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# DE-ZINCING OF LEAD-COPPER SULPHIDE MINERALS FLOTATION

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***ABSTRACT***

Typical lead-copper-zinc sulphide minerals were found hard to be separated selectively by froth flotation in the plant practice operation due to part of sphalerite was progressively entrapped into lead-copper bulk flotation. A series of de-zincing on lead-copper flotation tests has been conducted in the laboratory to reduce unwanted zinc content. Dosages variation of common sphalerite depressant such as sodium cyanide, sodium bisulphite combined with zinc sulphate were employed in addition to flotation time and pH regulators alteration. The flotation results exhibit that lesser zinc component is still presence in lead-copper concentrate due to intricate association of the minerals particle. However, this study in general performs tendency of de-zincing requirement as well as improvement of lead-copper recovery by increasing consumption of such depressant agent and may regrind otherwise.

Keywords: froth flotation, lead-copper-zinc sulphide ore, de-zincing, flotation reagent

## INTRODUCTION

Typical ore that has been received in the laboratory is required to be tested for understanding the characteristics of lead-copper and zinc sulphide minerals flotation. According to the plant practiced that the ore has been extremely difficult to be separated selectively by froth flotation, especially due to the component of zinc mineral distributes intensively into lead-copper concentrate. This problem affects the low recovery of lead-copper, hence it could reduce the economic value of the flotation products. The present typical ore might be considered as complex sulphide minerals containing such intricate minerals association of chalcopyrite ($CuFeS_2$), sphalerite (ZnS), pyrite ($FeS_2$) and galena (PbS) in fine sizes. The fine sizes of those minerals theoretically have been no difficulty to be separated by selective flotation as long as the minerals are well liberated and no activated zinc. In the case of complex sulphide minerals, however, the flotation separation is difficult when lead, copper and zinc need to justify concentration into three perfect separate products. Flotation problems in general are due to geological origin; hence, the ore have controlling influence on flotation practice. More specifically, the resultant copper or lead ions in slurry may activate the sphalerite causing zinc is naturally float together into lead-copper concentrate (El-Shall et al. 2000; Bulatovic, 2007:156). Therefore, the depression of sphalerite is importantly required.

Flotation is principally a physico-chemical process of minerals concentration due to different mineral surfaces characteristic among other minerals content. Certain mineral surface is affected primarily by collecting agent and is supported by modifying agent to become hydrophobic and hydrophilic surfaces. Then, the hydrophobic mineral tends to attach on air bubbles and is lifted up to the froth zone leaving the hydrophilic minerals. The flotation mechanism may be illustrated as shown in Figure 1.

Selective differential flotation of lead, copper, zinc sulphide minerals has been practiced for many years (Cyanamid, 1989); however, separation problems are still encountered. In accordance with dezincing flotation process, Quast and Hobart (2006) conducted marmatite (ZnFeS) depression in galena flotation by combining two depressants of zinc sulphate ($ZnSO_4$) and potassium metabisulphite ($K_2S_2O_5$) under alkaline condition that produced

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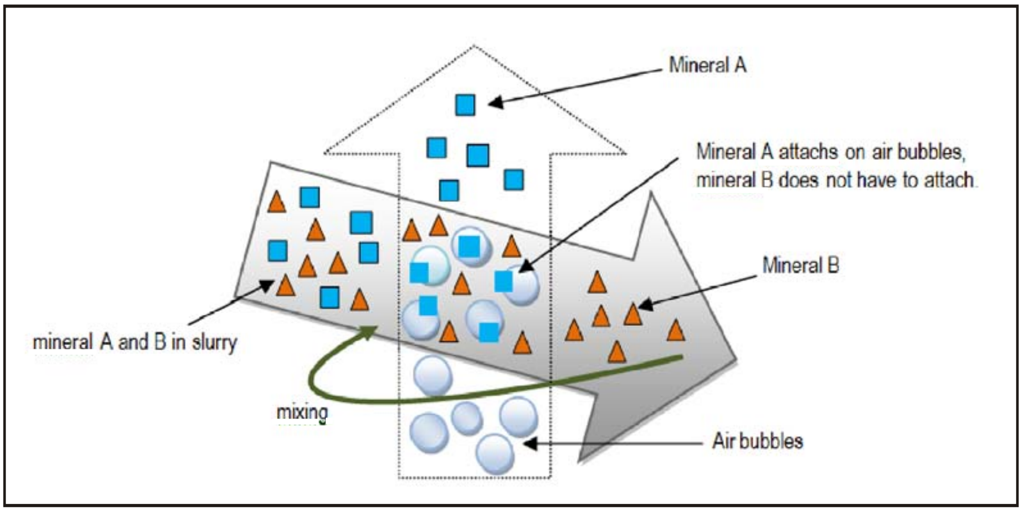


