The effect of mode of occurrence of galena and sphalerite on the selective flotation of ore samples from the Rosh Pinah Mine

by M.D. Seke* and P.C. Pistorius*

Synopsis

The flotation response of a Pb-Zn sulphide ore from the Rosh Pinah Mine (Namibia) was studied in the presence of inorganic depressants such as sodium cyanide and zinc sulphate. Poor flotation selectivity was observed in the rougher concentrate of the galena circuit despite the use of a large amount of sodium cyanide. Batch flotation tests have shown that the use of cyanide alone is not efficient for the depression of sphalerite from the Rosh Pinah ore when milling is carried out according to the current plant particle size distribution. The use of both cyanide and zinc sulphate improved the selectivity between galena and sphalerite much better than cyanide alone.

Flotation selectivity is limited by the mineralogical texture of the Rosh Pinah ore sample. Microscopic analysis has shown that the presence of sphalerite in the galena concentrate is also due to poor liberation between galena and sphalerite, especially in the middlings. Hence, selectivity could be improved by regrinding the rougher concentrate prior to the cleaning stage.

Keywords: sulphide ores, flotation, flotation depressant, mineralogy.

Introduction

The Rosh Pinah zinc-lead sulphide deposit occurs in the southwestern part of Namibia, close to the Orange (Gariep) River. The Rosh Pinah Mine treats a composite of copper-lead-zinc sulphide ores from various sites. Pyrite is the main sulphide gangue mineral in the Rosh Pinah composite sample. Traces of chalcopyrite, gold and silver are found in the ore sample (Figure 1).

At the Rosh Pinah plant, the composite feed is processed by selective flotation, in which galena is floated first with sodium propyl xanthate (SNPX) as collector, while sphalerite and pyrite are depressed with cyanide. The sphalerite is floated further with xanthate in the zinc flotation circuit after activation with copper sulphate. Selectivity against sphalerite poses a challenge in the lead flotation circuit at the Rosh Pinah Mine, where cyanide dosages as high as 150–180 g/t are being used to suppress the flotation of sphalerite and pyrite at the concentrator.

Apart from the significant contribution to the loss of precious metals such as silver and gold by forming soluble metal complexes, the excessive use of cyanide is a cause for concern on environmental grounds. Furthermore, this necessitates the use of more copper sulphate to activate sphalerite for its subsequent flotation in the zinc circuit. Despite the high dosage of cyanide used in the lead flotation circuit, it is estimated that approximately 1 250 tons of zinc are lost every year in the lead concentrate.[1]

In most plants, sodium cyanide is usually used in conjunction with zinc sulphate for the effective depression of sphalerite from Cu-Pb-Zn sulphide ore at alkaline pH values. Examples are presented in Table I.[2] The metallurgical results of the Rosh Pinah plant are also given for comparison purposes.

As seen in Table I, the dosage ratio of $ZnSO_4$ to NaCN varied from approximately 2.5 to 4. The high dosage of depressant used at Société Algérienne du Zinc was probably due to the high content of zinc (24.3%) in the feed material as compared with 6–9% Zn at the Rosh Pinah plant. The mineralogy of the ore treated at the Bunker Hill and Société Algerienne du Zinc concentrators is similar to the Rosh Pinah ore, despite the differences in their respective chemical compositions and metallurgical results (Table I).

A study on the deportment of sphalerite in the lead flotation circuit was carried out in the work reported here for a better understanding of the high dosage of cyanide required for the depression of sphalerite at the Rosh Pinah plant; the aim of the work was to test whether the mineralogical texture of the Rosh Pinah concentrator feed contributes to poor flotation selectivity between galena and sphalerite even in the presence of cyanide.

* *Department of Materials Science and Metallurgical Engineering, University of Pretoria, Pretoria, South Africa.*

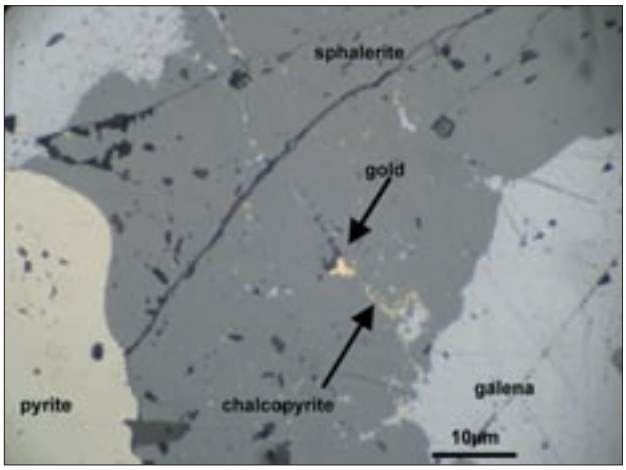


Figure 1—Photomicrograph showing sulphide minerals and gold in the Rosh Pinah ore sample

Table I

Selective flotation of complex lead-zinc sulphide minerals (Adapted from Tveter and McQuiston[2])

Mine/mineralogy	Depressants (g/t) Lead circuit	Product	Metallurgical results			
			Assays (%)		Distribution (%)	
			Pb	Zn	Pb	Zn
Bunker Hill Co., Kellogg, Idaho (Galena, sphalerite, pyrite, quartz)	NaCN: 46 $ZnSO_4$: 115	Mill Feed	7.1	2.5	100	100
		Pb Conc.	66.0	5.9	96.7	23.9
		Zn Conc.	1.8	54.1	0.8	65.2
		Tails	0.2	0.3	2.5	10.9
Société Algerienne du Zinc, Bou Beker, Morocco (Galena, saphalerite, pyrite, dolomite)	NaCN: 130 $ZnSO_4$: 511	Mill Feed	3.55	24.3	100	100
		Pb Conc.	73.1	2.98	93	1
		Zn Conc.	0.39	62.4	4	98
		Tails	0.21	0.55	3	1
Rosh Pinah Mine (Galena, sphalerite, pyrite, chalcopyrite, dolomite, quartz)	NaCN: 150–180	Mill Feed	1–3	6–9	100	100
		Pb Conc.	55–60	5–7	70–75	2–3
		Zn Conc.	1–2	52–55		80–85

Experimental

Materials, reagents and solutions

The lead-zinc ore sample (-9 mm) used in this study was obtained from the crushing plant at the Rosh Pinah Mine in Namibia. The sample was removed from the actual feed to the milling circuit. The sample was screened at 1.7 mm and the oversize fraction crushed to -1.7 mm. A sub-sample was removed for head assays. The remainder of the sample was used for the flotation test work. The chemical composition of the ore was determined using a sequential XRF spectrometer ARL 9400-241XP+ the results of which are shown in Table II.

