency (%)
	Pb	Zn	Fe	Pb	Zn	Fe	Zn
TF1	1.23	19.50	26.45	22.12	70.49	36.22	47.56
TF2	1.03	20.27	29.06	17.74	68.02	33.81	47.65
TF3	1.04	22.72	27.86	16.39	70.65	30.59	52.29
TF4	1.25	21.01	25.86	23.35	73.18	31.40	52.06
TF5	1.40	21.29	26.68	22.60	70.49	29.71	50.86
TF6	1.56	17.58	25.94	33.18	75.25	38.75	47.20
TF7	1.73	19.56	31.20	29.20	68.70	38.32	46.77
TF8	0.87	19.58	28.77	14.82	68.82	35.61	46.78
TF9	0.88	21.69	25.80	15.17	70.63	30.24	51.15
TF10	1.18	17.92	30.30	21.55	62.40	38.02	39.74
TF11	1.85	20.14	27.16	31.97	70.56	33.47	48.50
TF12	1.08	19.46	27.90	19.92	72.01	36.93	49.01
TF13	1.01	20.02	29.31	17.72	68.61	34.40	47.28
TF14	0.99	19.21	28.21	16.20	69.48	36.35	46.71
TF15	1.13	17.70	29.60	23.63	66.62	36.79	42.92
TF16	1.04	20.89	27.57	16.91	69.80	32.64	50.05
TF17	1.53	16.70	28.68	28.64	70.47	40.67	44.93
TF18	1.48	21.21	27.74	23.94	72.41	32.14	52.21
TF19	1.31	17.29	31.45	25.83	71.36	44.85	44.10
TF20	1.55	22.24	24.75	26.03	74.71	29.90	54.71
TF21	1.10	16.65	28.56	22.25	69.90	42.38	42.30
TF22	1.27	18.77	25.24	24.88	74.51	35.18	49.07
TF23	1.72	18.78	28.57	34.54	72.87	37.63	49.21
TF24	1.62	17.76	29.71	30.00	67.33	40.31	42.39
TF25	1.25	17.61	28.93	23.73	71.22	41.87	44.64
TF26	1.47	16.99	31.55	28.93	75.31	48.06	46.32
TF27	1.45	20.03	28.60	23.18	70.12	34.40	48.84
TF28	1.45	20.95	23.67	24.70	74.13	29.63	52.44
TF29	1.01	18.69	28.60	20.42	72.29	37.81	47.61
TF30	1.13	20.03	28.92	19.09	71.52	35.62	49.47
TF31	1.32	18.70	28.25	23.71	71.06	40.86	45.74
TF32	1.32	17.75	29.40	24.25	73.31	39.76	47.76
TF33	1.65	17.84	26.11	33.52	75.10	38.37	48.14
TF34	0.91	21.98	26.77	14.14	72.83	28.66	53.07
TF35	1.57	17.09	29.42	26.95	69.67	38.44	45.75
TF36	1.34	23.03	30.51	17.12	69.77	26.25	53.40
TF37	1.32	16.20	29.76	26.73	72.47	45.22	43.78
TF38	1.15	16.56	28.60	21.34	69.64	43.47	41.38
TF39	1.40	18.19	28.30	23.48	70.44	35.43	47.94
TF40	1.39	20.78	27.10	23.07	72.07	32.14	51.35

Table 5. Proposed models.

Stage	Responses	The proposed coded models
Pb Flotation stage	Pb recovery	$= +26.34^*A^*E + 27.12^*A^*F + 24.85^*B^*E + 24.7^*B^*F + 21.01^*C^*E + 25.51^*C^*F + 19.11^*D^*E + 26.85^*D^*F - 21.42^*A^*B^*F - 30.36^*A^*E^*F + 14.72^*C^*D^*E + 33^*D^*E^*F - 142.42^*A^*C^*E^*F - 108.06^*B^*C^*E^*F$
	Pb S.E	$= +21.64^*A + 17.12^*B + 15.89^*C + 18.45^*D - 14.31^*A^*B - 23.89^*A^*C - 11.3^*A^*D$
	Zn recovery	$= +2.9^*A^*E + 2.89^*A^*F + 2.19^*B^*E + 2.67^*B^*F + 2.69^*C^*E + 3.78^*C^*F + 3.24^*D^*E + 3.4^*D^*F + 4.08^*A^*D^*F - 3.94^*B^*C^*F + 2.35^*B^*D^*E - 3.91^*C^*D^*F$
	Fe recovery	$= +13.14^*E + 13.43^*F + 5.27^*E^*F - 36.74^*E^*F^*(E-F)$
Zn Flotation stage	Zn recovery	$= +69.17^*A^*G + 75.16^*A^*H + 69.64^*B^*G + 72.22^*B^*H + 62.42^*C^*G + 70.32^*C^*H + 73.58^*D^*G + 72.9^*D^*H + 14.16^*A^*B^*G + 18.03^*A^*C^*G - 8.28^*A^*C^*H - 8.99^*A^*D^*H + 8.32^*A^*G^*H + 10.57^*B^*C^*G - 13.42^*D^*G^*H - 104.43^*A^*B^*G^*H$
	Zn S.E	$= +51.37^*A^*G + 49.29^*A^*H + 49.79^*B^*G + 51.46^*B^*H + 38.13^*C^*G + 44.83^*C^*H + 46.26^*D^*G + 49.9^*D^*H$
	Fe recovery	$= +25.51^*A^*G + 35.83^*A^*H + 28.59^*B^*G + 32.12^*B^*H + 36.61^*C^*G + 41.26^*C^*H + 43.5^*D^*G + 33.55^*D^*H + 39.28^*A^*B^*G + 50.17^*A^*C^*G + 33.65^*B^*C^*G$