Figure 1. General mechanism of flotation process (Ngurah et al. 2010)

proper result at 98% of lead recovery with less zinc content. Based on zeta potential and adsorption measurements as well as thermodynamic calculations, El-Shall et al. (2000) substantiated that sphalerite is naturally activated by soluble lead ions in the flotation pulp; causing unwanted zinc component was floated into the lead flotation. However, zinc sulphate agent could depress the lead-activated sphalerite by considering the equilibrium constant of the reaction. Instead of soluble lead ions, Seke and Pistorius (2005) found that the present of complexed cuprous-cyanide would activate sphalerite in the pulp; hence, the depression of activated sphalerite should be fully concerned. Sodium bisulphite ($Na_2SO_3$) was reported as an effective depressant for the flotation of cuprous-activated sphalerite (Khmeleva et al. (2005). Cyanamid (1989:91) noted that zinc minerals might be activated by lime, so the use of soda ash as pH regulator would be necessary. Singh et al. (2009) conducted differential flotation of lead-zinc sulphide ore, where the ore was predominantly contained of sphalerite and pyrite in association with fine sizes of galena, but even the ore was about oxidized condition. Using alkyl quinoline-based chelating type collector improved the selectivity of flotation. Magdalinovic et al. (2004) employed succsesfully lead-zinc sulphide flotation using new SKIK-BZ2000 reagent. This typical reagent was claimed capable to completely eliminate the common reagents of NaCN, $ZnSO_4$ and PEX (potassium ethyl xanthate); even this new reagent confirmed economic benefit and has solved environmental problem. Effort on flotation separation of copper and zinc from complex sulphide of Bogor ore containing high grade sphalerite that causing a problem of zinc depression; however, by applying bulk flotation found that the use of single collector of sodium-dithiophosphate with higher carbon chain length type provided proper result (Ngurah-Ardha et al. 2010). Suppression of sphalerite at Prieska zinc-copper mine was succesful using $Zn(CN)_2$ at pH value of 6.2 to 7.2; which was controlled with sulphuric acid. The $Zn(CN)_2$ is obtained by premixing of cyanide and zinc sulphate (Michaeldev, 2010). All references cited above indicate that efforts on separating lead-copper-zinc sulphide minerals are always innovated. In the case of present work that is due to poor selectivity between galena, chalcopyrite and sphalerite; hence, de-zincing of typical lead-copper bulk flotation is studied in terms of using common parameters such as: selection of pH regulators, zinc depressant consumption using sodium cyanide, sodium bisulphite and zinc sulphate. The experimental works of de-zincing were concerned with flotation of PbCu rougher stage.

## METHODOLOGY

### Experimental and Reagents

Prior to the froth flotation tests, mineralogical characteristic of the ore was firstly observed. Based on references elsewhere (Cyanamid, 1989; Gupta and Yan, 2006; Bulatovic, 2007; Wills and Napier-Munn, 2007), flotation tests were employed in terms of standard practice of lead-copper bulk flotation at rougher stage (PbCu rougher) by suppressing zinc mineral. The zinc mineral was then activated and floated from tailing of the lead-copper flotation. In accordance with de-zincing action, the present laboratory tests of PbCu rougher flotation were concerned with parameters of flotation time (1 to 7 minutes), pH regulator of lime ($Ca(OH)_2$) compared to soda ash ($Na_2CO_3$), flotation with and without sodium cyanide (NaCN), the use of sodium–bisulphite ($Na_2SO_3$) and varying zinc-sulphate consumption ($ZnSO_4.7H_2O$; 600 – 3000 g/ton). The fixed parameter was set up firmly for collecting agent of sodium-isobuthyl-xanthate (SIBX) 200 g/ton, the frothing agent of methyl-isobutyl-carbinol (MIBC) 20 g/ton, conditioning time was held at 5 minutes. All reagents were diluted with distilled water (aquadest) at 5% solution, except the frothing agent.

### Equipment, Material and Analysis

The experimental works used standard Denver flotation tester with impeller speed of 1000 rpm. Air bubbles were introduced from natural air inlet in line with the impeller speed. Cell volume was 1000 ml and the ore feed weight was 500 g. The slurry contained 33% solid and the particles size of ore feed was made 80% of -200+325 mesh or 80% of -75+45 µm. The equipment and flotation tests were conducted in considerable standard procedure as performed in Figure 2. Test results were analyzed in terms of metal content of Pb, Cu, Zn by atomic absorption spectrometer (AAS) method.

## RESULTS AND DISCUSSION

### Ore Characteristics

Based on optical microscopic results, the numbers and kinds of minerals as well as mineral association in the ore were denoted at respective particles sizes. Hence, the ore were found containing:

- sphalerite (ZnS) as much as 14.27%; sizes between 10 µm and 1.7 mm;
- pyrite ($FeS_2$) as much as 11.24%; sizes between 5 µm and 1.2 mm;
- chalcopyrite ($CuFeS_2$) as much as 4.03%; sizes between 4 µm and 1.2 mm;
- galena (PbS) as much as 1.74%; sizes between 60 µm and 1.2 mm;
- covellite (CuS) as much as 0.15%; sizes between 40 µm and 0.34 mm; and
- gangue minerals as much as 68.57%.