Sodium propyl xanthate (SNPX) and Senfroth 9325 (polypropylene glycol) from Senmin (South Africa) were used as collector and frother, respectively. The xanthate was purified by dissolution in acetone and reprecipitation with petroleum ether as proposed by Rao, S.R., Xanthates and related compounds, 1971, Marcel Dekker. Xanthate solutions were prepared daily. Sodium cyanide (NaCN) from Saarchem (South Africa) and zinc sulphate ($ZnSO_4.7H_2O$) from BDH Laboratory Supplies (England) were used for the depression of sphalerite. Reagents were made up as 1% (w/w) stock solutions with distilled water, with the exception of Senfroth 9325, which was used neat. The cyanide solution was made up at pH 10.5 using NaOH to prevent the formation of HCN (this was done for the sake of laboratory safety; the same type of stock solution is not used on the plant itself). As noted below, flotation tests were performed at the natural pH of the pulp.

Flotation

A Denver D12 flotation machine, a 3 dm^3 flotation cell and Pretoria tap water were used for the batch flotation tests. All flotation experiments were carried out at about 33% (w/w) solids. The impeller speed was set at 1250 rpm and the air

Table II

The average chemical analysis of the lead-zinc ore sample used in this study (weight %)

Pb	Zn	Cu	Fe	S	CaO	MgO	Al_2O_3	SiO_2
1.9	7.0	0.12	3.7	3.9	18.7	8.2	4.1	49.2

flow rate was 6 dm^3 min^{-1}. Although no attempt was made to control the pulp potential and dissolved oxygen, they were monitored throughout the experiments.

Batches of 1 kg batch ore were milled at 67% solids (w/w), in an unlined laboratory mild steel mill (Ø 200 x 250 mm) with approximately 11 kg of mild steel rods (the plant grinding medium is also mild steel, in the form of balls). Grinding continued for 8 minutes to achieve the target grind of 80% passing 100 micron. The mill was not vented during the grinding stage. The particle size distributions of the ground products were determined using a Malvern Mastersizer 2000 instrument. After milling for the required time, the slurry was transferred into the flotation cell and the pulp was diluted to 33% using laboratory tap water. The collector (50 g/t SNPX) was added and the pulp was conditioned for 3 minutes after which the frother (60 g/t Senfroth 9325) was added and conditioned for a further 1 minute. For the depression of sphalerite with cyanide and zinc sulphate, the depressant was added simultaneously with the collector. After starting the air flow, the froth was removed by hand scraping every 15 seconds. Incremental rougher concentrates were collected after 1, 2, 4 and 8 minutes. The volume of the pulp in the flotation cell was kept constant by additions of tap water using a pulp level control device. Flotation tests (in duplicate unless stated otherwise) were carried out at the natural pH (8.5 ± 0.2) of the pulp and room temperature (22 ± 2°C).

Since the objective of this study was to understand the flotation selectivity of galena and sphalerite in the lead rougher flotation circuit, the test work programme did not include cleaner and locked cycle tests.

Scanning electron microscopy (SEM)

A JEOL JSM-6300 scanning electron microscope with an attached Noran EDS was used for image analysis. Backscattered electron images were useful to distinguish the differences in mineral composition. The acceleration voltage was 30 kV. The samples used for SEM examination had previously been prepared for XRF analysis.

Results and discussions

Effect of sodium cyanide on the flotation response of the Rosh Pinah composite

The recovery and grade of sphalerite in the flotation concentrate, at various dosages of sodium cyanide, are presented in Figures 2 and 3. The recovery of sphalerite decreased from 37% to 32% with the addition of 50 g/t NaCN, and to 28% with 100 g/t NaCN. There was only a slight decrease of approximately 1%, which is within experimental error, in the recovery of sphalerite upon increasing the amount of cyanide from 100 to 150 g/t.

The decrease in the recovery of sphalerite is likely to be due to the deactivation of copper-activated sphalerite by cyanide ions. Chalcopyrite, which is present in the Rosh Pinah composite, can release copper ions which activate sphalerite.

The effects of sodium cyanide on the recovery and grade of galena are shown in Figure 4. As seen in Figure 4, the recovery of galena was not adversely affected by the presence of sodium cyanide. This is as expected; if anything, cyanide might improve lead recovery: Prestidge *et al.*[5] and Ralston[6] proposed that cyanide depleted the galena surface of sulphur, forming CNS^-, leaving a residual lead-rich surface, which is more receptive to ethyl xanthate interaction.

Figure 5 shows the effect of sodium cyanide on the flotation selectivity between galena and sphalerite. As expected, the flotation selectivity was improved by the addition of sodium cyanide. As stated above, increasing the cyanide dosage above 100 g/t NaCN gave no further improvement in flotation selectivity.

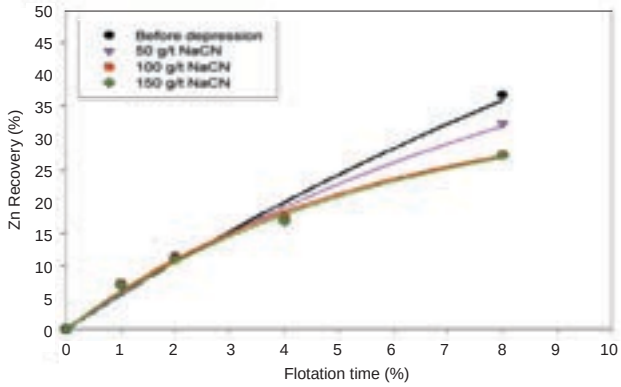


Figure 2—Flotation recovery of zinc from a galena-sphalerite composite from Rosh Pinah at various dosages of sodium cyanide, 50 g/t SNPX and pH 8.5

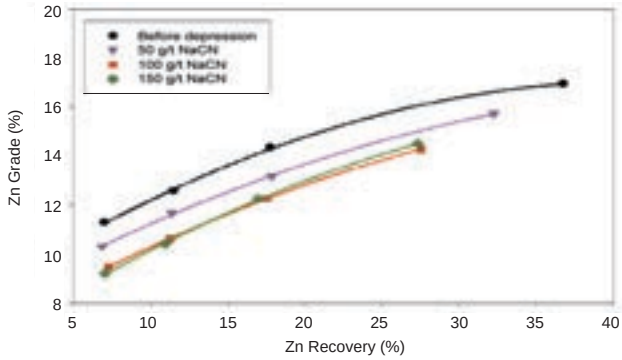


Figure 3—Recovery and grade of zinc from a galena-sphalerite composite from Rosh Pinah at various dosages of sodium cyanide, 50 g/t SNPX and pH 8.5