Fig. 6 shows the Zn and Fe recoveries and Zn separation efficiency in the Zn rougher flotation stage. Fig. 6 depicts that the mixture of collectors Aero 3477, Aerophine 3418 and Florrea 2255 with PAX improved Zn recovery 3.42%, 4.36% and 6.85% respectively in comparison with the test in which PAX was added alone (total use of collector 100 g/t). The highest Zn separation efficiency achieved by Aerophine 3418 when it was mixed with PAX (3418=7.5 g/t + PAX=92.5 g/t) and the lowest pyrite was floated in this condition.

3.4. Effect of pH on the optimum condition of flotation

In the previous flotation tests, the optimum mixture of Pb and Zn

reagents were found in the pH 8.5-10.5, respectively. In this section, the effect of pH is evaluated in the Pb and Zn rougher flotation stage. The results are presented in Fig. 7a. It can be observed that the best pH for the Pb flotation was set at 8.5. Increasing pH from 7.5 to 8.5 improved the galena recovery by about 2.5%; however, increasing pH up to 11.5 dramatically decreased the Pb floatability. Galena depression in higher pH values can refer to production of lead hydroxide and $Pb(OH)_3^-$ on the mineral surface inhibiting appropriate collector adsorption on the surface that decreases the floatability [28].

Fig. 7b illustrates the Zn flotation responses. Based on the results, as pH increased from 9 to 12, the pyrite floatability dropped significantly

from 37% to about 7%. For the highest pyrite depression, the highest Zn separation efficiency occurred at pH=12. The Zn separation efficiency improved markedly from 47.47% to 65.31% when the flotation pH changed from 9 to 12. It can be concluded that increasing pH up to 12 using lime depressed pyrite successfully without any significant negative effect on sphalerite recovery.

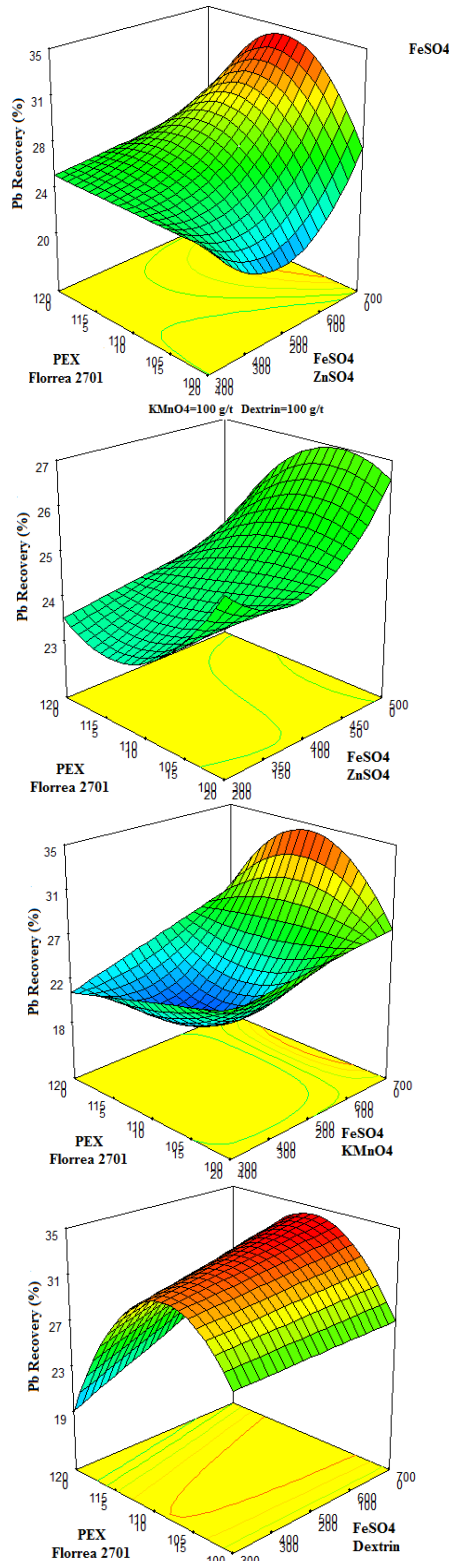


Fig. 2. Effect of collectors and Pyrite depressants on the Pb recovery in the Pb flotation stage.