The existence of fine sphalerite (20 µm or less) shows inclusion as dark dots in pyrite particles as shown in Figure 3A. While Figure 3B shows the fine size minerals of sphalerite and chalcopyrite, which are presented as intricate association in pyrite particles. Galena (approx. 30 µm) was found in complex association with sphalerite (approx. 30 µm), covellite, and small amount of chalcopyrite in addition to gangue minerals (Figure 3C). Based on AAS data for chemical analysis, it is known that the metal contents of the ore

Figure 2. Laboratory PbCu bulk flotation tests work

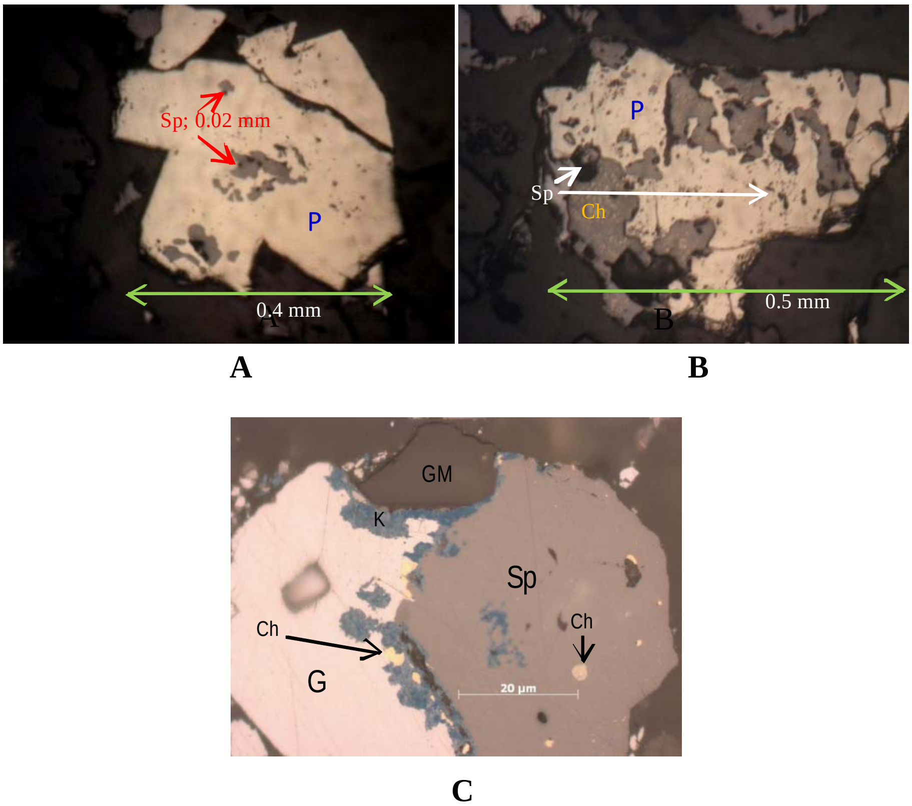


Figure 3. Optical microscopic observation of ore minerals
(A). Inclusion of fine sphalerite (Sp, 20 µm) in pyrite (P) particle.
(B). Intricate association of fine sphalerite with chalcopyrite (Ch) and pyrite.
(C). Intricate association of galena (G), sphalerite, covellite (K) and small chalcopyrite with gangue mineral (GM) within total size of 77µm

are 5.96% Pb; 1.53% Cu and 9.87% Zn. This typical ore is considered as high-grade lead-zinc complex sulphide ore.

**Flotation Test Result with Condition Similar to Plant Practice**

The flotation test was performed by rougher flotation of lead-copper minerals (rougher PbCu) with condition and reagents dosage were similar to the plant practice operation. The procedure and condition were set up at pH 8.5 using milk of lime as much as 3000 g/ton, NaCN 100 g/ton, $ZnSO_4$ 600 g/ton, SIBX 200 g/ton and MIBC 20 g/ton. Conditioning time of each reagent and flotation time of PbCu-rougher were set up 3 minutes consecutively. The result is shown in Table 1, in which the recovery of lead and copper at PbCu-rougher stage is found low at about 72 and 49% respectively, while the zinc component is highly distributed (57.8%). Consequently, the recovery of zinc at rougher sphalerite flotation is poor (36%). This phenomenon indicates that sphalerite does not effectively depressed yet by $ZnSO_4$. In tailing, however, lead and zinc still exist significantly that means selective flotation of lead and zinc was considerably fail. This laboratory flotation data approves the plant-practiced performance, which show that the ore has been difficult to be separated selectively by flotation, especially due to the component of zinc mineral distributes intensively into the lead-copper concentrate. Microscopic observation as shown in Figure 4 clearly indicates that numerous sphalerite of sizes less than 50µm is intensively entrapped into PbCu rougher concentrate. The difficulty of depressing sphalerite is likely induced high content of zinc (9.9%) in the ore, besides considerable presentation of cuprous-cyanide complexes and soluble lead ions, which might activate sphalerite (El-Shall et al. 2000; Seke and Pistorius, 2005). Further more, Seke and Pistorius (2005) who tested flotation selectivity between galena and sphalerite, found that the use of NaCN alone was not effective to depress sphalerite, however, the use of both NaCN and $ZnSO_4$ showed effective depression of sphalerite. The present flotation tests also have used both NaCN and $ZnSO_4$, therefore, it needs further flotation tests to solve the problem by increasing either NaCN or $ZnSO_4$ consumption as well as flotation rate.

tion as fix as required to provide high grade and recovery of lead-copper concentrate, in contrast to retain low grade and distribution of zinc component.

