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Effect of grinding time on flotation recovery of copper smelting slags in Bardaskan district

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Abstract

Copper smelting slags are hard materials. Therefore,to recover their copper by flotation method, grinding should be carried out to obtain optimal particle size. Copper smelting slags in the Bardeskan district, with work index of 16.24 kwh/st, were grinded for 65 minutes to reach an acceptable degree of freedom for the flotation tests, with particle size of 80%, smaller than 70 µm. With this grinding time, degree of freedom for copper-bearing minerals was achieved 85-90%. The floatation method performed and the procedure used for the optimization of the effective parameters were described in this paper. The results obtained for the flotation tests, carried out at the optimal conditions after grinding the slags (with a grinding time of65 minutes), showed 62.23% of copper recovery, while, by flotation of copper slags at optimal conditions after increasing the grinding time to 85 minutes (d_{80} = 48µ), the Cu recovery was increased to 79.89%.

Keywords: *Grinding Time, Copper Slags, Flotation, Recovery, Work Index.*

1. Introduction

Copper slags usually contain a quantity of valuable metals (except for the blast furnace slags), and they are actually a secondary resource of metals. By applying mineral processing technologies, such as crushing, grinding, magnetic separation, eddy current separation, flotation, leaching, and roasting, it is possible to recover the metals such as Fe, Cr, Cu, Al, Pb, Zn, Co, Ni, Nb, Ta, Au, Ag, from the slags. Recovery of the metals from the slags and utilization of the slags are important not onlyfor saving metal resources, but also for protecting the environment [1].

Utilization of copper slags for applications such as the Portland cement replacement in the concrete, and/or as a cement raw material has the dual benefit of eliminating the disposal costs and lowering the cost of the concrete [2].

Some of the properties of the copper slags are favorable for their use as an aggregate in the asphalt-paving applications. Dumping or disposal of these slags causes the wastage of metal values, and leads to the environmental problems. Rather than disposing, these slags can be used, taking full advantage of their physico-mechanical properties. Therefore, their use has been explored by several investigators, and the slags have been used in diversified ways like recovery of metal values, preparation of value-added products like cement, cement replacement for the concrete, fill, ballast, abrasive, aggregate, glass, tiles, etc.

Copper recovery at high temperatures takes place by the smelting process followed by converting. About the matte formation, it can be said that the major constituents of a smelting charge are sulfides and oxides of iron and copper. The charge also contains the oxides such as Al_2O_3, CaO, MgO, and principally SiO_2 which are either present in the original concentrate or added as flux. It is the iron, copper, sulfur, oxygen, and these oxides that largely control the chemical and physical constitutions of the smelting system. A further important influence is the oxidation/reduction potential of the gases used to heat and melt the charge. The first purpose of the matte smelting is to ensure sulfidization of all the copper present in the charge, so that it enters the matte phase [3].

The converter slags are recycled to the smelting furnace, where their oxide content is sulfidized to Cu_2S. Oxidized copper may be present in the form of CuO, $CuSO_4$, $CuO.CuSO_4$ or $CuO.Fe_2O_3$. These compounds also react to form Cu_2S during the smelting process. In short, the entire amount forms Cu_2S, CuS, and FeS_2 are unstable at high temperatures due to their high sulfur pressures (CuS: pS_2 =100 atm at 600 °C; FeS_2: pS_2 = 5 atm at 700 °C). These decompose during smelting to form Cu_2S and FeS [4].

About the copper slags, it should be said that, in the ore oxides and sulfides, they combine covalently to form the Cu-Fe-O-S phase in the absence of silica. During smelting, when silica is added, they combine with the oxides to form strongly-bonded silicate anions, which group together to form the slag phase. The sulfides show no tendency to form these anionic complexes, and hence, they remain as the distinct covalent matte phase, quite dissimilar to the silicate slag. Silica is added directly for the most complete isolation of the copper present in the matte that occurs at the near-saturation concentration with SiO_2 [5].

Certain amounts of lime and alumina are added to stabilize the slag structure. The molten slag is discharged from the furnace at 1000-1300 °C. When the resulting liquid slag is cooled slowly, it forms a dense, hard and crystalline product, where as quick solidification by pouring the molten slag into water, gives an amorphous granulated slag. The trade names for the copper slags include Kleen Blast and Tru-Grit. Copper slags are usually found is an impure iron silicate glass with small inclusions of copper and coppe sulfide. The chemical composition of the slags varies with the type of furnace or treatment process. The typical composition of the copper slags was given as Fe: 30-40%, SiO_2: 35-40%, Al_2O_3 < 10%, and Cu: 0.5-2.1% [6].

The results obtained for the chemical analysis of various copper slags are shown in Table 1. The copper content is limited to 0.5-2%. The major constituents are iron, silica, alumina, and calcium oxide.

Table 1. Typical chemical compositions of copper slags.

Origin	Fe, %	SiO_2, %	CaO, %	MgO, %	Al_2O_3, %	S, %	Cu, %	Co, mg/kg	Mn, mg/kg	Ni, mg/kg	Zn, mg/kg
Iranian National Copper Industries Co [7]	44.78	40.97	5.24	1.16	3.78	1.06	-	-	-	-	-
EtibankErgani Copper Plant, Elazig-Turkey [8]	39.65	31.94	3.95	2.82	2.4	-	1.01	1040	420	150	7220
Caletone Smelter Chile [9]	41.53	37.13	-	-	-	0.11	0.79	-	-	-	-
Indian Copper Plants [10]	47.8	29.9	-	-	-	-	0.7	-	-	-	-
Indian Copper Plants [10]	44.7-47.7	28.5-32	1.6-3.9	-	-	0.3-0.9	0.5-0.95	Tr-8	-	14-20	1700-2850
Indian Copper Plants [10]	47.13	-	-	-	-	1.47	0.68	2200	300	500	500
Indian Copper Plants [10]	44	28	-	-	-	-	0.6	1300	-	600	-
Kure Copper Slag [11]	47.8	26.1	0.7	1.0	6.8	1.5	0.82	4000	-	-	1500
Copper Queen, Prince, USA [2]	44.8 (oxide)	24.7	10.9	1.7	15.6	0.28	2.1	-	4000 (oxide)	-	-
Copper Queen, Prince, USA [2]	34.62 (oxide)	27.16	17.42	3.51	14.7	0.33	1.64	-	4900 (oxide)	-	-

Copper recovery from the secondary copper slag by the flotation method is an alternate route [12]. The partial enrichment of Sn and Ni is also possible by this method. A flotation scheme used at the Almalyk copper smelter for the processing slags together with Cu-Mo ores was described, and the ways to optimize the slag flotation process have been examined [13]. The results obtained for the flotation of the converter slags were presented, and the effect of the combined flotation of the slags with the ores on the copper recovery has also been evaluated. The recovery of the useful metals from the gangue material, obtained as a result of the flotation process of pyrometallurgical copper slags, was discussed [14].The gangue material generated from the flotation contained 0.30-0.65% of Cu. To recover copper, these gangues were treated with concentrated sulfuric acid. For the solubilization of the sulfates formed, the material was treated with water at 80-90 °C. Finally, a solution with 0.14 g/L of Cu was obtained, from which copper could be extracted by well-known processes.

