

# Improving Fine Sulfide Mineral Recovery — Plant Evaluation of a New Technology

D Engelhardt<sup>1</sup>, K Ellis<sup>2</sup> and B Lumsden<sup>3</sup>

## ABSTRACT

A full-scale statistically-based on/off plant trial of a new flotation conditioning technology designed to improve fine mineral recovery was undertaken to determine whether the recovery of selected fine sulfide minerals can be increased. The technology is selective for certain sulfide minerals only, including chalcopyrite and sphalerite. The technology has been shown to selectively aggregate the fine chalcopyrite and sphalerite minerals giving increased flotation recovery. The plant trials have shown an increase in the mineral recovery, without decreasing selectivity. Size by size recovery shows a significant increase in the fine (<20 µm) mineral recovery levels, such that the losses in the finest of these fractions were reduced significantly.

## INTRODUCTION

Froth flotation, like all mineral separation technologies, has limits. For froth flotation a major limitation is the efficient separation of fine minerals. A great challenge to the future of froth flotation in the minerals industry is to increase the efficiency of flotation separation of fine minerals.

While the particle size range where efficient flotation occurs varies depending on the mineral, generally the recovery of sulfide mineral by froth flotation is an efficient process for particles in the 10 µm - 100 µm range (Trahar, 1981; Trahar and Warren, 1976). For sulfide mineral particles less than 10 µm in size the poor recovery is generally accepted to be primarily due to the low collision efficiency of the particles with the air bubble (Trahar, 1981).

The recovery of fine particles is becoming increasingly important as many newer ore discoveries require fine grinding of the ore for efficient separation. Also, more stringent concentrate grade requirements, higher freight costs and management's focus on continuous improvement are leading to mineral processing operations grinding finer, most particularly their cleaner feed streams.

Finer grinding leads to increased costs, not just in the grinding stage, but also in flotation where the slower flotation kinetics of fine minerals requires greater flotation capacity.

A number of excellent reviews of fine mineral flotation have been written, including those by Trahar and Warren, 1976; Fuerstenau, Chander and Abouzeid 1979; Sivamohan, 1990. Sivamohan (1990) defines very fine particles as those less than 20 µm, and ultrafine as those less than 5 µm, but greater than 1 µm. It is these very fine and ultrafine particles that this research addresses.

This paper outlines a novel method to improve fine sulfide mineral recoveries. Primarily the paper seeks to give a brief theoretical perspective and detail on plant testing of the technology.

## Fine particle flotation

A model for particle collection by a bubble during flotation was developed by Schuhmann (1942).

Finch and Dobby (1990) utilised the model to calculate the collection efficiencies. For very fine and ultrafine particles, where detachment probability is small, collection depends on particle-bubble collision and particle-bubble attachment. The model can be represented by the equation:

$$E_K = E_A \cdot E_C$$

where:

$E_K$  is the collection efficiency

$E_A$  is the attachment efficiency

$E_C$  is the collision efficiency

As particle size decreases, particle momentum decreases, and  $E_C$ , which is proportional to particle size, also decreases. Finch and Dobby (1990) calculate that under fairly normal conditions  $E_C$  for a 40 µm particle is 0.43, but for a 10 µm particle under the same conditions  $E_C$  is 0.0024, a reduction of two orders of magnitude.

Finch and Dobby (1990) show that  $E_A$ , the attachment efficiency, increases with decreasing particle size.

While  $E_C$  and  $E_A$  show an opposing trend with particle size, Finch and Dobby (1990) show that for very fine and ultrafine particles, under most conditions, as particle size decreases so does  $E_K$ . The model supports Trahar's (1976) assessment that poor recovery of very fine and ultrafine particles is due to the particles poor collision efficiency with bubbles.

## Fine particle aggregation methods

Fuerstenau, Chander and Abouzeid (1979) outline in detail some of the different strategies that have been investigated to improve the recovery of fine mineral. These strategies include; chemisorbing collectors, carrier flotation, column flotation, agglomerate flotation, and electro-flotation. High intensity conditioning (shear flocculation), selective aggregation, flocculation with polymers and smaller bubble generation are detailed by Sivamohan (1990). Among the aggregation methods detailed are: selective aggregation by the addition of inorganic salts; shear flocculation; carrier flotation; and selective flocculation with polymers.

To be industrially successful fine particle aggregation must meet a number of criteria. Firstly, the method must be selective. Aggregation of all fine particles will not aid selective recovery of fine sulfide minerals. Secondly, the method must produce aggregates that are strong enough to survive the industrial flotation processes. Thirdly, the method must be cost-effective so that there is a net financial benefit to the mine.

## Magnetic aggregation of paramagnetic minerals

Some minerals recovered by froth flotation, including chalcopyrite, marmatite (sphalerite containing iron), bornite and cassiterite are paramagnetic. Just as importantly, some of the main minerals sulfide flotation attempts to reject like siliceous gangue are diamagnetic, or like pyrite, have a negligible

1. Manager Ore Processing, Cobar Management Pty Ltd, PO Box 31, Cobar NSW 2835.
2. Project Metallurgist, Newmont Golden Grove Operations, PMB 7, Geraldton WA 6530.
3. Centre for Multiphase Processes, University of Newcastle, University Drive, Callaghan NSW 2308.

magnetic susceptibility. This difference in magnetic properties between the recovered and rejected froth flotation minerals enhances selectivity. The magnetic susceptibility of some of the minerals in a flotation slurry is given in Svoboda (1987) and is reproduced in Table 1.