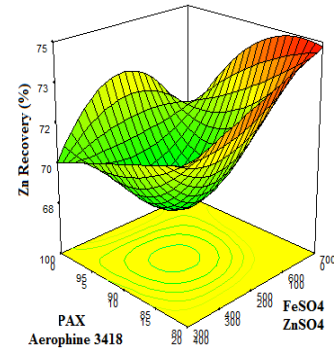


Fig. 3. Effect of collectors and depressants on the Zn recovery in the Zn flotation stage.

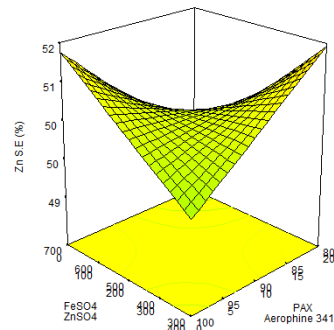


Fig. 4. Zn separation efficiency against mixtures of collectors and depressants.

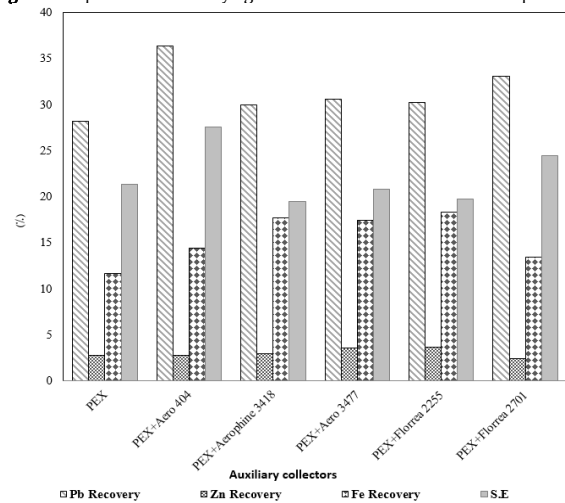


Fig. 5. Effect of auxiliary collector types on the Pb flotation responses.

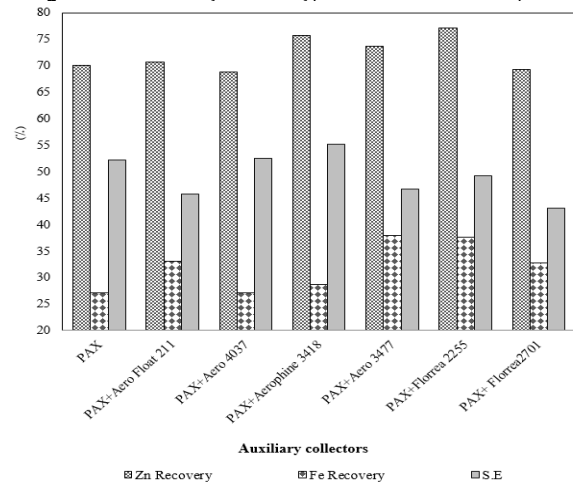


Fig. 6. Effect of auxiliary collector type on the Zn flotation responses.

Table 6. The results of ANOVA analysis of the developed models.

	Source	Sum of squares	df	Mean Square	F- value	P- value		
Pb flotation stage	Pb Recovery (%)	Model	429.81	13	33.06	10.01	<0.0001	significant
		Residual	85.9	26	3.3			
	Pb S.E.(%)	Model	168.91	6	28.15	3.01	0.0187	significant
		Residual	309.09	33	9.37			
	Zn Recovery (%)	Model	7.27	11	0.66	4.22	0.0010	significant
		Residual	4.38	28	0.16			
Fe Recovery (%)	Model	63.46	3	21.15	2.12	0.1147	not significant	
	Residual	359.06	36	9.97				
Zn flotation stage	Zn Recovery (%)	Model	227.61	16	14.23	10.65	<0.0001	significant
		Residual	30.73	23	1.34			
	Zn S.E.(%)	Model	338.98	7	48.43	10.52	<0.0001	significant
		Residual	147.29	32	4.6			
	Fe Recovery (%)	Model	584.55	10	58.45	4.53	0.0007	significant
		Residual	374.33	29	12.91			

Table 7. Optimum test condition of Pb flotation stage.