Characterization studies were carried out on the tailings of the slag flotation mill from the Khetri copper project in Rajasthan [15] containing an average 0.2-0.4% of Cu to understand the nature, distribution, and type of locking-up of the Cu-bearing mineral. The sulfuric acid leaching process was also discussed for recovering Cu from the Khetri dump slag tailings.

White Pine Copper Div. (Copper Range Co.) has developed a variety of ways [16] to use its accumulated reverberatory furnace slags. Four ways were listed for the copper losses to occur in the slags, and justification was given for constructing a heavy media plant to recover this metal. Exploratory studies on copper recovery was conducted using the crushing, grinding, and froth flotation methods in a White Pine concentrator.

The aim of this research work was to improve the copper slag recovery through a flotation route. Almost in all of the flotation methods used to recover copper from slags, the Cu recovery was associated with many problems such as high grade of copper in tailing. In the present study, the use of grinding time increased the copper slag recovery a lot. In this research work, the flotation method and optimization of the parameters affecting the Cu recovery were also described.

2. Experimental

2.1. Identification

The copper smelting slags in the Bardeskan district (located in the NE of Iran) were investigated in this work. The floatation tests were carried out in the mineral processing laboratory of the University of Tehran, Iran. The chemical analysis results obtained for the slag sample is shown in Table 2. The mineralogical analysis results obtained showed that silicates, pyroxene and magnetite have made the main constituent, while metallic copper and chalcocite were considered as copper-bearing particles.

Table 2. Chemical analysis (XRF) of copper smelting slags inBardeskan district.

Composition	SiO_2	Al_2O_3	Fe_2O_3	CaO	Na_2O	K_2O	MgO	Cu	TiO_2	MnO_2	P_2O_5	SO_3	L.O.I
Content (%)	51.34	15.89	6.85	9.02	3.25	5.94	3.45	0.98	0.585	0.110	0.254	0.260	1.63

2.2. Crushing

Analysis of the particle size distribution in the slag sample showed that 80% of the copper slags was smaller than 40 mm. To perform the flotation tests, the copper slag samples had to be crushed to to reach appropriate size. Three crushing steps were considered including the primary jaw crusher (d_{80}=28mm), secondary jaw crusher (d_{80}=10.04mm) and roll crusher (d_{80}=1.6mm). All of these crushing steps were performed in the laboratory scale. The results obtained for the particle size distribution, obtained from the first crushing step using a primary jaw crusher, are shown in Table 3 and Figure 1.

Table 3. Sieve analysis for discharge of primary jaw crusher.

Size (mm)	**Weight remaining on screen (g)**	**Weight percentage remaining on screen**	**Cumulative weight percent passing from screen**
+50.8	0	0	0
-50.8+25.4	2870	26.11	73.89
-25.4+15.875	4625	42.07	31.82
-15.875+9.525	1793	16.31	15.51
-9.525	1705	15.51	0
Sum	10993	100	-

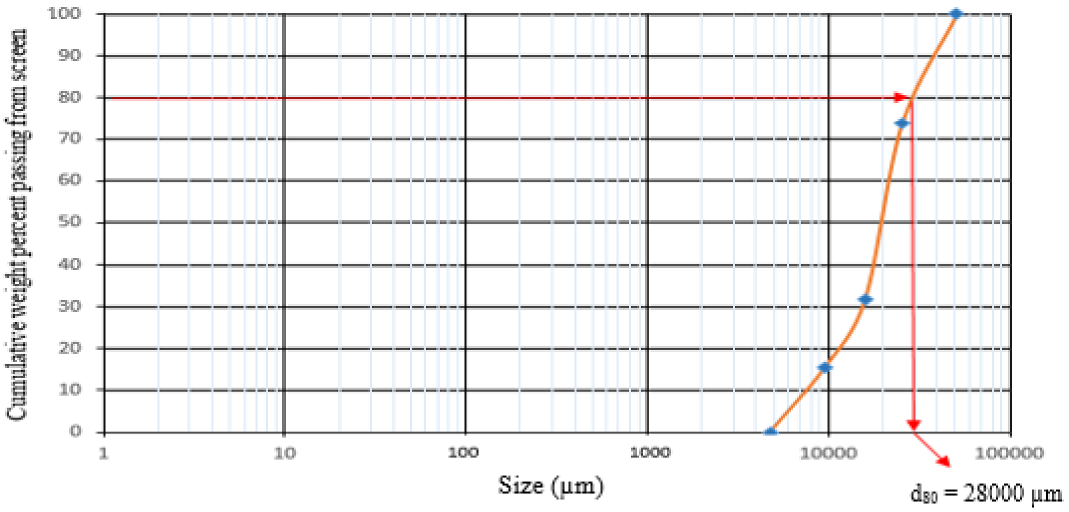


Figure 1. Particle size distribution curve for discharge of primary jaw crusher.

As it can be seen in Figure 1, 80% of the sample crushed using aprimary jaw crusher was smaller than 28 mm. The sieve analysis and particle size distribution curve obtained from the second crushing step by the secondary jaw crusher is shown in Table 4 and Figure 2.

As it can be seen in Figure 2, 80% of the sample crushedusing a secondary jaw crusher was smaller than 10043 µm. The sieve analysis and particle size distribution curve obtained from the third crushing step using the roll crusher is shown in Table 5 and Figure 3.

As it can be seen in Figure 3, 80% of the sample crushed using a roll crusher was smaller than 1600 µm.

Table 4. Sieve analysis of crushed particles by secondary jaw crusher.

Size (µm)	Weight remaining on screen (g)	Weight percentage remaining on screen	Cumulative weight percent passing from screen
+25000	0	0	100
-25000+16000	137.2	0.51	99.49
-16000+9500	5895	21.87	77.62
-9500+4800	8770	32.54	45.08
-4800+1000	7300	27.08	18
-1000	4850	18	0
Sum	26952.2	100	-

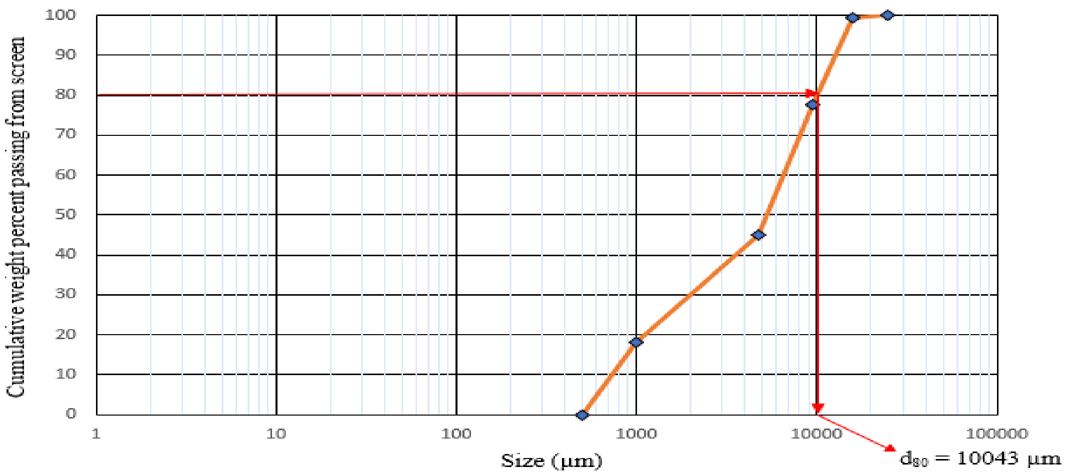


Figure 2. Particle size distribution curve for samplecrushed using asecondary jaw crusher.

