Original Article

The critical importance of pulp concentration on the flotation of galena from a low grade lead–zinc ore

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A B S T R A C T

The Qixia orebody is a complex lead–zinc sulfide system with pyrite gangue and minor amounts of copper. In order to improve the flotation results, laboratory scale flotation testing of ore samples taken from this operation was performed. Flotation tests used a sequential recovery protocol for selective flotation of first the lead and thereafter the zinc. The key parameters that influence flotation performance of lead mineral were tested in this paper. The test data show that, for comparable collector, grinding time, flotation pH and solid-in-pulp concentration, the increase of solid-in-pulp concentration has the most significant effect on the recovery and selective separation of lead mineral. The increase of solid-in-pulp concentration from 27% to 55% makes the recovery of lead mineral increased from 60% to 80% and the lead grade increased from 27.5% to 29.1%.

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1. Introduction

Lead and zinc metals are widely used in the fields such as electrical, mechanical, military, metallurgical, chemical, light and medical [1]. As an important resource of lead and zinc mineral, sulfide ore plays an important role in the development of world economy. However, in most cases, lead–zinc–iron sulfide ores are grouped together in the deposits [2].

In general, there are two basic approaches to achieve lead–zinc sulfide mineral separation: to depress zinc sulfide and float lead sulfide, or a bulk lead sulfide and zinc sulfide concentrate is floated first, followed by the Pb–Zn separation [3,4]. In the past decades, many studies on flotation separation of lead–zinc sulfide ores have been carried out [5–7].

The Qixia orebody located in Jiangsu province of China is a complex lead–zinc sulfide system with pyrite gangue and minor amounts of copper. In order to improve the flotation
performance of this ore, many studies have been done. However, the flotation performance is difficult to be improved although many reagents and flotation flowsheets have been tested. In this study, some factors that may influence the flotation performance of this ore were tested. The objective of this study is to point out a solution to the flotation problem of low grade lead–zinc sulfide ore.

2. Experimental

2.1. Materials and reagents

The lead–zinc ore was supplied by Nanjing Yinmao Lead–zinc Mining Co., Ltd., China. The ore is of low lead grade (approximately 1.3%). The results of the chemical analysis are shown in Table 1. A quantitative mineralogy determination using X-ray powder diffraction (XRD) and mineral liberation analyzer (MLA) were done by the analytical laboratory and the results indicated that the main valuable minerals are galena, sphalerite and the gangue minerals are pyrite, quartz and calcite (Table 2). According to this result, the flotation flowsheet and the reagents can be determined.

The reagents used were: depressant (Lime, Zinc sulfate and Sodium silicate), collector (Diethyldithiocarbamate; Ammonium butyl aerofloat; Potassium butylxanthate and Thiamine ester) and frother (Methyl isobutyl carbinol).

2.2. Methods

2.2.1. Grinding

Ore samples were crushed to –2 mm, riffled into representative samples of 500 g, purged with nitrogen during storage. For each flotation experiment, samples were ground in a mild steel rod mill for a certain time. Lime, as pH regulator was added at the grinding stage.

2.2.2. Flotation

As shown in Fig. 1, the flotation tests were performed in a XFD-63 flotation cell (self aeration) whose volume for flotation was 1.5L using an agitation speed of 1800 rpm. The solid density in the flotation cell was changed according to the experimental requirements. During the conditioning, depressant (Zinc sulfate and Sodium silicate), collector and frother (Methyl isobutyl carbinol) were added and conditioned for 5 min, respectively, to allow reagent adsorption. After the conditioning time, the flotation of lead minerals started with the injection of air in the flotation cell, the air flow rate was kept at 0.1 Nm³/h monitored with a flow meter. Flotation was performed for 5 min and the concentrate was collected. In this paper, only the flotation of lead minerals was shown.

3. Results and discussion

As most minerals are finely disseminated and intimately associated with the gangue minerals, they must be initially liberated before separation can be undertaken [8–10]. The effective liberation of lead minerals is the foundation of separating lead minerals. Thus, the effect of grinding time on lead mineral flotation was investigated and the results are shown in Fig. 2. The grinding pulp density was about 66%. The detailed flotation experimental conditions are as follows:

<p>| Table 1 – The main chemical composition of the lead–zinc ore sample (mass fraction,%). |</p>
<table>
<thead>
<tr>
<th>Element</th>
<th>Cu</th>
<th>Pb</th>
<th>Zn</th>
<th>Fe</th>
<th>S</th>
<th>Mn</th>
<th>CaO</th>
</tr>
</thead>
<tbody>
<tr>
<td>Content</td>
<td>0.12</td>
<td>1.30</td>
<td>3.15</td>
<td>28.88</td>
<td>30.06</td>
<td>2.27</td>
<td>6.68</td>
</tr>
</tbody>
</table>