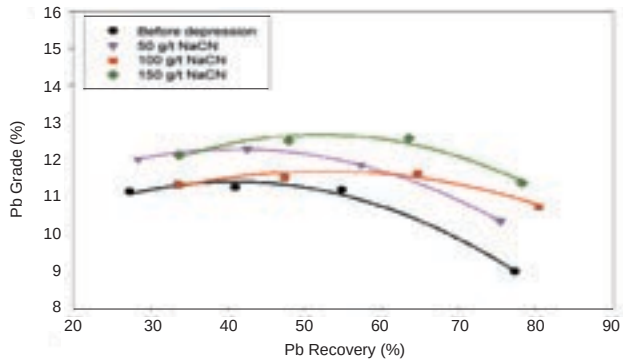


Figure 4—Recovery-grade of lead from a galena-sphalerite composite from Rosh Pinah at various dosages of sodium cyanide, 50 g/t SNPX and pH 8.5

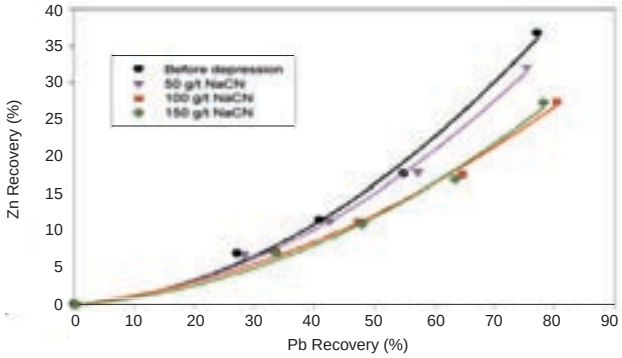


Figure 5—Recovery-grade of lead from a galena-sphalerite composite from Rosh Pinah at various dosages of sodium cyanide, 50 g/t SNPX and pH 8.5

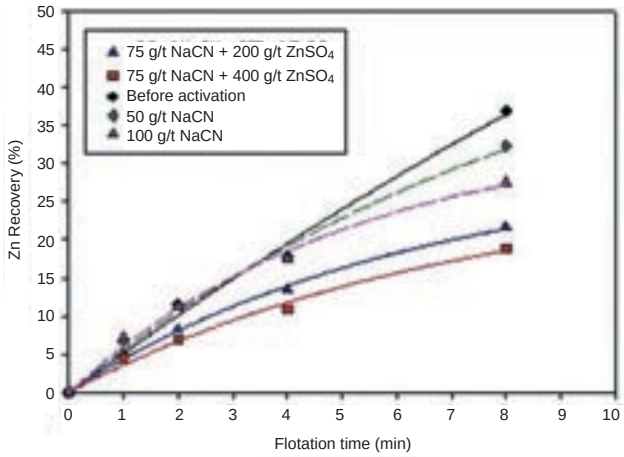


Figure 6—Flotation recovery of zinc from a Rosh Pinah composite sample at various dosages of sodium cyanide and zinc sulphate, 50 g/t SNPX and pH 8.5

Based on the chemical and mineralogical composition of the Rosh Pinah ore, it is possible that the sphalerite is activated by both copper and lead ions. It is not possible to depress the lead-activated sphalerite with cyanide ions, which is the proposed role of the second depressant, zinc sulphate. The combined effect of sodium cyanide and zinc sulphate on the flotation of sphalerite in the lead circuit is presented in the next section.

Effect of sodium cyanide and zinc sulphate on the flotation response of the Rosh Pinah composite

Flotation test work was conducted at the natural pH (8.5±0.1) of the ore in the presence of various concentrations of sodium cyanide and zinc sulphate, as explained in the previous section. Both sodium cyanide and zinc sulphate were added simultaneously with xanthate in the flotation cell. The cyanide dosage of 75 g/t was used based on the flotation results presented in Figure 5. In addition, the zinc sulphate dosages of 200 and 400 g/t were used to give $ZnSO_4$ to NaCN dosage ratios of approximately 3 and 5.

The recovery of sphalerite decreased from approximately 37% to 22% with the addition of 200 g/t zinc sulphate, decreasing further to 19% at 400 g/t $ZnSO_4$ (together with 75 g/t NaCN in both cases), as Figure 6 shows; this is better than the depression achieved with 100 or 150 g/t NaCN (see Figures 2 and 6). The final grade of zinc in the lead concentrate decreased from 17.0% to 11.8% with the addition of 200 g/t zinc sulphate, decreasing further to 10.9% for 400 g/t of zinc sulphate (together with 75 g/t NaCN in both cases), as shown by Figure 7.

The recovery of galena decreased slightly—from 77% to 73% with 200 g/t $ZnSO_4$, and 72% at 400 g/t $ZnSO_4$ (Figure 8). The slight decrease in the recovery of galena might be caused by the presence of hydrophilic zinc hydroxide on the surface of galena; zinc hydroxide is not expected to adsorb/precipitate selectively on galena or sphalerite. As shown in Figure 8, the grade of lead in the concentrate increased after the additions of depressant, due to the decreased sphalerite recovery in the lead concentrate; the grade of lead increased from 8.9% to 11.7% with addition of 200 g/t $ZnSO_4$ (and was the same for the higher addition of zinc sulphate).

Selectivity between galena and sphalerite improved with the addition of both cyanide and zinc sulphate (Figure 9).

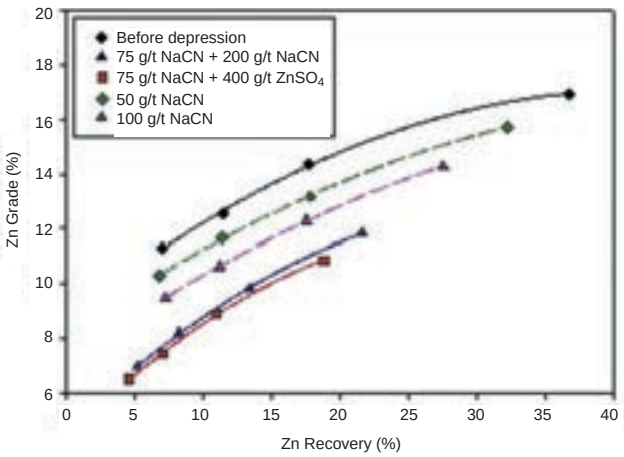


Figure 7—Recovery-grade relationship of zinc from a Rosh Pinah composite sample at various dosages of sodium cyanide and zinc sulphate, 50 g/t SNPX and pH 8.5