Table 5. Sieve analysis of sample crushed by roll crusher.

Size (µm)	Weight remaining on screen (g)	Weight percentage remaining on screen	Cumulative weight percent passing from screen
+2000	72.8	7.95	92.05
-2000+1000	303.6	33.17	58.88
-1000+595	182.5	19.94	38.94
-595+297	161.8	17.68	21.26
-297+210	42.5	4.64	16.62
-210+150	54.4	5.94	10.68
-150+74	53.8	5.88	4.8
-74	44	4.8	0
Sum	915.4	100	-

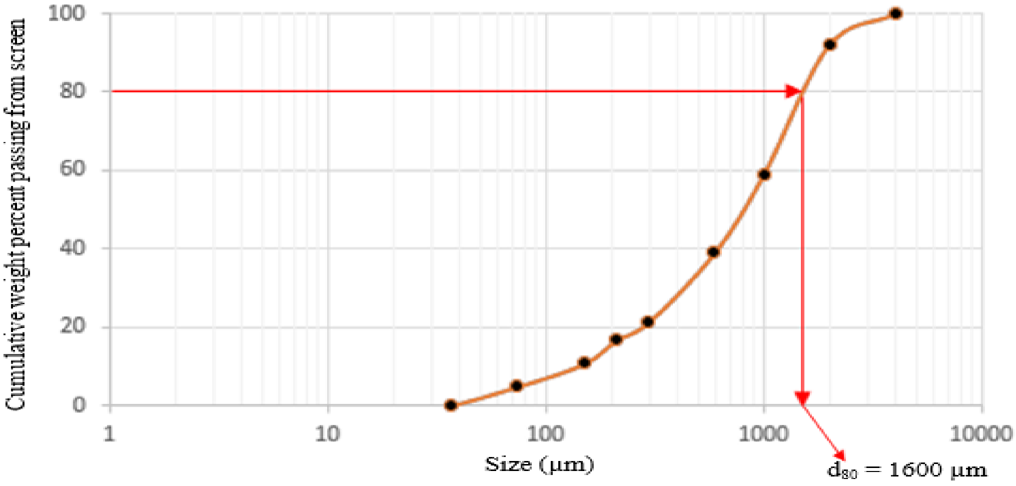


Figure 3. Particle size distribution curve for sample crushed by a roll crusher.

2.3. Specific gravity and work index

The apparent specific gravity and real specific gravity for the copper smelting slags in the Bardaskan district were achieved to be 1.35 and 2.33 g/cm^3, respectively. The ratio of real specific gravity to apparent specific gravity was obtained to be 1.73 (this ratio is usually 1.5-2). The high specific gravity for theses lags causes the slag particles to sink quickly to the bottom of the rougher cells, and on the other hand, causes problems in grinding circuits and classification. Also no gushing to waste, and obstruction of transmission path of cyclones underflow are the other disadvantages of the high specific gravity of the copper smelting slags.

Despite the progresses made in the mill modeling, the Band's work index is still widely used in designing industrial mills. Ore grindability using the Band's standard test was presented to achieve 250% of the circulating load at the laboratory scale. This index indicates the resistance of materials against grinding. Determination of the Band's work index is time-consuming, and it is usually obtained after 7-10 grinding cycles. To determine the work index of the studied copper slags, the Band's ball mill was used with a size of 20cm×20cm (diameter×length), which was calibrated in the mineral processing laboratory in the University of Tehran. After doing the tests using this ball mill, the work index for the Bardaskan copper smelting slags was obtained to be 16.24 kilowatt hours per small ton. The high work index for the copper slags indicates that grinding this material to reach the optimal particle size requires too much energy.

2.4. Determination of particle size for flotation tests

According to the results obtained for the polished sections of the roll crusher's product to determine the degree of freedom, an acceptable particle size was achieved for the copper slags to do the flotation tests. The polished sections in the range of 10-200 meshes are shown in Figure 4.