At the beginning of the flotation, lead mineral has the fastest floatability followed by copper minerals. After 2 minutes, the recovery rate of lead mineral gets slower and tends to form asymptotic line. The maximum recovery of lead mineral is approximately 80% after 5 minutes flotation time. On the other hand, the flotation rate of copper mineral is

Table 1. Laboratory flotation test result with condition similar to the plant practice

NaCN = 100 g/t, $ZnSO_4$ = 600 g/t, pH 8.5 (lime), SIBX=100 g/t, MIBC = 20 g/t, conditioning time 5 minutes, flotation time 5 minutes.

| Product | Wt (g) | Wt (%) | Grade(%) | | | Recovery (%) | | |
|---|---|---|---|---|---|---|---|---|
| | | | Pb | Cu | Zn | Pb | Cu | Zn |
| CPbCu-Ro | 99 | 19.8 | 23.6 | 3.7 | 30.1 | 72.18 | 49.63 | 57.87 |
| CZn-Ro | 85 | 17 | 4.61 | 3.63 | 21.99 | 12.11 | 41.81 | 36.30 |
| TZn-Ro | 316 | 63.2 | 1.61 | 0.2 | 0.95 | 15.72 | 8.56 | 5.83 |
| Feed calc. | 500 | 100 | 6.47 | 1.48 | 10.30 | 100 | 100 | 100 |

Note: C = concentrate, T = tailing, Ro = rougher

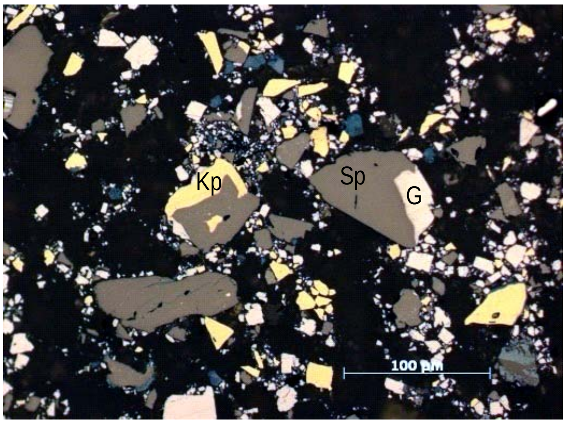


Figure 4. Entrapment of sphalerites, sizes less than 50µm, into PbCu rougher concentrate

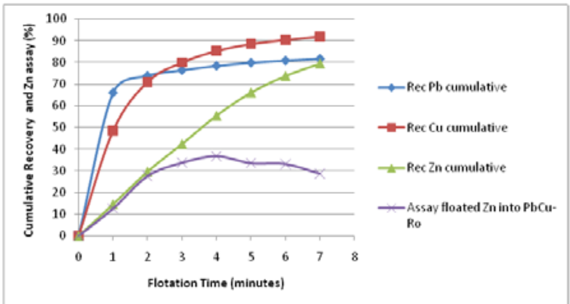


Figure 5. Flotation time vs recovery of Pb,Cu,Zn minerals in rougher PbCu flotation

## Flotation Rate of Minerals at PbCu Rougher Stage

The flotation rate of lead, copper and zinc minerals at rougher stage of PbCu is presented in Figure 5. Test condition and reagents were similar to the flotation as shown in Table 1. The obtained data would inform the time limit for rougher PbCu flota-

suddenly fast over the flotation rate of lead mineral after 2 minutes flotation time with maximum recovery is approximately 90% after 5 minutes flotation time. However, the recovery of unwanted-floated zinc into PbCu rougher flotation is significantly increased as the flotation time increases. Similarly, the grade of floated zinc increases up to approximately 35%Zn within 4 minutes flotation time. This Pb-Zn flotation time characteristic compared to Pb-Zn flotation time data found by Seke and Pistorius (2005) is relatively resemble. Therefore, the flotation rate of the present ore indicates that PbCu component could be floated to reach 80 to 90% recovery of PbCu at time limit of 5 minutes. However, unwanted Zn component cannot be avoided from the present flotation, hence; the use of zinc depressant agent does the most important case to be concerned.

### Selective Differential Flotation Test

Based on data obtained from flotation tests (Table 1 and Figure 5), it was further examined the selective differential flotation to perform the capability of grade and recovery of respective Pb, Cu, Zn sulphide minerals. The condition of the process was set up at pH 8.5 using milk of lime and NaCN 200 g/ton, $ZnSO_4$ 600 g/ton, SIBX 100 g/ton, MIBC 20 g/ton with conditioning time 5 minutes and flotation time 5 minutes. The test was held from rougher stage to re-cleaner one for lead-copper and zinc respectively. The result is shown in Table 2, which shows that recovery of Pb and Cu at flotation of PbCu rougher stage is 87 and 67% respectively, and grade of 21.6%Pb and 3.6%Cu. When the flotation was continued to cleaner and re-cleaner stages, the Pb and Cu recoveries become 68.7 and 38% respectively, performing grade of 52.3%Pb, 6.3%Cu and 9.4%Zn. The characteristic of zinc presented in the PbCu re-cleaner concentrate is observed as reveal in Figure 6, which indicates that few sphalerite of sizes less than 50µm is still entrapped (Figure 6A). This phenom-