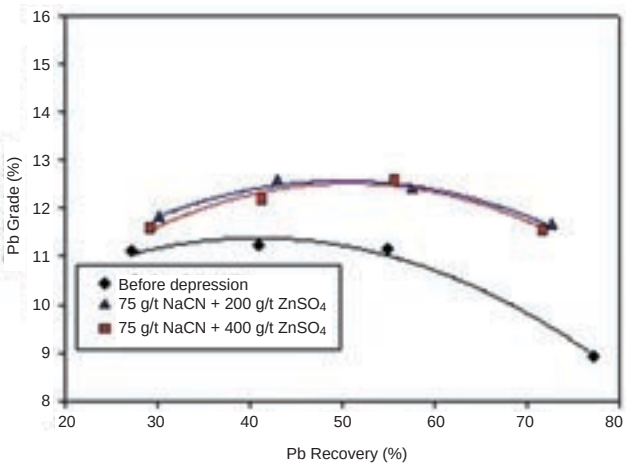


Figure 8—Recovery-grade of lead from a Rosh Pinah composite sample at various dosages of sodium cyanide and zinc sulphate, 50 g/t SNPX and pH 8.5

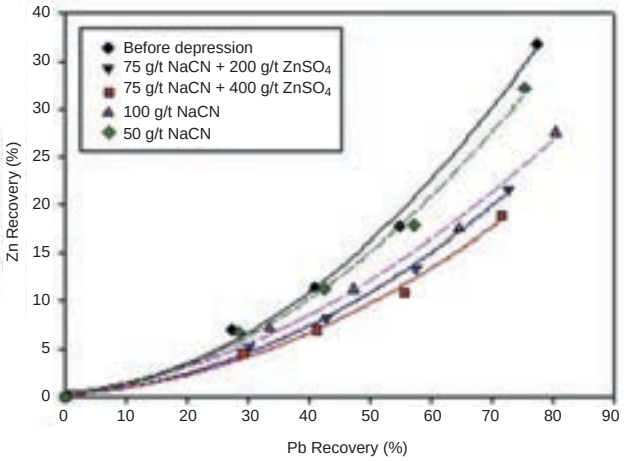


Figure 9—Lead and zinc recoveries from a Rosh Pinah composite sample at various dosages of sodium cyanide and zinc sulphate, 50 g/t SNPX and pH 8.5

The additional effect of zinc sulphate on the flotation selectivity can be related to the depression of lead-activated sphalerite.

The selectivity achieved with 100 g/t NaCN alone was similar to that achieved with the combination of 75g/t NaCN and 200 g/t $ZnSO_4$, but the recoveries of galena and sphalerite were lower when zinc sulphate was used in conjunction with sodium cyanide. Since the recovery of galena in the lead rougher concentrate has to be maximized in plant practice, it would be convenient to use 100 g/t NaCN for the depression of sphalerite in the roughing stage followed by the optimization of the depressant in the cleaning stage.

Deactivation with zinc sulphate and sodium cyanide

The solubility of zinc species as functions of pH in the presence of zinc sulphate and sodium cyanide is shown in Figure 10; the precipitation of zinc hydroxide can occur at the alkaline pH values used at the Rosh Pinah plant during the selective flotation of galena and sphalerite, if both zinc sulphate and sodium cyanide are used. However, sphalerite from the Rosh Pinah composite feed is primarily activated by copper ions present in the flotation pulp.[7] Thus cyanide is expected to react with the copper at the surface of sphalerite to form cuprous cyanide complexes. The cuprous cyanide complexes such as $Cu(CN)_3^{2-}$ and $Cu(CN)_2^-$ are predicted to be dominant cyanide species at alkaline pH values when sodium cyanide and zinc sulphate are used to depress sphalerite (Figure 11). Copper complex formation results in a lower concentration of $Zn(CN)_4^{2-}$ in solution.

The concentration of free cyanide in solution will decrease with decreasing pH (Figure 11). Thus it is also important to monitor the pH of the flotation pulp for an efficient consumption of sodium cyanide during the depression of sphalerite to avoid the loss of free cyanide at pH values lower than 8.

Based on the thermodynamic information presented in Figures 10 and 11, it appears possible to depress sphalerite when it has been activated by both copper and lead ions by using sodium cyanide and zinc sulphate at alkaline pH values. The most plausible mechanisms of depression would be the complexation of surface copper with free cyanide and the precipitation of hydrophilic zinc hydroxide on the surface of sphalerite.

Although the recoveries and grades of sphalerite were decreased with the use of both cyanide and zinc sulphate, separation between galena and sphalerite remains rather poor, and it is important to understand why depression of zinc in the galena concentrate is only partially successful. Mineralogical analysis was carried out to further understand the poor flotation selectivity between galena and sphalerite, which persists even in the presence of sodium cyanide.

Deportment of sphalerite through the flotation products

Deportment of sphalerite in the lead rougher concentrate

Qualitative mineralogy by image analysis on scanning electron microscopy (SEM) was performed on the flotation products after flotation of a composite ore from Rosh Pinah in the presence of 100 g/t NaCN and 50 g/t SNPX (Figure 12). These flotation results are similar to those presented in Figures 2–4.

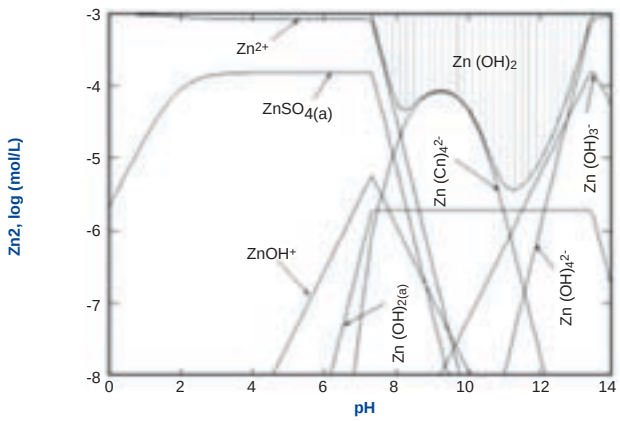


Figure 10—Speciation diagram for Zn(II) as a function of pH in the presence of 10^{-3}M NaCN and 10^{-3}M $ZnSO_4$ at 25°C. Stabcal software. NBS database Huang, H.H. 2003. Stabcal Software: Stability Calculation for Aqueous Systems. Metallurgical Engineering, Montana Tech (USA)

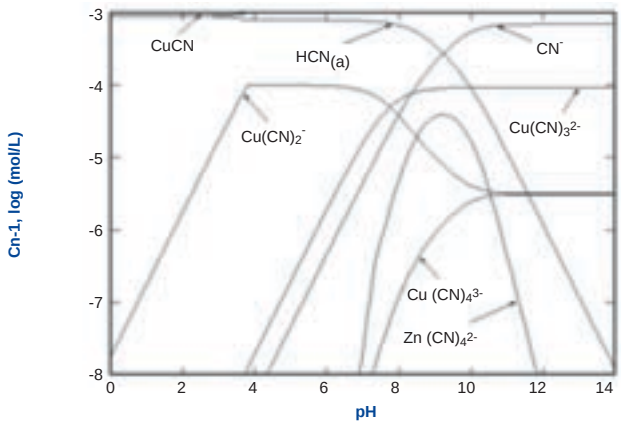


Figure 11—Speciation diagram for CN- as a function of pH in the presence of 10^{-3}M NaCN, 10^{-3}M $ZnSO_4$, and 10^{-4}M Cu(I) at 25°C. Stabcal software. NBS database Huang, H.H. 2003. Stabcal Software: Stability Calculation for Aqueous Systems. Metallurgical Engineering, Montana Tech (USA)

The flotation results shown in Figure 12 indicate that galena and pyrite were the fast floating minerals, while sphalerite was the slow floating mineral. Figure 12 also indicates that approximately 36% of pyrite, 34% of galena and 7.2% of sphalerite were recovered in the first minute of flotation. However, 16% of galena, 10% of sphalerite and 7% of pyrite were recovered in the last incremental concentrate (4–8 minutes).