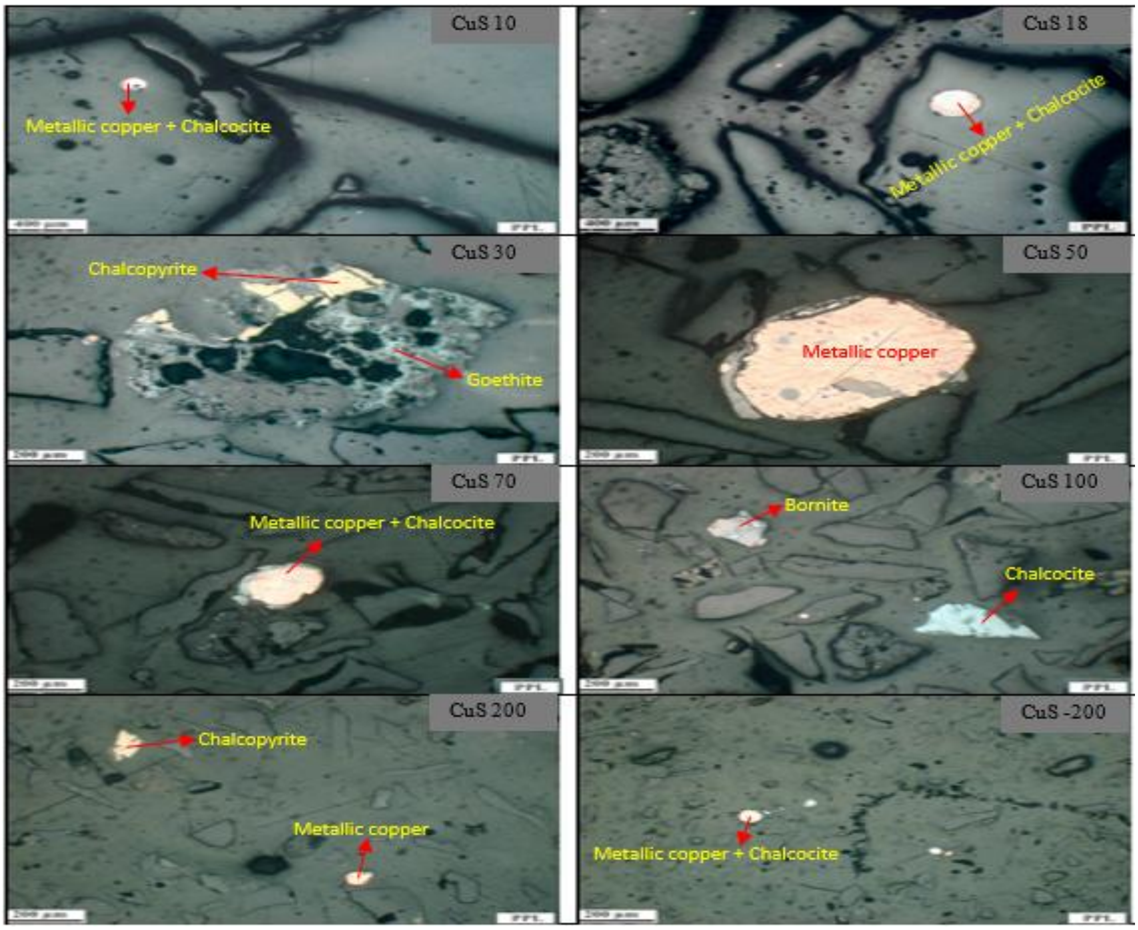


Figure 4.Polished sections of roll crusher discharge in the range of 10-200 meshes.

To determine a suitable particle size, the degree of freedom for the copper-bearing minerals should be acceptable. The degree of freedom for various particle sizes of the copper slags is presented in Table 6. As it can be seen in this table, the degree of freedom of the copper-bearing minerals in the copper slags is 90% for the particle sizes smaller than 200 meshes. Since this fraction has the highest degree of freedom, the best particle size used to perform the flotation tests was selected to be smaller than 200 meshes (74 µm).

Table 6. Degree of freedom for various particle sizes of Bardaskan's copper smelting slags.

Size (mesh)	+10	+18	+30	+50	+70	+100	+200	-200
Degree of freedom (%)	5	5-10	10	10	15-20	25-30	70-75	90

2.5. Selection of grinding time to achieve acceptable flotation particle size

For grinding the copper slags to reach an acceptable floatation particle size, the Denver's rod mill was used. 1 kg of the copper slag sample with a size smaller than 3.2 mm, which was the product of a roll crusher, together with 1 L of water was poured into the Denver's rod mill. To determine the grinding time using this mill, three grinding times including 40, 55, and 65 min were considered. After grinding, filtration and sieve analysis of the copper slag sample were carried out. After the particle size distribution, the best grinding time was obtained for the copper smelting slags.

Based on the results obtained, a 65 min grinding time was selected to achieve an acceptable particle size; with this grinding time, 85%of the copper slag particles were smaller than 74 µm. The results obtained for the particle size distribution for discharging the Denver's rod mill, after grinding to 65 min are shown in Table 7 and Figure 5.

Table 7.Sieve analysis for discharge of Denver's rod mill (grinding time was 65 min).

Size (µm)	Weight remaining on screen (g)	Weight percentage remaining on screen	Cumulative weight percent passing from screen
–150	0	0	100
–150+74	32.43	15	85
–74+53	66	30.53	54.47
–53+38	32.8	15.17	39.3
–38	84.97	39.3	0
Sum	216.2	100	-

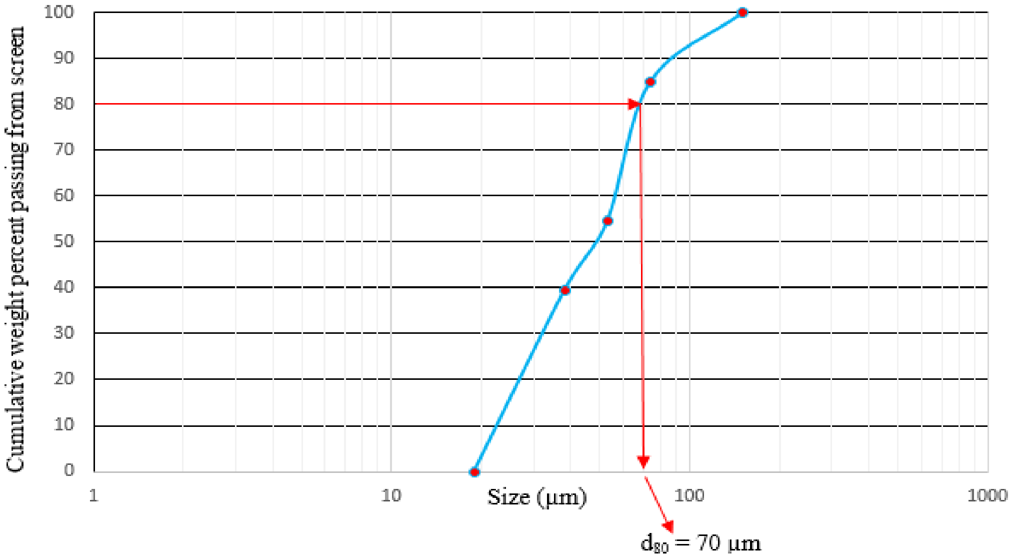
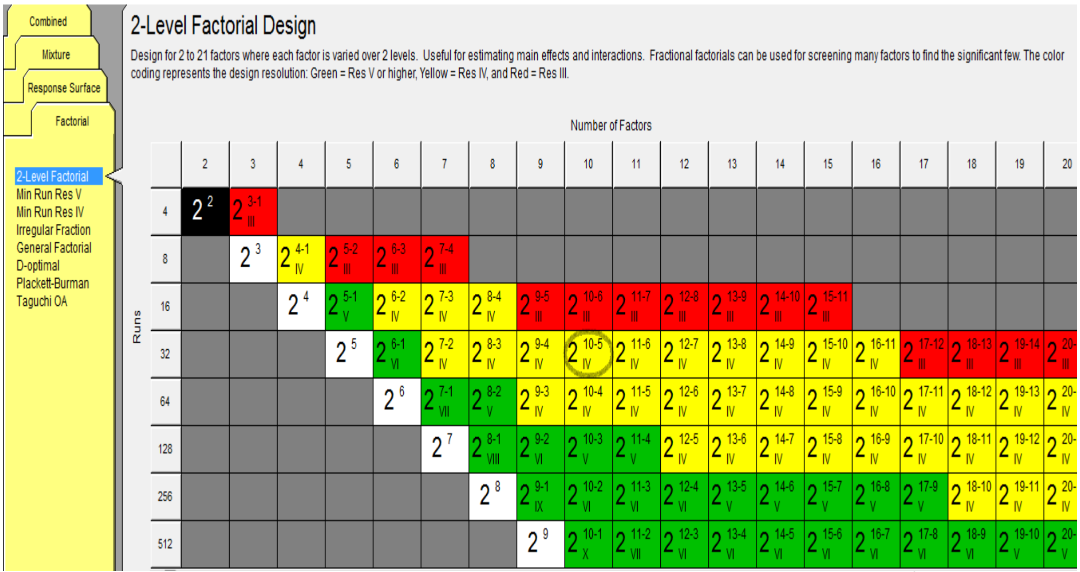


Figure 5.Particle size distribution curve for discharge of Denver's rod mill (grinding time was 65 min).