Table 2. Selective differential flotation tests of Pb,Cu,Zn sulphide minerals

| Product | Wt (g) | Wt (%) | Grade(%) | | | Recovery (%) | | |
|---|---|---|---|---|---|---|---|---|
| | | | Pb | Cu | Zn | Pb | Cu | Zn |
| CPbCu-RCl | 37 | 7.81 | 52.29 | 6.3 | 9.38 | 68.70 | 37.95 | 8.78 |
| TPbCu-RCl | 10 | 2.11 | 14.73 | 5.18 | 26.98 | | | |
| CPbCu-Cl | 47 | 9.92 | 44.30 | 6.06 | 13.12 | 73.93 | 46.38 | 15.60 |
| TPbCu-Cl | 67 | 14.14 | 5.76 | 1.89 | 32.70 | | | |
| CPbCu-Ro | 114 | 24.05 | 21.65 | 3.61 | 24.63 | 87.63 | 66.99 | 71.02 |
| TPbCu-Ro | 360 | 75.95 | 0.97 | 0.56 | 3.18 | 12.37 | 33.01 | 28.98 |
| CZn-RCl | 20 | 4.22 | 1.48 | 3.21 | 36.62 | 1.05 | 10.45 | 18.53 |
| TZn-RCl | 38 | 8.02 | 3.27 | 1.05 | 3.92 | | | |
| CZn-Cl | 58 | 12.24 | 2.65 | 1.79 | 15.20 | 5.46 | 16.95 | 22.29 |
| TZn-Cl | 35 | 7.38 | 3.47 | 0.45 | 1.72 | | | |
| CZn-Ro | 93 | 19.62 | 2.96 | 1.29 | 10.12 | 9.78 | 19.51 | 23.82 |
| TZn-Ro | 267 | 56.33 | 0.27 | 0.31 | 0.76 | | | |
| CZnScav | 22 | 4.64 | 2.09 | 2.21 | 1.71 | 1.63 | 7.91 | 0.95 |
| TZnScav | 245 | 51.69 | 0.11 | 0.14 | 0.68 | | | |
| Feed | 474 | 100 | 5.94 | 1.30 | 8.34 | 100 | 100 | 100 |

Note: C = concentrate, T = tailing, Ro = rougher, Cl = cleaner and RCl = re-cleaner

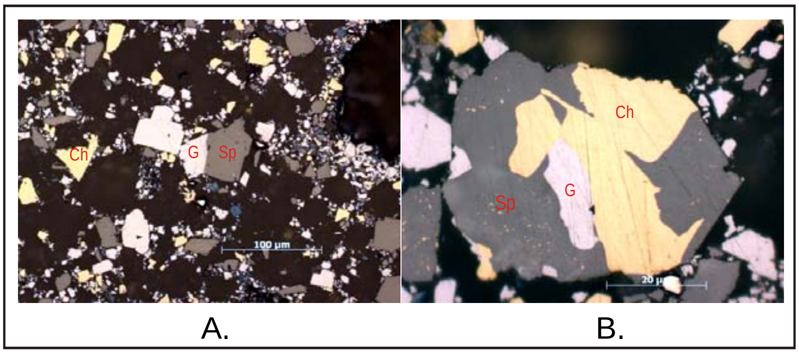


Figure 6. Microphotographs of the particles of PbCu re-cleaner concentrate (A). Particle of galena (G) associates with sphalerite (Sp) (B). A particle (50 µm) contains intricate association between sphalerite, galena and chalcopyrite (Ch)

enon is likely due to intricate association among galena, sphalerite and chalcopyrite in a particle size of approximately 50µm (Figure 6B).

However, performance of selective differential flotation as shown in Table 2 presents that trends to perform high grade and recovery of lead are possible. Figure 7 exhibits that froth phase of PbCu flotation performs grey-bluish color with few brownish, while froth phase of Zn flotation performs clear brownish, which indicates the true color of galena and sphalerite respectively. Hence, further study must be concerned with depressing Zn component at rougher stage of PbCu flotation by focusing on such kind of pH regulator, depressing agents of zinc component using Na-bisulphite or $ZnSO_4$.

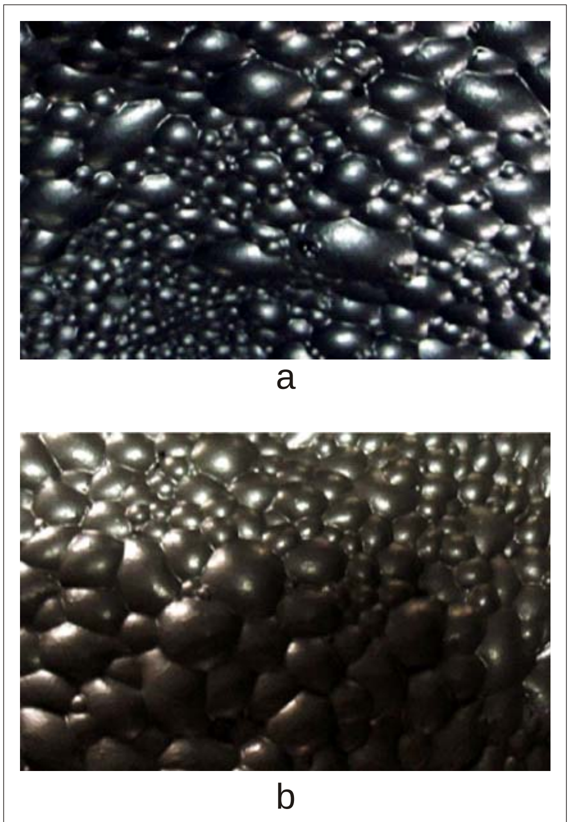

Figure 7. Froth phase photographs taken at the flotation tests (a) Re-cleaner PbCu froth phase, (b) Re-cleaner Zn froth phase

## pH-Regulator

Condition of the flotation test was set up similar to the preceding flotation test. The use of milk of lime compares to soda ash as pH-regulators is shown in Figure 8, in which the consumption of lime and soda ash is relatively the same amount of 2500 g/ton for adjusting alkalinity (pH 8.5) of the slurry. The PbCu rougher flotation is shown to significantly improve the Pb recovery (86.5%) compared to the other Pb recovery (76%). The former is supposed to be caused by soda ash agent while the later is due to milk of lime. Both of the pH regulators that are in line with de-zincing requirement could not provide the desired low Zn grade. Lime is cheaper than soda ash; however, lime could not improve Pb recovery. It may be due to lime has function as sphalerite activator (Cyanamid, 1989:91), causing to obstruct the floatability of PbCu minerals. Therefore, it is likely that lime has a negative effect on flotation of PbCu sulphides; on the other hand, soda ash is considerably useful pH regulator to improve the recovery of PbCu sulphides flotation. The absence of soda ash results in an decrease of Pb recovery.