The mineralogical textures of the concentrates obtained after one and 8 minutes of flotation are shown in Figures 13 and 14. As seen in Figures 13 and 14, the fractional amounts of galena and pyrite recovered in the concentrate decreased with the flotation time, while that of sphalerite and gangue increased. It was clear that the concentrate recovered in the first minute of flotation contained mainly liberated galena and pyrite. Figure 13 shows that liberated particles of galena were usually fine grained (about 25 micron in diameter), while pyrite particles seemed to be much coarser. The mineralogical texture of the concentrate recovered after 8 minutes of flotation showed that the recovery and grade of gangue minerals (mainly silicate and dolomite) increased in the last concentrate, compared with the concentrate of the first minute. Figure 13 also shows that most of the slow-floating materials were large sphalerite particles (+50 µm). Their presence in the lead concentrate would be detrimental to flotation selectivity.

A striking feature of the texture of the concentrates was the large quantity of binary locked galena and sphalerite (Figure 15).

The occurrence of galena locked and/or attached to sphalerite increased with increasing particle size, especially for particles larger than 50 micron in diameter. Thus, the poorly liberated sphalerite particles from the middlings would contribute to the problem of zinc deportment into the lead concentrate at the Rosh Pinah Mine. Hence, increasing the dosage of depressant would not solve the problem without affecting the recovery of galena: with severe depression of sphalerite, galena particles, which are occluded in sphalerite, may also be lost in the rougher tailings, as is shown in Figure 16. In addition, the loss of galena in the rougher tailings can be increased due to the presence of slow floating particles when the retention time is not long enough to account for their flotation. As seen in Figure 17, the rougher tailings mostly contained liberated sphalerite and gangue, which are sent to the zinc flotation circuit. The sphalerite is then intentionally activated with copper sulphate followed by its flotation with xanthate at high pH values to depress the flotation of pyrite.

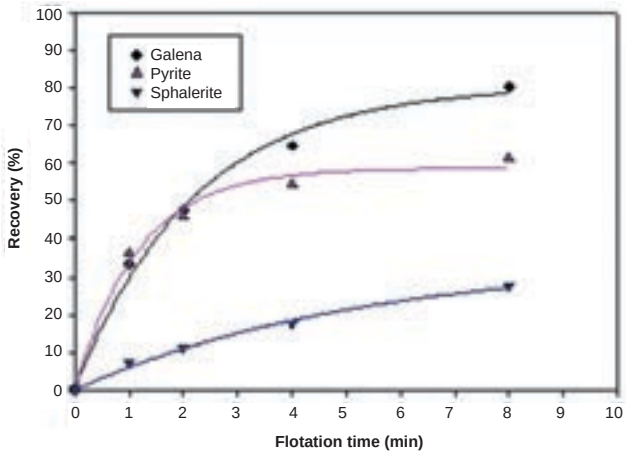


Figure 12—Recoveries of galena, pyrite and sphalerite after flotation of a composite from Rosh Pinah in the presence of 50 g/t SNPX and 100 g/t NaCN at pH 8.5

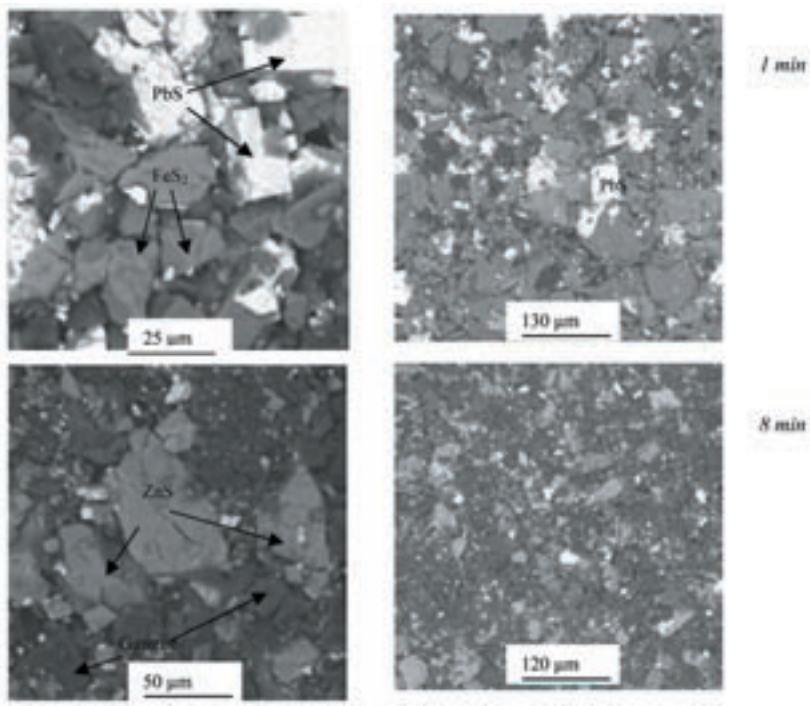


Figure 13—Backscattered electron images showing the general appearance of the rougher concentrates after 1 and 8 minutes. The flotation experiment was carried out in the presence of 100 g/t NaCN and 50 g/t SNPX