2.6. Copper slag flotation

To do the flotation tests, which were designed by the DX7 software, a Denver's flotation machine was used. A plan of 2^{10-5} was considered for these tests. This plan that designed 32 tests is shown in Figure 6.

After designing the experiments, 1 kg of the copper slag sample was poured into a flotation pulp. The pulp volume of this flotation machine was 2 L. In the tests performed, the stirrer speed was considered to be 1400 rpm, and lime was used to adjust the pH value. Air with a flow rate of 8 dm^3/min was blown into the pulp, and the pulp temperature was between 18 and 25°C.



2-Level Factorial Design

Design for 2 to 21 factors where each factor is varied over 2 levels. Useful for estimating main effects and interactions. Fractional factorials can be used for screening many factors to find the significant few. The color coding represents the design resolution: Green = Res V or higher, Yellow = Res IV, and Red = Res III.

Number of Factors

Runs	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
4	2^2	2^{3-1}_{III}																	
8		2^3	2^{4-1}_{IV}	2^{5-2}_{III}	2^{6-3}_{III}	2^{7-4}_{III}													
16			2^4	2^{5-1}_{V}	2^{6-2}_{IV}	2^{7-3}_{IV}	2^{8-4}_{IV}	2^{9-5}_{III}	2^{10-6}_{III}	2^{11-7}_{III}	2^{12-8}_{III}	2^{13-9}_{III}	2^{14-10}_{III}	2^{15-11}_{III}					
32				2^5	2^{6-1}_{VI}	2^{7-2}_{IV}	2^{8-3}_{IV}	2^{9-4}_{IV}	2^{10-5}_{IV}	2^{11-6}_{IV}	2^{12-7}_{IV}	2^{13-8}_{IV}	2^{14-9}_{IV}	2^{15-10}_{IV}	2^{16-11}_{IV}	2^{17-12}_{III}	2^{18-13}_{III}	2^{19-14}_{III}	2^{20-}_{III}
64					2^6	2^{7-1}_{VII}	2^{8-2}_{V}	2^{9-3}_{IV}	2^{10-4}_{IV}	2^{11-5}_{IV}	2^{12-6}_{IV}	2^{13-7}_{IV}	2^{14-8}_{IV}	2^{15-9}_{IV}	2^{16-10}_{IV}	2^{17-11}_{IV}	2^{18-12}_{IV}	2^{19-13}_{IV}	2^{20-}_{IV}
128						2^7	2^{8-1}_{VIII}	2^{9-2}_{VI}	2^{10-3}_{V}	2^{11-4}_{V}	2^{12-5}_{IV}	2^{13-6}_{IV}	2^{14-7}_{IV}	2^{15-8}_{IV}	2^{16-9}_{IV}	2^{17-10}_{IV}	2^{18-11}_{IV}	2^{19-12}_{IV}	2^{20-}_{IV}
256							2^8	2^{9-1}_{IX}	2^{10-2}_{VI}	2^{11-3}_{VI}	2^{12-4}_{VI}	2^{13-5}_{V}	2^{14-6}_{V}	2^{15-7}_{V}	2^{16-8}_{V}	2^{17-9}_{V}	2^{18-10}_{IV}	2^{19-11}_{IV}	2^{20-}_{IV}
512								2^9	2^{10-1}_{X}	2^{11-2}_{VII}	2^{12-3}_{VI}	2^{13-4}_{VI}	2^{14-5}_{VI}	2^{15-6}_{VI}	2^{16-7}_{VI}	2^{17-8}_{VI}	2^{18-9}_{VI}	2^{19-10}_{V}	2^{20-}_{V}

Figure 6. Designing flotation tests using DX7 software (partial factorial plan of 2^{10-5} was selected).

The effects of ten parameters including the type and dosage of collector, type and dosage of modifier, type and dosage of frother, type and dosage of depressant, sodium silicate dosage, and pH value were studied in the flotation process. Potassium amyl xanthate (Z_6) and sodium isopropyl xanthate (Z_{11}) as collectors, AERO 3477 and AERO 208 as modifiers, starch and carboxy methyl cellulose (CMC) as depressants, methyl isobutyl carbonyl (MIBC) and propylene glycol (A_{65}) as frothers, and sodium silicate as divisive were used. These parameters and their levels are shown in Table 8.

Table 8.Parameters and their levels to design flotation tests.

Parameter	Parameter name	Parameter type	Unit	Low level	High level
A	Type of collector	categoric	-	Z_6	Z_{11}
B	Collector dosage	numeric	g/t	10	40
C	Type of modifier	categoric	-	AERO 3477	AERO 208
D	Modifier dosage	numeric	g/t	10	50
E	Type of frother	categoric	-	MIBC	A_{65}
F	Fротherdosage	numeric	g/t	20	40
G	Type of depressant	categoric	-	starch	CMC
H	Depressant dosage	numeric	g/t	0	150
J	Divisive dosage	numeric	g/t	0	500
K	pH	numeric	g/t	8	11

3. Results and discussion

After carrying out the flotation tests, the results obtained were investigated for the primary evaluation using the DX7 software. Based on the results obtained, the following equation was suggested for Cu recovery as the model:

$$Cu\ recovery = 54.92 + 3.22B + 4.09D + 9.71E + 2.85F - 1.99G + 4.50K - 3.39AF - 2.29BD - 2.17BG + 1.50DF$$

In this model, all the factors involved were coded; the factors that had more coefficient values were more effective on the recovery of copper. The most significant parameters affecting the Cu recovery were the modifier dosage (D), frother type (E), and pH value (K). The results obtained for the analysis of variance (ANOVA) for the suggested model were tabulated in Table 9.

The results obtained for ANOVA (Table 9) showed that the selected model used for the copper recovery was in the confidence level of 95% (the Pvalue for the model was less than 0.05), and that it was significant. Also the most significant parameters involved including the modifier dosage (D), frother type (E), and pH value (K) had Pvalues less than 0.0001, and so they were the most effective parameters. The effects of these parameters, and the other significant ones involved, on the Cu recovery are described in the following section.

Table 9. ANOVA results for Cu recovery.

Source	Sum of squares	Degree of freedom	Mean square	F-value	P-value Prob> F
Model	5678.93	10	567.89	24.83	< 0.0001
B	330.82	1	330.82	14.47	0.0010
D	535.71	1	535.71	23.43	< 0.0001
E	3020.00	1	3020.00	132.07	< 0.0001
F	259.18	1	259.18	11.33	0.0029
G	127.32	1	127.32	5.57	0.0281
K	648.27	1	648.27	28.35	< 0.0001
AF	368.09	1	368.09	16.10	0.0006
BD	167.31	1	167.31	7.32	0.0133
BG	150.55	1	150.55	6.58	0.0180
DF	71.67	1	71.67	3.13	0.0912
Residual	480.21	21	22.87		
Cor Total	6159.14	31			

3.1. Effects of significant parameters on Cu recovery

3.1.1. Effect of modifier dosage

This parameter had a significant effect on the copper recovery, such that by increasing the modifier dosage from 10 g/t to 50 g/t, the Cu recovery was enhanced from 59.11 to 71.70%.This effect occurred when the pH value was adjusted on its high level and A_{65} was used as the frother. The effect of the modifier dosage is shown in Figure 7.