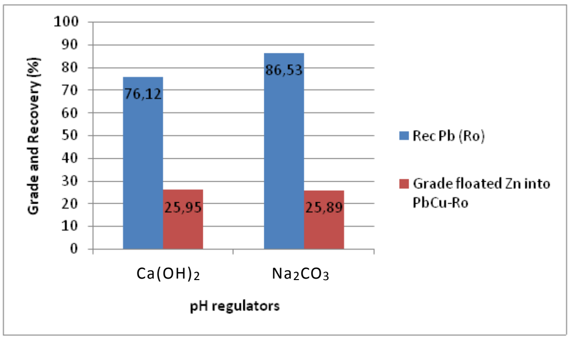


Figure 8. Comparison of lime and soda ash as pH regulators

## De-Zincing of PbCu Rougher with and without NaCN

Dosage of NaCN was increased up to 200 g/ton combined with $ZnSO_4$ as much as 600 g/ton, where cyanide is known as depressant of sulphides especially to depress pyrite. The result is presented in Table 3, while the capability of de-zincing process could be depicted in Figure 9. When a dosage of NaCN is increased doubled from 100 up to 200 g/t, the assay of Zn in rougher PbCu decreases from 30%Zn down to 24.6%Zn, while the recovery of Pb increases from 72% up to 87.6%. The Zn assay may tend to progressively decrease if the dosage of NaCN is increased. Therefore, NaCN in combination with $ZnSO_4$ is useful reagents that significantly affect the de-zincing process. Cyanide may react with zinc sulphate to form zinc cyanide which is relatively insoluble, and presipitates on the sphalerite surface, rendering it hydrophilic (Wills and Napier-Munn, 2007). The present result is in agreement with the work of Seke and Pistorius (2005). NaCN, however, is a toxic chemical reagent, eventhough many mining industries in the world are using NaCN as flotation reagent that is considerably effective to depress pyrite as well as sphalerite; even many gold processing in the world currently are still using cyanide as common gold leach solution. However, "Sase" mine in Srebrenica-Yugoslavia has eliminated cyanide (Magdalinovic et al. 2004). Therefore, in accordance with environmental friendly regulation, the use of NaCN must be minimized and that is importantly avoided up to the further flotation tests.

Table 3. PbCu rougher flotation test with increasing NaCN dosage

NaCN = 200 g/t, $ZnSO_4$ = 600 g/t, pH 8.5 (lime), SIBX=100 g/t, MIBC = 20 g/t, conditioning time 5 minutes, flotation time 5 minutes.

| Product | Wt (g) | Wt (%) | Grade(%) | | | Recovery (%) | | |
|---|---|---|---|---|---|---|---|---|
| | | | Pb | Cu | Zn | Pb | Cu | Zn |
| CPbCu-Ro | 114 | 24.05 | 21.65 | 3.61 | 24.63 | 87.63 | 66.99 | 71.02 |
| TPbCu-Ro | 360 | 75.95 | 0.97 | 0.56 | 3.18 | 12.37 | 33.01 | 28.98 |
| Feed | 474 | 100 | 5.94 | 1.30 | 8.34 | 100 | 100 | 100 |

Note: C = concentrate, T = tailing, Ro = rougher

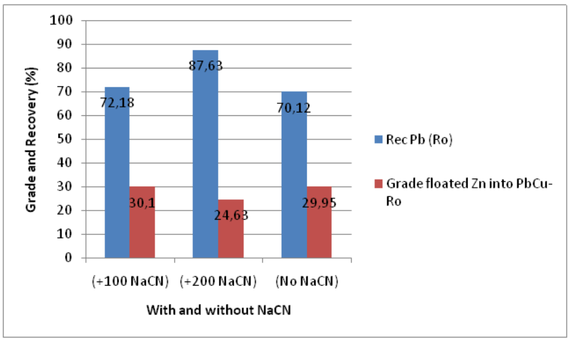


Figure 9. Comparison of Pb rougher recovery and de-zincing with/without NaCN

Flotation of PbCu rougher without cyanide was also tested as shown in Figure 9, in which the recovery of Pb decreases down to nearly 70%, while the floated Zn is relatively high that is almost similar to the flotation with NaCN 100 g/ton. Therefore, it is clear that the use of NaCN is important to improve the recovery of PbCu or to decrease the floated Zn. However, further tests are looking for alternative reagent of zinc depressant that NaCN may be substituted by sodium bisulphite ($Na_2SO_3$) as stated by Khmeleva (2005). However, SIKBZ2000 reagent as proposed by Magdalinovic et al. (2004) cannot yet be applied in the present tests because the typical reagent has not been served in the laboratory.