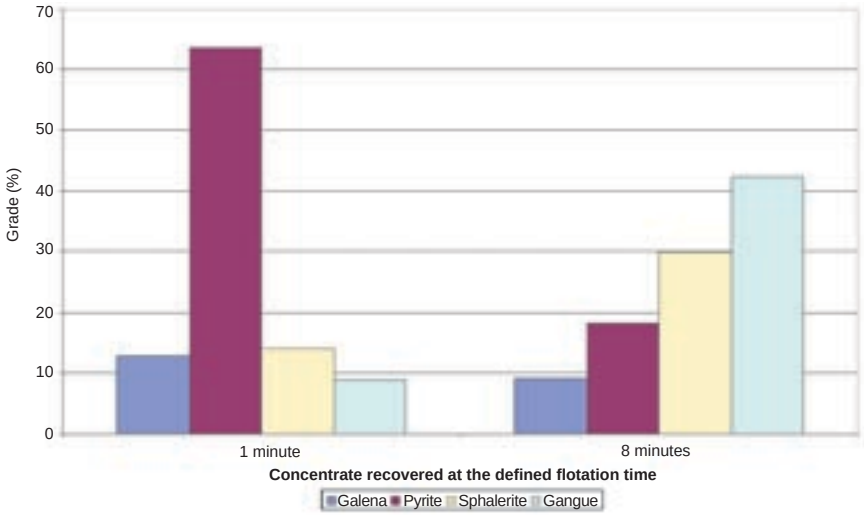


Figure 14—Mineralogical composition of the first and last concentrates after flotation in the presence of 50 g/t SNPX and 100 g/t NaCN at pH 8.5

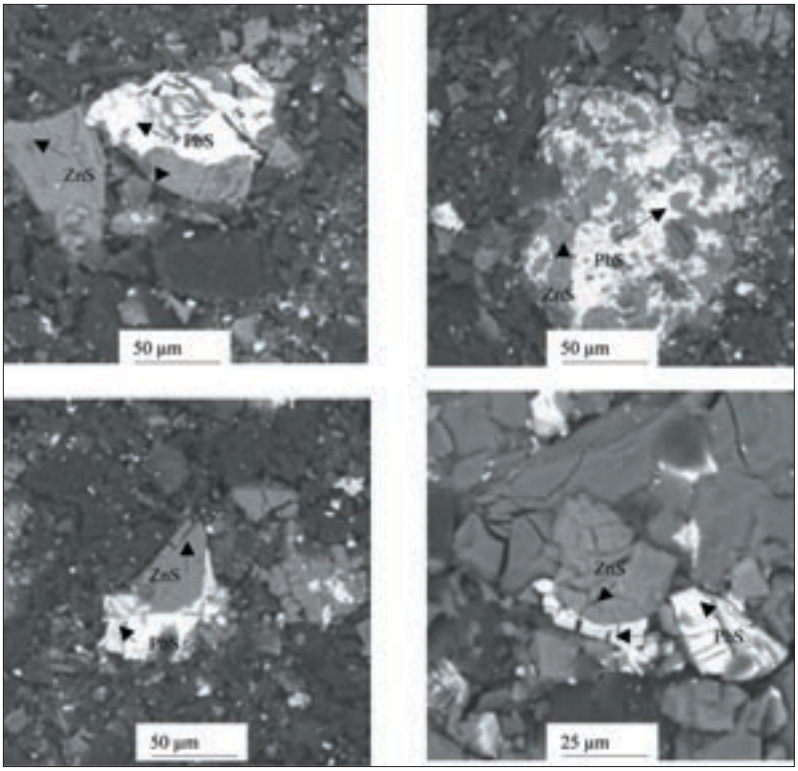


Figure 15—Backscattered electron images of concentrate showing the association between galena (white) and sphalerite (grey) in the galena concentrate

The results obtained in this study were compared with those obtained when using the flotation products from the Rosh Pinah plant, as discussed below

Deportment of sphalerite in the flotation products from the Rosh Pinah plant

Mineralogical examination of flotation products from the Rosh Pinah plant was conducted at Kumba Resources R&D (Pretoria) to study the presence of zinc in the galena concentrate in spite of the high dosage of cyanide used to decrease the recovery of sphalerite. The textural properties of the Rosh Pinah final lead concentrate were semi-quantitatively determined by optical particle counting, and the results are presented in Figure 18.

As seen in Figure 18, most of liberated galena particles were recovered in the -75µm size fraction, while the amounts of liberated sphalerite and gangue particles increased in the +75 µm size fraction. (Since flotation in the lead circuit is carried out at a primary grind of 80% passing 100 µm, the concentrate mass pull in the +106 µm fraction size will be negligible, and the results of particle counting of the +106 µm size fraction were omitted from Figure 18.) The distribution of liberated sphalerite and sphalerite attached to galena is shown in Figure 19.

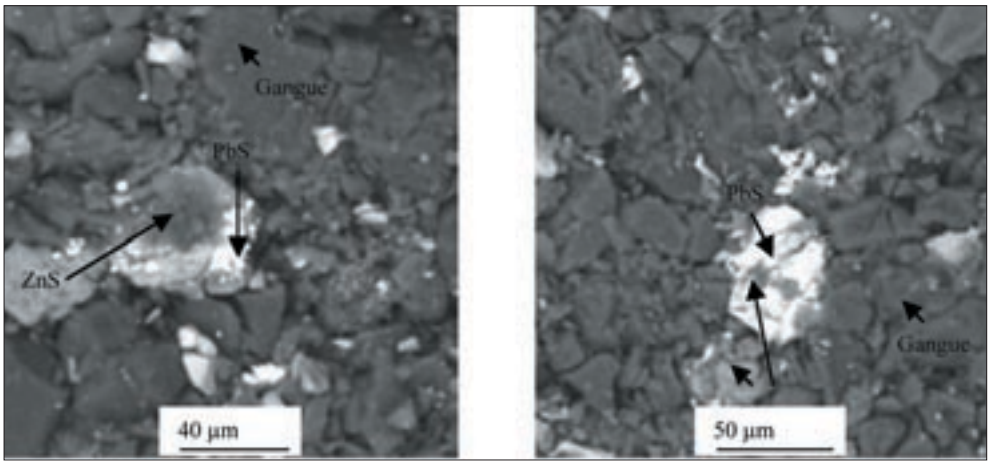


Figure 16—Backscattered electron images showing the association between galena and sphalerite in the lead rougher tailings

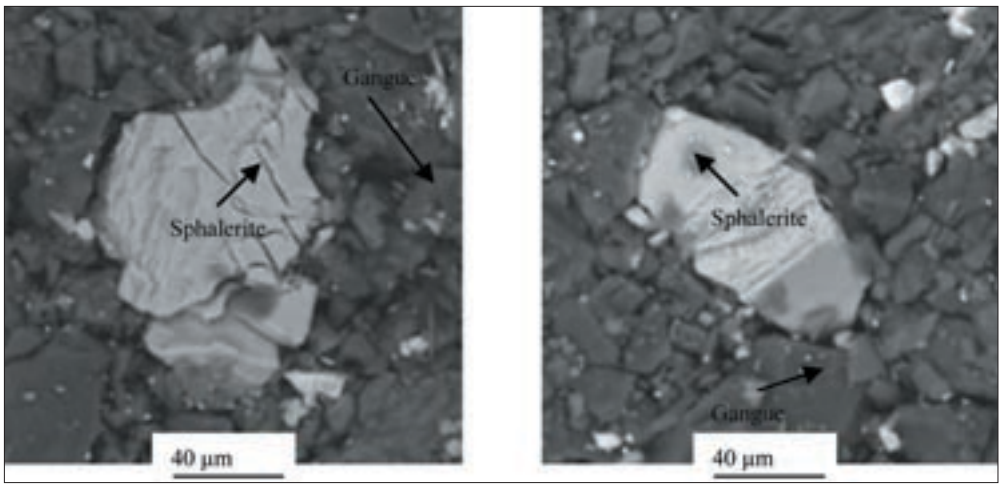


Figure 17—Backscattered electron images showing the general appearance of the rougher tailings. The flotation experiment was carried out in the presence of 100 g/t NaCN and 50 g/t SNPX