Parameter	Objective	Depressant (g/t)				Collector (g/t)			Pb (%)			Zn recovery %	Fe recovery %
		FeSO ₄	ZnSO ₄	KMnO ₄	Dextrin	PEX	Florrea 2701	Grade	Recovery	S.E			
Pb recovery	Maximum												
Pb S.E	Maximum	700	0	0	0	113.5	6.5	6.1	33	21.64	2.9	11.57	
Zn recovery	Minimum												
Fe recovery	Minimum												

Table 8. Optimum test condition of Zn flotation stage.

Parameter	Objective	Depressant (g/t)				Collector (g/t)			Zn%			Fe recovery
		FeSO ₄	ZnSO ₄	KMnO ₄	Dextrin	PAX	Aero phine 3418	Grade	Recovery	S.E		
Zn recovery	Maximum											
Zn S.E	Maximum	700	0	0	0	92.5	7.5	21.79	73	50.60	29	
Fe recovery	Minimum											

Table 9. Confirmation test at the predicted optimum test condition presented in Tables 6 and 7.

	Mass pull	Grade			Recovery			S.E	
	%	Pb	Zn	Fe	Pb	Zn	Fe	Pb	Zn
Pre- flotation	3.22	3.4	3.51	23.04	7.3	1.51	3.64		
Pb concentrate	7.18	6.53	2.75	37.25	31.27	2.63	13.12	24.51	
Zn rougher concentrate	30.85	0.76	18.22	23.02	15.63	74.95	34.83		49.64

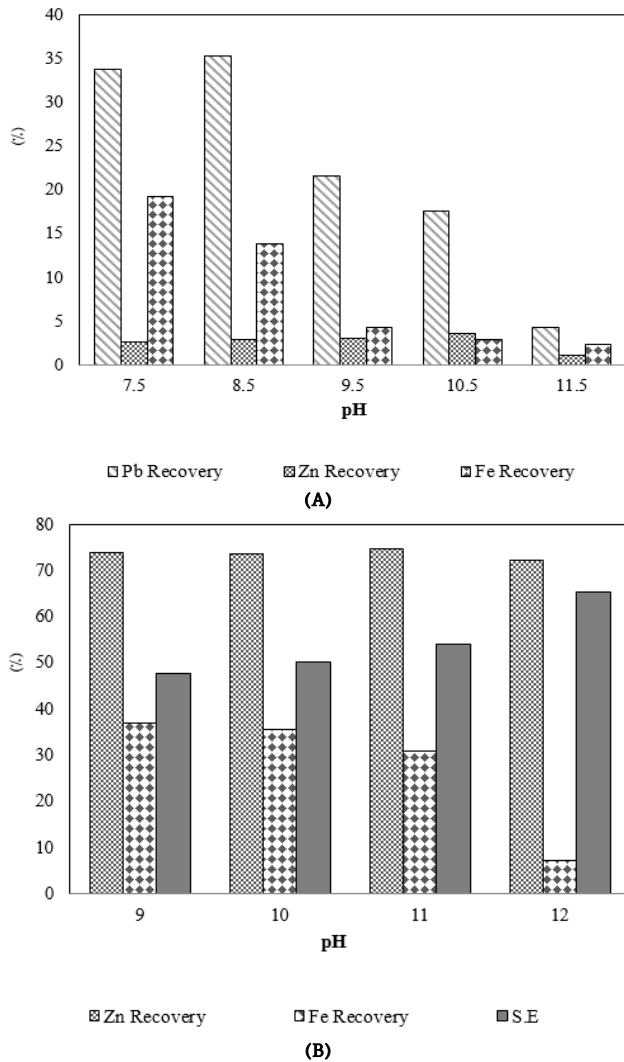


Fig. 7. Effect of pH on the A: Pb flotation responses B: Zn flotation responses.

4. Conclusion

A pyrite-rich lead -zinc sample contaminated with carbonaceous matters was prepared from the Kooshk mineral processing plant. Selective flotations of galena and sphalerite were evaluated against pyrite. Forty experiments were designed based on the combined designed containing different mixtures of depressants and collectors. Having fitted a model to the results, the optimum flotation test condition was found as follow:

- Pb flotation stage: FeSO_4 :700 g/t, PEX: 1135 g/t+ Florrea 2701: 65 g/t,
- Zn flotation stage: FeSO_4 :700 g/t, PAX: 92.5 g/t+ Aerophine 3418: 7.5 g/t,
- At the optimum conditions, the effect of different collectors was evaluated through founding a mixture of collectors. The results revealed that the collectors Aero 404 and Aerophine 3418 were the most appropriate and selective auxiliary collectors in the Pb and Zn flotation stages, respectively.
- The most appropriate pH values in the optimum flotation point were achieved to be 8.5 and 12 for Pb and Zn respectively.

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