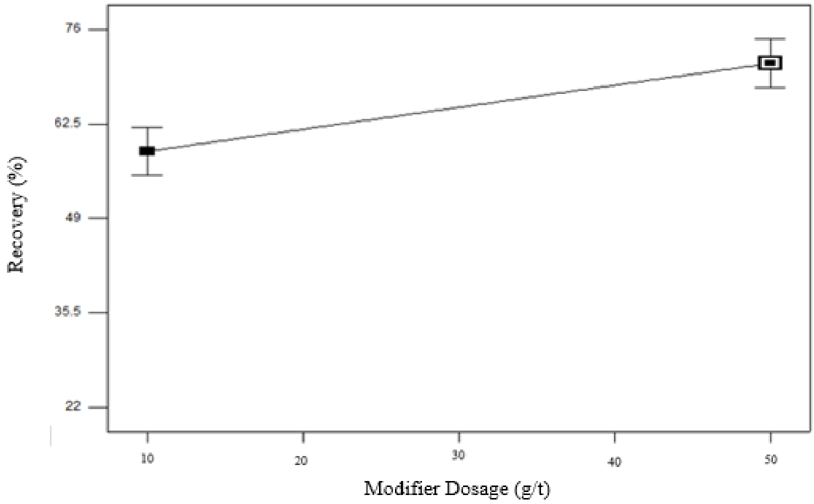


Figure 7. Effect of modifier dosage on recovery of Cu from copper slags.

3.1.2. Effect of pH value

This effect is shown in Figure 8. As it can be seen in this figure, when A_{65} was used as the frother, the recovery of Cu from the copper slags with the pH value of 11 was more than the copper recovery with the pH value of 8 (30.71% versus 30.62%). However, when the modifier dosage was enhanced to its high level, a little increase was observed in the Cu recovery.

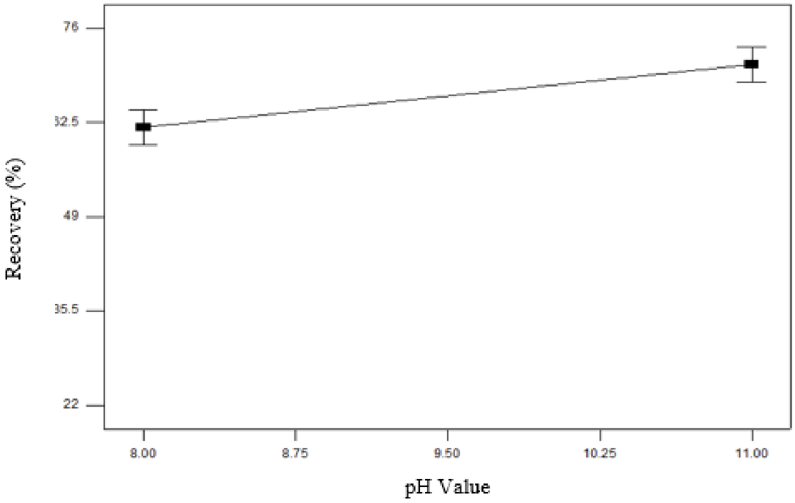


Figure 8. Effect of pH on recovery of Cu from copper slags.

3.1.3. Effect of frother type

Figure 9 shows the effect of the frother type on the recovery of Cu from the copper slags. According to Figure 9, with a change in the frother type from MIBC to A_{65}, Cu recovery increased from 47.20% to 66.63%. In the same conditions, by enhancing the modifier dosage and pH values to their high levels, a little increase was observed in the copper recovery.

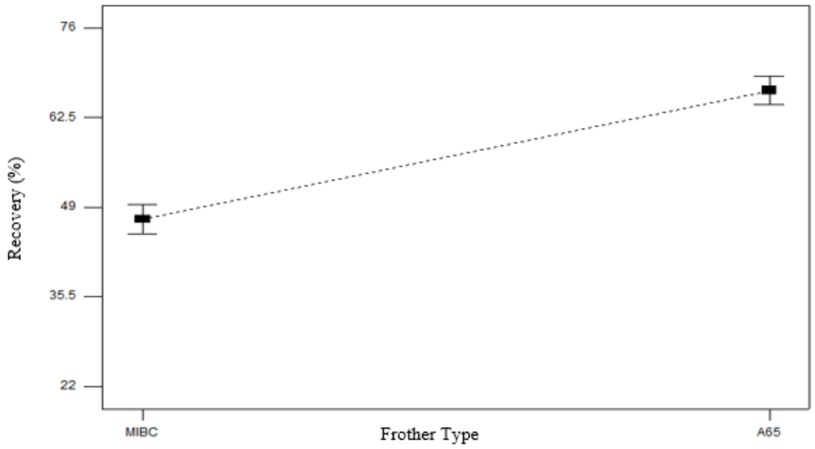


Figure 9. Effect of frother type on recovery of Cu from copper slags.

3.1.4. Effect of interaction between collector dosage and modifier dosage

Another factor that affects the recovery of copper is the interaction between collector dosage and modifier dosage, in which CMC is used as a depressant. When the modifier dosage was adjusted at 50 g/t (red line in Figure 10), the Cu recovery decreased by increasing a low level of the collector dosage to a high level of this parameter, so that when the collector dosage was adjusted at 10 g/t, the copper recovery was obtained to be 67.76%, and when the collector dosage was adjusted at 40 g/t, the copper recovery was 65.28%. Conversely, when the modifier dosage was adjusted at 10 g/t (black line in Figure 10), the Cu recovery increased with increase in the collector dosage. Therefore, it is obvious that by changing the modifier dosage, the trend in the copper recovery from a low level of the collector dosage to a high level of this parameter changed.

According to Figure 10, when the modifier dosage was adjusted at 50 g/t and the collector dosage was adjusted at 10 g/t, a maximum Cu recovery (67.76%) was achieved.