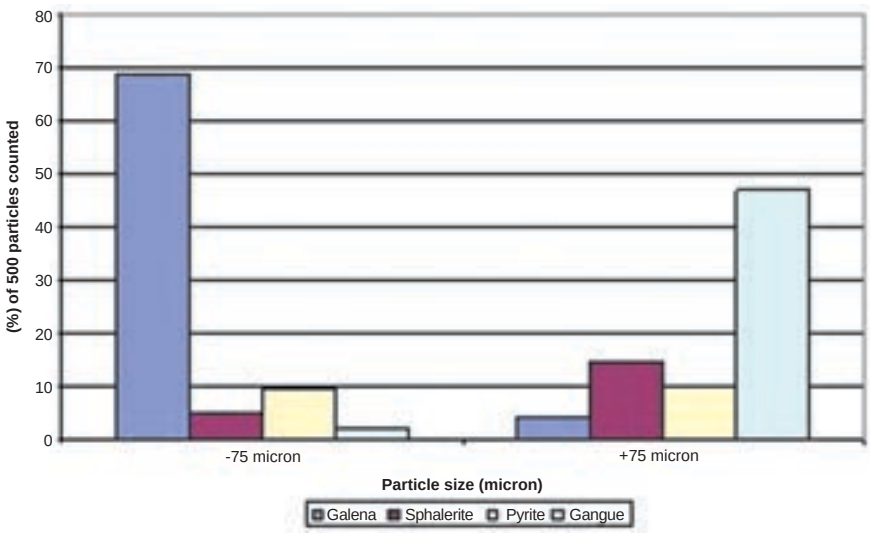


Figure 18—Mineral distribution in the lead concentrate from the Rosh Pinah Mine as a function of particle size (after Reyneke[8]). (Fully liberated minerals only.)

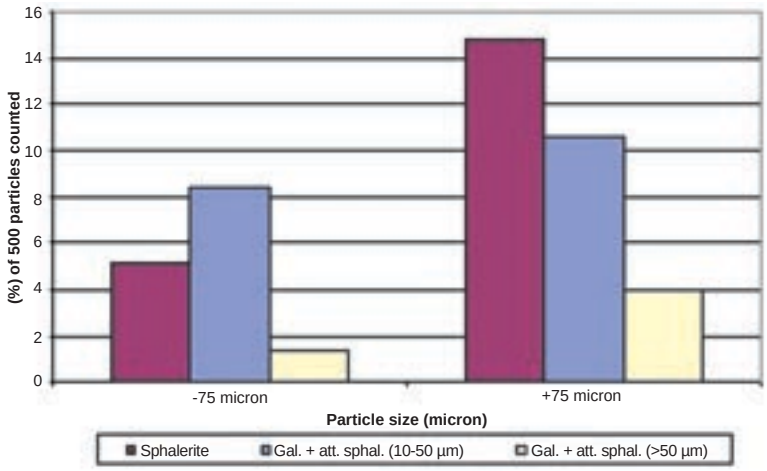


Figure 19—Sphalerite distribution in the lead concentrate from the Rosh Pinah Mine as a function of particle size (after Reyneke[8])

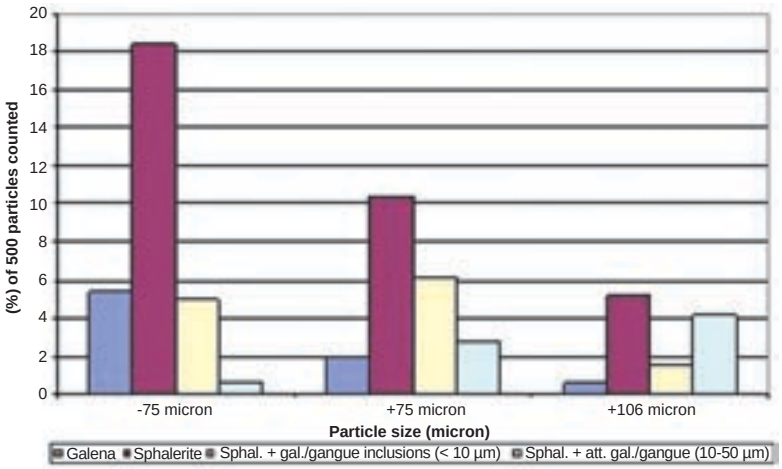


Figure 20—Galena and sphalerite distribution of the feed sample from the Rosh Pinah Mine as a function of particle size (after Reyneke[8])

The fraction of both liberated sphalerite and sphalerite particles attached to galena increased with increasing particle size. However, the fraction of binary locked particles of galena and sphalerite was higher than that of liberated sphalerite in the -75 µm size. The significant fraction of sphalerite particles (10–50 µm in diameter) attached to galena would affect the flotation selectivity in the lead circuit adversely. Clearly, liberation of sphalerite and galena has to be optimized to achieve maximal depression—increasing the depressant dosage during galena flotation is unlikely to be successful.

Since the flotation response of ores is usually a function of the primary grind, the mode of occurrence of the Rosh Pinah feed sample was also semi-quantitatively determined by optical particle counting and the results are presented in Figure 20.

As seen in Figure 20, it was clear that the fraction of liberated galena and liberated sphalerite increased with decreasing particle size. In addition, it was observed that the fraction of sphalerite and attached galena/gangue (10–50 µm) particles decreased with decreasing particle size of the feed sample. However, a considerable number of sphalerite particles with galena inclusions of less than 10 µm in size was observed in all size fractions. These binary sphalerite-galena particles would be difficult to depress.

Selectivity can be improved by better liberation of galena from sphalerite in the milling circuit, or alternatively by regrinding the rougher concentrate before the cleaning stage. However, practical implementation of this would need to take into account the softness of galena. In practice, it would be recommended to install a classifying cyclone before the regrind mill in order to avoid the over-grinding of fine particles from the rougher concentrate.