## De-Zincing of PbCu Rougher with Na-bisulphite

Na-bisulphite was reported as an effective depressant for the flotation of cuprous-activated sphalerite. According to Khmeleva et al. (2005), the depression mechanism is that the sulphite ions may react with the surface of copper-activated sphalerite and subsequently decompose hydrophobic sulphide into solution, which is then oxidized to sulphate. In the same time, zinc hydroxide is formed at sphalerite surface rendering hydrophilic. Hence, Na-bisulphite as much as 1000 g/ton was added into rougher flotation of PbCu, which is used as NaCN substitution. The result is shown in Figure 10, where Pb recovery and Zn grade that obtain by using Na-bisulphite 1000 g/ton compared to Pb recovery and Zn grade using NaCN 200 g/ton are relatively resemble. The difference is revealed by dosages, in which the process requires Na-bisulphite 1000 g/ton, while the same process requires NaCN 200 g/ton. When the addition of Na-bisulphite is increased up to 2000 g/ton, the grade of floated Zn in PbCu rougher could successfully be reduced to nearly 10%Zn. However, the recovery of Pb in PbCu rougher is consequently reduced as shown in Figure 11. This phenomenon generally informs that Na-bisulphite could replace

NaCN significantly, but the excess dosage of Na-bisulphite would tend to suppress galena. Therefore, economic consideration might be taken in serious attention whether using Na-bisulphite rather than NaCN.

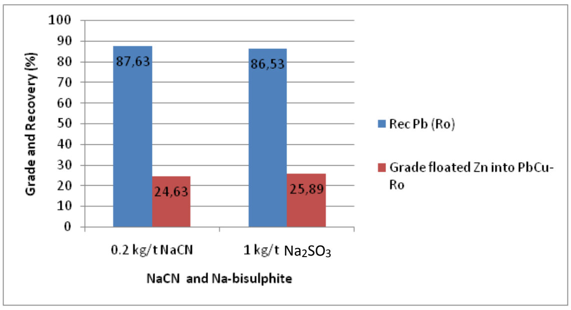


Figure 10. Comparison of flotation with NaCN and Na-bisulphite

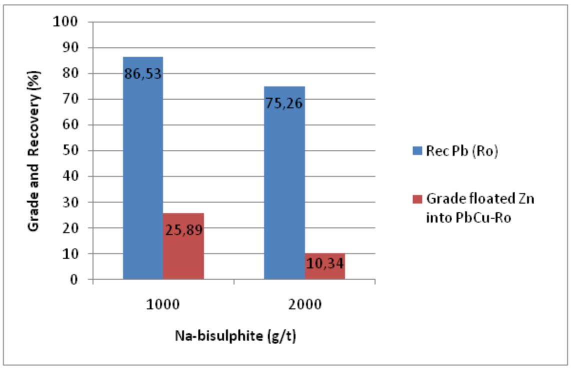


Figure 11. Effect of Na-bisulphite dosage on grade and recovery

### De-Zincing of PbCu Rougher with Zinc Sulphate

Flotation tests of rougher PbCu performed the difficulty in depressing Zn, even though the flotation has used high dosage of $ZnSO_4$ (1000 g/ton) and Na-bisulphite (2000 g/ton) but the entrapped zinc mineral is still high as much as 10 - 30%Zn. $ZnSO_4$ is a kind of conventional reagent that is acted as pyrite or sphalerite depressant. However, Bulatovic (2007:163) stated that $ZnSO_4$ alone does not depress sphalerite. Depressing action occurs only in the presence of hydroxyl ions (OH") in alkaline condition, where sphalerite surface changes into zinc hydroxide leading to hydrophilic form and preventing adsorption of xanthate. The present flotation tests of de-zincing of PbCu rougher, however, are concerned with increasing the dosage of $ZnSO_4$ to understand the capability of depressing Zn component in the flotation of rougher PbCu at pH 8.5. The de-zincing performance of rougher PbCu flotation tests can be seen in Figure 12, which clearly reveals that the higher is the $ZnSO_4$ dosage, the lower is the grade of unwanted-floated Zn in rougher PbCu. Starting from $ZnSO_4$ dosage of 600 g/ton results the grade of floated Zn around 30%Zn, when the dosage of $ZnSO_4$ is increased up to 3000 g/ton, the grade of floated Zn in rougher PbCu is successfully reduced to 8.6%Zn (Table 4). Even, based on prediction in log-log scale as depicted in Figure 13, the grade of floated Zn in rougher PbCu might be reached less than 5%Zn if it were used approximately 5500 g/ton of $ZnSO_4$. Therefore, the phenomenon of de-zincing is considerably significant by increasing dosage of $ZnSO_4$ in combination with Na-bisulphite. However, according to Figure 12, tendency of saturated $ZnSO_4$ are found when its dosage is more than 3000 g/ton. It may reduce the effectiveness of Zn depressing action. Consequently, the recovery of Pb would significantly decrease. In this case, economic consideration may be pointed out because of high consumption and/or price of the chemical reagent. It is important to recommend that the optimum consumption of $ZnSO_4$ is approximately 1500 g/ton with capability in depressing action around 15%Zn and to maximize the recovery of Pb approximately 78%. Sphalerite might be depressed until reaching 5%Zn at cleaner and re-cleaner stages. Besides such depressant, frother also affects the recovery of PbCu, Hadler et al. (2005) stated that 20% of frother is lost in the presence of SIBX and after 2 minutes, 63% of SIBX had adsorbed to particles. This means that conditioning time as well as increasing concentration of SIBX would yield a higher concentrate grade. However, varying SIBX and conditioning time were not employed yet in the present work.