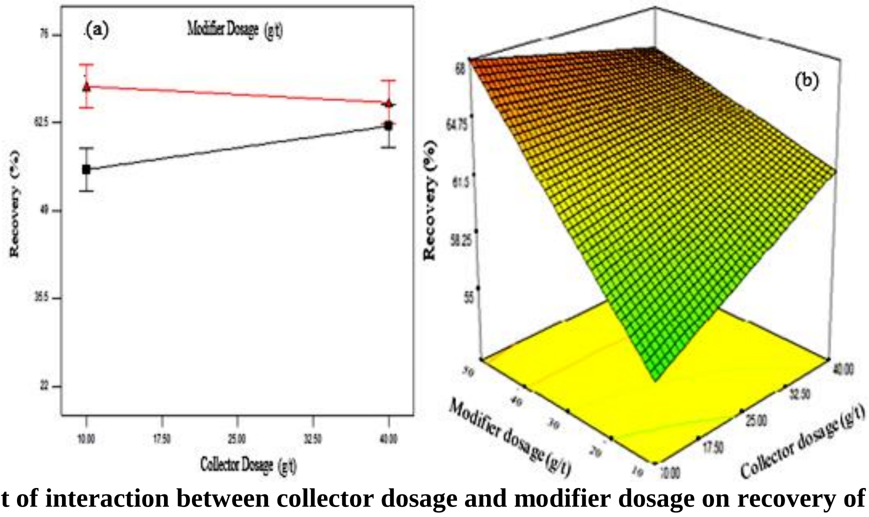


Figure 10. Effect of interaction between collector dosage and modifier dosage on recovery of Cu from copper slags (a:2D state, and b: 3D state).

3.1.5. Effect of interaction between collector dosage and depressant type

Effect of the interaction between collector dosage and depressant type on the recovery of copper is shown in Figure 11. As it can be seen in this figure, when the modifier dosage was adjusted at its highest level and CMC was used as the depressant (red line), with changing a low level of the collector dosage to a high level of this parameter, the Cu recovery decreased from 68.01 to 65.65%, and when starch was used as the depressant (black line), with increasing a low level of the collector dosage to a high level of this parameter, the copper recovery increased from 67.66 to 74.03%.

Thus it is clear that by changing the type of depressant, the trend of Cu recovery from a low level of the collector dosage to a high level of this parameter changed.

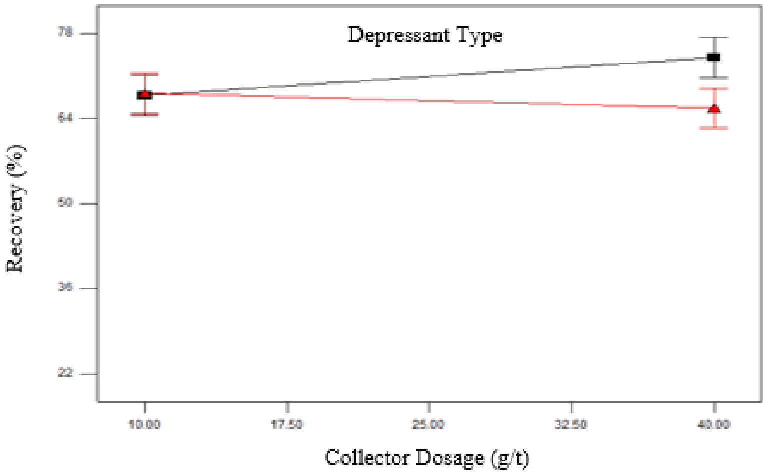


Figure 11. Effect of interaction between collector dosage and depressant type on recovery of Cu from copper slags in the presence of 50 g/t of modifier dosage.

3.1.6. Effect of interaction between collector type and frother dosage

The last effective factor involved is the interaction between the collector type and frother dosage. When the frother dosage was adjusted at 40 g/t (red line in Figure 12), with change in the collector type from Z_6 to Z_{11}, the Cu recovery decreased from 72.89 to 66.11%, and when the frother dosage was adjusted at 20 g/t (black line in Figure 12), with change in the collector type from Z_6 to Z_{11}, the copper recovery increased from 60.41 to 67.20%.

As it can be seen in Figure 12, when the frother dosage was adjusted at 40 g/t and Z_6 was used as the collector, maximum Cu recovery (72.89%) was achieved.

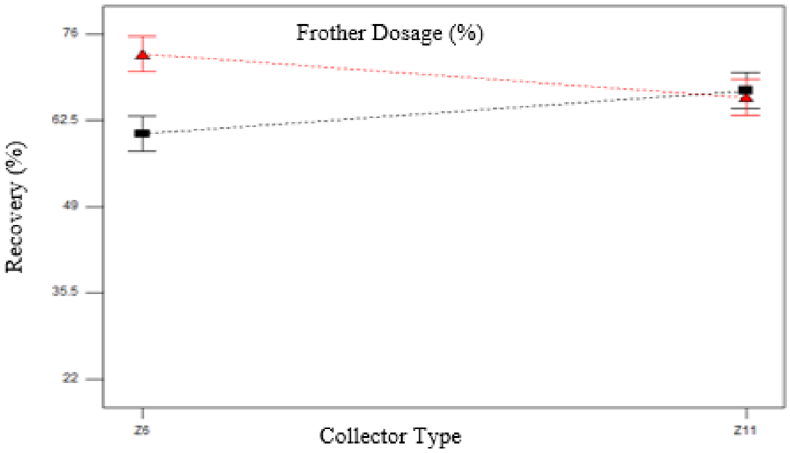


Figure 12. Effect of interaction between collector type and frother dosage on recovery of Cu from copper slags.

3.2. Optimization

To optimize the experiments, the DX7 software was used to maximize the recovery of copper from the slags. The optimal conditions predicted by the DX7 software used to obtain the maximum Cu recovery are shown in Table 10.

Table10. Optimal conditions predicted by DX7 software.

Parameter name	Collector type	Collector dosage (g/t)	Modifier type	Modifier dosage (g/t)	Depressant type	Depressant dosage (g/t)
Proposed level	Z_6	10	AERO3477	50	CMC	140.39
Parameter name	Frother type	Frother dosage (g/t)	pH value	Cu recovery (%)	Cu grade (%)	d
Proposed level	MIBC	40	11	60.78	5.47	0.90

Sodium silicate was practically eliminated; because this material did not have a significant effect on the Cu recovery. Although A_{65} as the frother, was more effective on the copper recovery, MIBC was used as the frother, because when A_{65} was used in the tests as the frother, the Cu grade was greatly reduced.

Optimization with the DX7 software was confirmed after doing a test at the optimal conditions. The results obtained for this test are presented in Table 11.

Table 11 shows that the results predicted by the DX7 software are approximately closed to experimental ones. Thus optimization of the tests was performed truly. Also, according to Table 11, it is obvious that by doing a test under the optimal conditions, the recovery and grade of copper were achieved to be 62.23% and 5.38%, respectively, in the rougher step.

Table 11. Predicted results by software and experimental results.

Results	Predicted results	Experimental results
Cu recovery (%)	60.78	62.23
Cu grade (%)	5.47	5.38

3.3. Effect of grinding time on recovery of copper

The high work index for the Bardaskan copper slags indicates that for grind the copper smelting slags, too much energy should be spent, and that these materials require a lot of grinding time. The grinding time for these copper slags was obtained to be 65 min, and with this grinding time, 90% of the copper-bearing minerals were released. To evaluate the effect of grinding time on the copper recovery, this factor was increased, and the results obtained were tabulated in Table 12.

Table 12.Effect of grinding time on recovery of copper from smelting slags.