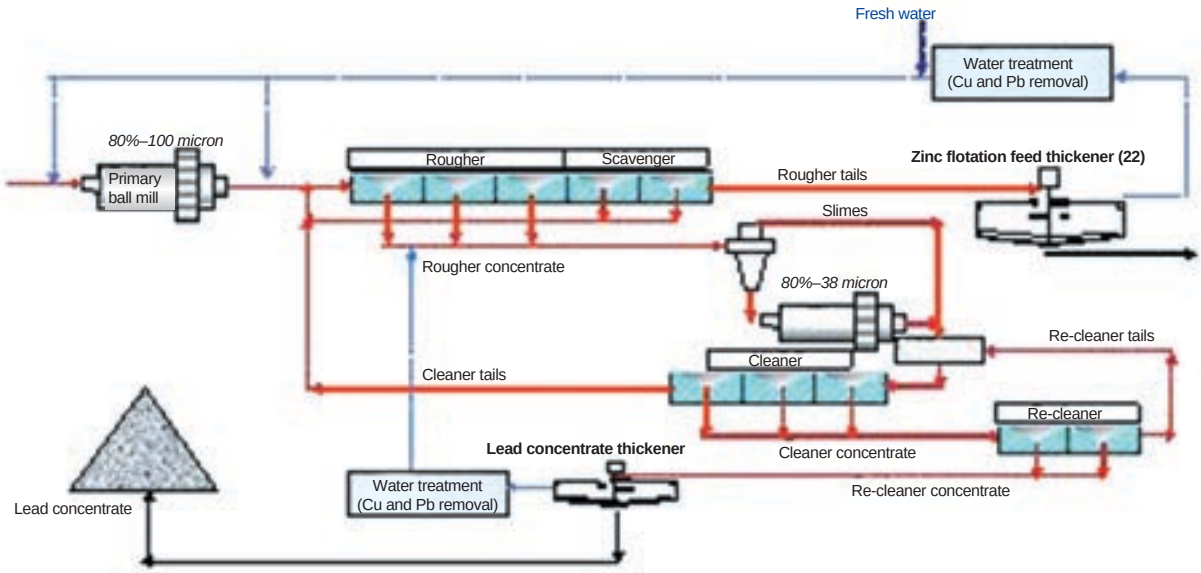
Based on the flotation and mineralogical results presented here, it is suggested that the flotation selectivity between galena and sphalerite could be improved by changing the current flow sheet by including a cyclone and regrind mill after the rougher flotation stage as shown in Figure 21.

The modified flow sheet can be summarized as follow:

Using a primary grind of 80% passing 100 micron to avoid the over-grinding of galena

Using up to 100 g/t NaCN to depress mainly copper-activated sphalerite in the lead rougher-scavenger flotation circuit and to maximize the recovery of galena

Using a cyclone to split the fine fraction (-38 micron)



Note: *The primary and regrind mills are in closed circuit with hydrocyclones that are not shown in this flowsheet*

Figure 21—Proposed flow sheet diagram of the lead flotation circuit at the Rosh Pinah Mine

from the middlings to avoid the over-grinding of fine galena particles (Figure 21)

Regrinding of the middlings from the rougher concentrate to improve the liberation of galena and sphalerite particles prior to the cleaning stages (Figure 21)

Cleaning of the rougher concentrate to achieve the required smelter grade (Figure 21). Figure 14 shows that pyrite and sphalerite were the major impurity sulphides in the lead rougher concentrate. Thus, it is recommended to increase the pH during the cleaning stage for an effective depression of pyrite (pyrite can be depressed at pH values higher than 9)

Using sodium cyanide and zinc sulphate in the cleaning stages to depress sphalerite and pyrite.

Conclusion

Batch flotation tests have shown that the use of cyanide alone is not efficient for the depression of sphalerite from the Rosh Pinah ore when milling is carried out according to the current plant particle size distribution. The use of both cyanide and zinc sulphate improved the depression of sphalerite much better than cyanide alone. In addition, an increase in the recovery and grade of galena was observed when cyanide or both cyanide and zinc sulphate were used.

Flotation selectivity is limited by the mineralogical texture of the Rosh Pinah ore sample. Microscopic analysis has shown that the presence of sphalerite in the galena concentrate is also due to poor liberation between galena and sphalerite, especially in the middlings. Hence selectivity could be improved by regrinding the rougher concentrate prior to the cleaning stage.

It is recommended that the flotation products such as rougher, scavenger and cleaner concentrates be analysed statistically using the QEM-SCAN to determine the correct fraction of locked and associated sphalerite particles in the lead concentrate.

It is also recommended that variability test work be conducted on the Rosh Pinah Eastern and Western ore field samples using the proposed flow sheet. In addition, a locked cycle test, which is a series of repetitive batch tests conducted in the laboratory, is required to simulate plant conditions before implementing the proposed reagent suite and flow sheet at the Rosh Pinah Mine.

Acknowledgement

The authors would like to thank Kumba Resources (Rosh Pinah Mine) for their financial support and permission to publish this paper.

References

1. KATABUA, J. and MOLELEKOE, R. Rosh Pinah Water Project. Report: Doc. No. RD-AP-015. Kumba Technology, R&D. Kumba Resources, Pretoria, South Africa, 2003. p. 12
2. TVETER, E.C. and MCQUISTON, F.W. Plant practice in sulphide mineral flotation. In D.W. Fuerstenau (ed.), *Froth Flotation*. 50th Anniversary Volume. AIME, New York, USA, 1962, pp. 382–426.
3. GERSON, A.R., LANGE, A.G., PRINCE, K.E., and SMART, R.ST.C. The mechanism of copper activation of sphalerite. *Applied Surface Science*, 1999, vol. 137, pp. 207–223.
4. FINKELSTEIN, N.P. The activation of sulphide minerals for flotation: a review. *Int. J. Min. Process*, 1997, vol. 52, pp. 81–120.
5. PRESTIDGE, C.A., RALSTON, J., and SMART, R.C. Role of cyanide in the interaction of ethyl xanthate with galena. *Colloids and Surfaces A: Physicochemical and Engineering Aspects*, 1993, vol. 81, pp. 103–119.
6. RALSTON, J. The chemistry of galena flotation: principles and practice. *Minerals Engineering*, 1994, vol. 7, pp. 715–735.
7. SEKE, M.D. and PISTORIUS, P.C. Effect of cuprous cyanide, dry and wet milling on the selective flotation of galena and sphalerite. *Minerals Engineering*, in press.
8. REYNEKE, L. Mineralogical composition and particle-counting of zinc, lead, feed and waste samples from Rosh Pinah, Namibia. Report No. M2000/33. Kumba Resources R&D, Pretoria, South Africa, 2000, p. 9. U