In fact, the de-zincing action of the present flotation tests exhibit that lesser zinc component is still presence at PbCu concentrate. It is most likely due to sphalerite still interlock with galena and chalcopyrite in fine sizes of about 50 µm. On the other hand, fine liberated sphalerite with sizes less than 20 µm is also shown to enter the PbCu concentrate, which is may be due to the presence of cuprous ions activated sphalerite. Association of sphalerite, galena and chalcopyrite particles as well as free sphalerite are apparently confirmed by the photomicrographs of PbCu re-cleaner flotation concentrate as shown in Figure 6. Hence, size classification combined with regrinding of the PbCu rougher flotation concentrate may be necessary. Liberated fine sphalerite that entered to PbCu concentrate may require high dosage of the depressant of sphalerite. The present study, in general, performs proper tendency of de-zincing requirement as well as improvement of lead-copper recovery by increasing depressing agent of sphalerite.

Table 4. De-zincing of PbCu rougher with high dosage of Na-bisulphite and Zn-sulphate

$Na_2SO_3$ = 2000 g/t, $ZnSO_4$ = 3000 g/t, pH 8.5 (soda ash), SIBX=100 g/t, MIBC = 20 g/t, conditioning time 5 minutes, flotation time 5 minutes.

| Product | Wt (g) | Wt (%) | Grade(%) | | | Recovery (%) | | |
|---|---|---|---|---|---|---|---|---|
| | | | Pb | Cu | Zn | Pb | Cu | Zn |
| CPb/Cu-Ro | 62.62 | 12.19 | 34.83 | 7.15 | 8.61 | 69.04 | 59.74 | 10.33 |
| CPb/Cu-Sc | 14 | 2.73 | 11.19 | 6.88 | 17.76 | 4.96 | 12.85 | 4.76 |
| TPb/Cu-Sc | 437 | 85.08 | 1.88 | 0.47 | 10.14 | 26.00 | 27.41 | 84.91 |
| Feed | 513.62 | 100 | 6.15 | 1.46 | 10.16 | 100 | 100 | 100 |

Note: C = concentrate, T = tailing, Ro = rougher, Sc = scavenger

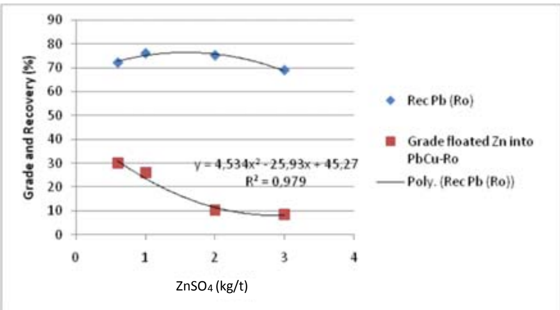


Figure 12. Effect of $ZnSO_4$ on de-zincing of PbCu rougher flotation

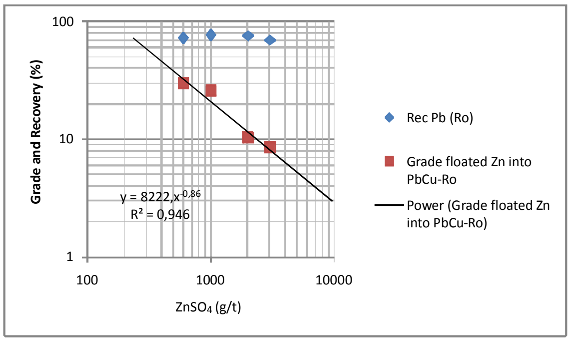


Figure 13. Effect of $ZnSO_4$ on de-zincing of PbCu rougher flotation in log-log scale

## CONCLUSIONS AND SUGGESTION

Initially, lead-copper rougher flotation was entrained by zinc minerals as much as 30%Zn. Experimental flotation results of de-zincing have shown that the use of Na-carbonate as pH regulator is better than that of milk of lime. Combination of Zn-sulphate and cyanide is a useful depressant for zinc mineral. Compared to Na-bisulphite, the use of Na-cyanide is resembles in improving the lead-copper recovery as well as de-zincing, however, the consumption of Na-bisulphite is four folds higher than that of Na-cyanide. Zn-sulphate combines with Na-bisulphite is considerably more effective in the de-zincing process. However, the consumption of Zn-sulphate is considerably high to reduce entrapped liberated fine zinc mineral at PbCu concentrate. The difficulty in depressing zinc mineral at PbCu flotation is apparently due to the intricate association of sphalerite-chalcopyrite-galena and might be due to the presence of activated sphalerite. However, the present flotation study generally performs tendency of de-zincing requirement for PbCu rougher flotation that contains about 8% Zn and PbCu recovery is 70%.

It is suggested that the flotation of PbCu rougher could be applied at condition of such reagents of Zn-sulphate 1500 g/ton, Na-bisulphite 1000 g/ton, pH 8.5 using Na-carbonate around 2000 g/ton, SIBX 100 g/ton and MIBC 20 g/ton, flotation time is 5 minutes. Rougher PbCu concentrate should be treated with size classification prior to regrinding continued with cleaner and re-cleaner flotation stage.

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