<p>| Table 2 – The mineral composition of the lead–zinc ore sample (%). |</p>
<table>
<thead>
<tr>
<th>Mineral</th>
<th>Content</th>
<th>Mineral</th>
<th>Content</th>
<th>Mineral</th>
<th>Content</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pyrite</td>
<td>43.45</td>
<td>Graphite</td>
<td>0.50</td>
<td>Marcasite</td>
<td>Trace</td>
</tr>
<tr>
<td>Sphalerite</td>
<td>9.50</td>
<td>Quartz</td>
<td>14.06</td>
<td>Magnetite</td>
<td>Trace</td>
</tr>
<tr>
<td>Galena</td>
<td>4.28</td>
<td>Clay minerals</td>
<td>1.52</td>
<td>Pyrrhotite</td>
<td>Trace</td>
</tr>
<tr>
<td>Chalcopyrite</td>
<td>0.26</td>
<td>Calcite</td>
<td>25.10</td>
<td>Covellite</td>
<td>Trace</td>
</tr>
<tr>
<td>Tetrahedrite</td>
<td>0.30</td>
<td>Sericite</td>
<td>1.00</td>
<td>Hematite</td>
<td>Trace</td>
</tr>
</tbody>
</table>
solid concentration: 35%; the dosages of lime, zinc sulfate, sodium silicate, thiamine ester and methyl isobutyl carbinol were 800, 600, 300, 60 and 21 g/t, respectively. The grinding time increased from 4.5 to 7 min. According to the results shown in Fig. 2, the increase of grinding time from 4 to 5.8 min increased the lead mineral recovery from 66% to 70.5% and decreased the lead grade from 31% to 29%. However, grinding time longer than 5.8 min produces little change in the flotation recovery and selective separation of the lead mineral. Thus, 5.8 min (5 min and 48 s) is considered as the optimum grinding time. When the grinding time is 5.8 min, the fraction of particles below 0.074 mm approximately accounts for 80 wt% of the feed.

The floatability of pyrite is similar to galena under low pH condition, which is harmful to the separation of galena from sulfur minerals. However, the galena will be depressed as well as pyrite if the pH is too high [11]. Thus, the pH value of the flotation pulp should be neither too high nor too low. The effect of lime dosage on the lead flotation was studied and the results are shown in Fig. 3. The detailed flotation experimental conditions are as follows: solid concentration: 35%; the dosages of zinc sulfate, sodium silicate, thiamine ester and methyl isobutyl carbinol were 600, 300, 60 and 21 g/t, respectively. It can be seen from the results shown in Fig. 3 that the increase of lime dosage from 500 to 800 g/t greatly improved the grade of the lead concentrate without the change of recovery. According to Fig. 3, the optimum lime dosage is 800 g/t and the pulp pH is 9 at this time.

The effects of distinct kinds of collectors on the flotation performance of lead mineral were studied and the results are shown in Fig. 4. The detailed flotation experimental conditions are as follows: solid concentration: 35%; the dosages of lime, zinc sulfate, sodium silicate and methyl isobutyl carbinol were 800, 600, 300 and 21 g/t, respectively. The results show that thiamine ester has selective collecting ability to lead mineral and produces a concentrate with lead grade of 32%, which is higher than other collectors. So, thiamine ester was chosen as the flotation collector of this ore.

The effect of collector (thiamine ester) dosage on the flotation of lead mineral was also studied and the results are shown in Fig. 5. The increase of collector dosage from 40 to 60 g/t increased the concentrate recovery from 64.5% to 68.5% and then the recovery never changed with further increase of collector dosage. On the other hand, the lead grade of concentrate decreases from 32.13% to 29.11% with the increase of collector dosage from 60 to 100 g/t. Hence, collector dosage 60 g/t was maintained as it gives the optimal flotation index.

All the conventional parameters that may influence the flotation performance of lead minerals were tested and
optimized. However, the recovery of lead mineral is still low. Drawing on the literature in flotation area, a dependence of mineral recovery upon solid-in-pulp concentration can be seen [12,13]. So, the effect of solid-in-pulp concentration on the flotation of lead mineral was studied and the results are shown in Fig. 6. The results in Fig. 6 indicate that the recovery of lead mineral increased from 60% to 80% and the lead grade increased from 27.5% to 29.1% with the increase of solid-in-pulp concentration from 27% to 50%. Thus, 50% solid concentration enables optimize action of reagents on the mineral, and improves the flotation results.

The mass balance for flowsheet 1 is shown in Table 3. The lead concentrate contains 2.40% of the original mass, and grades 46.17 wt% Pb, 5.78 wt% Zn, compared to a feed composition of 1.3 wt% Pb and 3.15 wt% Zn. A zinc concentrate with grades of 0.69 wt% Pb, 49.49 wt% Zn was also obtained.

4. Conclusions

The flotation of lead mineral from a low grade lead–zinc sulfide ore has been investigated using a laboratory scale mechanical flotation cell. The effect of various operating parameters including type and concentration of collectors, pulp pH, grinding time, solid-in-pulp concentration on the flotation behavior of lead mineral have been studied. The following conclusions can be made from the experimental results. (a) Thiamine ester is a suitable collector for the flotation of lead–zinc sulfide ore and the optimum collector concentration is 60 g/t. (b) The optimum pH for the process when using CaO as modifier is 9 and the optimum grinding time is 5.8 min. (c) The solid-in-pulp concentration is the most important parameter that will influence the flotation of lead mineral. The recovery and lead grade increased with the increase of solid-in-pulp concentration until 50%.

Conflicts of interest

The authors declare no conflicts of interest.

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