Test number	Grinding time (min)	d_{80} (μm)	Copper recovery (%)	Copper grade (%)
1	70	65	67.21	5.32
2	75	58	71.89	5.25
3	80	53	75.11	5.14
4	85	48	79.89	5.09
5	90	42	70.65	5.22
6	95	39	65.27	5.36

According to Table 12, it is clear that with increase in the grinding time to 85 min, the copper recovery from the slags was approximately enhanced to 80%, but with increase in the grinding time to 95 min, the recovery of copper decreased to 65.27%, which is the reason for its attribution to the presence of fine particles in excessive amount (these fine particles were considered as a negative factor in the flotation process). All the tests (with their results presented in Table 12) were performed at the optimal conditions (determined by the DX7 software), and only the grinding time of the copper slags was changed to reach the optimal particle size. The 65 min grinding time was changed to 85 min. The copper grade in these tests had no noticeable difference with the copper grade determined by the DX7 software at the optimal conditions, and it fluctuated in the range of 5.09%-5.36%.

4. Conclusions

Copper slags are hard materials that require a lot of grinding time to reach the optimal particle sizes for the flotation process. The copper smelting slags in the Bardeskan district with a work index of 16.24 kwh/st should be grinded to 65 minutes to reach the desired degree of freedom for carrying out the flotation tests with particle size of 80%, smaller than 70 μm. Byperforming the flotation tests with this grinding time at the optimal conditions (10 g/t Z_6 as collector, 50 g/t R3477 as modifier, 40 g/t MIBC as frother, 140.39 g/t CMC as depressant and pH=11), the copper recovery and grade were achieved to be 62.23% and 5.38%, respectively, in the rougher step. To increase the Cu recovery in the rougher step and to reach the optimal particle sizes for the flotation process, grinding time was increased. By enhancing the grinding time to 85 minutes, the copper recovery increased to 79.89%, which is due to the release of the copper-bearing minerals from gangue minerals in the slag and reaching the optimal particle size. However, with increasing the grinding time to 95 minutes, the recovery of Cu from the slags decreased to 65.27%, which is due to the presence of fine particles.

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References

[1]. Shen, H. and Forssberg, E. (2003). An overview of recovery of metals from slags. Waste Management. 23: 933-949.

[2]. Mobasher, B., Devaguptapu, R. and Arino, A.M. (1996). Effect of copper slag on the hydration of blended cemetitious mixtures, Proc, American Society of Civil Engineers on Fourth Materials Engineering Conference, Materials for the New Millennium, Washington, DC, 10-14 November, 1677-1686.

[3]. Johansen, E.B., Rosenqvist, T. and Torgerson, P.T. (1970). On the thermodynamics of continuos copper smelting. J Metals. 22: 39-47.

[4]. Biswas, A.K. and Davenport, W.G. (1980). Extractive metallurgy of copper,Oxford: Pergamon Press 2ndед.

[5]. Shi, C. and Qian, J. (2000). High performance cementing material from industrial slags-a review. Resour, Conserv Recycling. 29: 195-207.

[6]. Gorai, B., Jana, R.K. and Premchand. (2003). Characteristics and utilization of copper slag-a review. Resources, Conservation and Recycling. 39: 299-313.

[7]. Marghussian, V.K. and Maghsoodipoor, A. (1999). Fabrication of unglazed floor tiles containing Iranian copper slags. Ceramics Int. 25: 617-622.

[8]. Kiyak, B., Ozer, A., Altundogan, S.H., Erdem, M. and Tumen, F. (1999). Cr (VI) reduction in aqueous solution by using copper smelter slag. Waste Manage. 19: 333-338.

[9]. Imris, I., Rebolledo, S., Sanchez, M., Castro, G., Achurra, G. and Hernandez, F. (2000). The copper losses in the slags from the EL Teniente process. Can Metall Quart. 39(3): 281-290.

[10]. Agrawal, A., Sahu, K.K., Pandey, B.D. and Premchand. (2000). State of the art report on metallurgical waste, Report submitted to Ministry of Environment & Forests by National Metallurgical Laboratory, 64-67.

[11]. Yucel, O., Addemir, O., Tekin, A. and Nizamoglu, S. (1992). Recovery of cobalt from copper slags. Miner Process Extractive Metall Rev. 10: 99-107.

[12]. Osborn, G.A., Garner, F.A. and Veasey, T.J. (1990). Pilot-Plant study for recovery of copper metal from secondary copper slag by froth flotation, Proc, Recycling of Metalliferous Materials Conference, Birmingham, UK, April, 23-25.

[13]. Sigedin, V.N., Piletskii, V.M. and Glazunov, L.A. (1986). Removal of copper from slags of the Almalyk copper-smelting plant. Tsvetn Met. 10: 18-23.

[14]. Oprea, F.I. and Murgulescu, R. (1984). The recovery of copper from gangues resulted from copper slag flotation. Metalurgia (Bucharest). 36(10): 505-506.

[15]. Natarajan, K.A., Quadri, S.G. and Shivkumar, S. (1980). Recovery of copper from Khetri slag flotation tailings. Trans Indian Inst Met. 33(5): 377-381.

[16]. Clarkson, J.F., Johnson, R.H., Siegal, E. and Vlasak, W.M. (1978). Utilizing Smelter Slags at White Pine Copper Division Conference Materials Recycling, An Overview of the Sixth Mineral Waste Utilization Symposium, Chicago, 2-3 May, 169-173.

تأثیر زمان خردایش روی بازیابی فلوتاسیون سرباره‌های ذوب مس منطقه بردسکن

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چکیده:

سرباره‌های ذوب مس مواد سختی هستند که برای بازیابی مس از آن‌ها به روش فلوتاسیون، خردایش زیادی باید صورت گیرد تا به دانه‌بندی بهینه برای انجام فرآیند فلوتاسیون برسد. سرباره‌های مس منطقه بردسکن با اندیس کار kwh/st ۱۶/۲۴، نیاز به ۶۵ دقیقه خردایش داشت تا به درجه آزادی قابل قبول برای آزمایش‌های فلوتاسیون با دانه‌بندی ۸۰٪ کوچک‌تر از ۷۰ میکرون برسد. با این زمان خردایش، درجه آزادی ۹۰-۸۵ درصد برای کانی‌های مس‌دار حاصل شد. روش انجام آزمایش‌های فلوتاسیون، شناسایی پارامترهای مؤثر و بهینه‌سازی آزمایش‌ها در این مقاله شرح داده شده است. نتایج آزمایش‌هایی که در شرایط بهینه پس از ۶۵ دقیقه خردایش انجام شد، بازیابی مس را ۶۲/۲۳٪ نشان داد؛ در حالی‌که با فلوتاسیون سرباره‌های مس در شرایط بهینه پس از افزایش زمان خردایش تا ۸۵ دقیقه (d_{80} برابر با ۴۸ میکرون)، بازیابی مس تا ۷۹/۸۹٪ افزایش یافت.

کلمات کلیدی: زمان خردایش، سرباره‌های مس، فلوتاسیون، بازیابی، اندیس کار.