**TABLE 1**  
*Magnetic susceptibility of minerals.*

Mineral	Reported magnetic susceptibility (Svoboda, 1987) $M^3kg^{-1} \times 10^{-9}$
Chalcopyrite	1595.9
Bornite	100.5
Marmatite	38-5900
Cassiterite	2136.3
Pyrite	1-5
Quartz	-5.7

Svoboda (1981, 1982, 1987), Wang, Pugh and Forsberg (1994), Lu, Song and Dai (1988) and Tsouris, Scott and Harris (1995) are among some of the many researchers whose work has shown that strong magnetic fields will aggregate paramagnetic particles. Various workers (Svoboda and Zofka, 1983; Svoboda, 1987) have developed models to demonstrate the relationship between the variables that effect the aggregation of paramagnetic minerals. Interestingly, the testwork has been carried out almost exclusively on oxide minerals and has not included testwork on non-ferrous base metals sulfides, recovered by flotation. Svoboda and Zofka (1983) studied haematite, rutile and quartz, while Svoboda (1982) studied haematite, goethite and siderite, Tsouris, Scott and Harris (1995) haematite, Lu, Song and Dai (1988) rhodochrosite and siderite, and Wang, Pugh and Forsberg (1994) haematite and chromite.

The total energy of interaction of paramagnetic particles in a magnetic field is given by:

$$V_t = V_m + V_a + V_r$$

where:

$V_a$  is the energy of attraction known as the London-Van der Waals energy

$V_r$  is the energy of repulsion due to the electric double layer

$V_m$  is the energy of attraction of magnetised minerals

Svoboda (1987) shows that:

$$V_m = (8\mu_0\pi\kappa_1\kappa_2b_1^3b_2^3H^2) / (9\{h + b_1 + b_2\}^3)$$

where:

$\mu_0$  is the magnetic permeability of a vacuum

$\kappa_1$  and  $\kappa_2$  are the volume magnetic susceptibilities of the particles

$b_1$  and  $b_2$  is the radius of the particles

$H$  is the magnetic field strength

$h$  is the distance between the surfaces of the particles

The general form of the energy of interaction between two particles gives two minimum, a primary minimum and a secondary minimum. The particles can flocculate into either of these two minimum. Theoretically, Svoboda (1987) demonstrates how the particle-particle interaction varies with magnetic field strength and particle diameter for haematite. The model shows that under typical conditions two 3  $\mu m$  haematite particles will aggregate into the primary minimum in a magnetic field of

greater than about  $10^{-2}$  Tesla. Magnetic fields of at least an order of magnitude greater than  $10^{-2}$  T are readily produced by permanent magnets. Sulfide minerals like chalcopyrite and marmatite have a magnetic susceptibility comparable with haematite.

Svoboda (1987) also shows that as the particle size increases flocculation requires lower field strength. The model shows that two 8  $\mu m$  haematite particles will aggregate into a primary minimum in a magnetic field of about  $5 \times 10^{-2}$  T. In the flotation conditioning process, it is not necessarily a matter of aggregating a 3  $\mu m$  particle with another 3  $\mu m$  particle, though this will readily occur, but also the possibility of aggregating a 3  $\mu m$  particle with a 20  $\mu m$  particle. This is probably more important and occurs at much lower field strength conditions.

Aggregation into the secondary minimum is also possible and can occur at magnetic field strengths as much as one-tenth lower than that required for aggregation into the primary minimum.

$V_m$  depends only on: magnetic field strength, particle size and the magnetic susceptibility of the mineral. For sulfide minerals like chalcopyrite or marmatite, commercially available permanent magnets produce sufficient magnetic field strength to aggregate fine and ultrafine minerals.

### Practical application of paramagnetic aggregation

The magnetic susceptibility of particular sulfide minerals, the magnetic field strength available from commercial rare earth magnets, and the particle size of minerals that are poorly recovered by flotation, suggests that magnetic aggregation should work in practise. Others have shown that magnetic aggregation of paramagnetic minerals should occur (Svoboda, 1987), and others that aggregation of fine particles should improve flotation (Sivamohan, 1990), successfully combining these two processes for improved recovery of fine paramagnetic minerals in practise, has not been documented. Solving the problems of applying magnetic aggregation of fine paramagnetic minerals in a commercial flotation separation plant required the design and testing of a prototype. There are a range of variables that will affect the performance of this technology. Not only are there the variables associated with  $V_m$  (magnetic field strength, mineral magnetic susceptibility and particle size), but also plant variables, such as residence time, flow rate, flow turbulence, installation position within the process circuit and ferro-magnetic interference. Moreover, installation must be relatively simple and not interfere with other plant processes.

Added to these issues is the problem of finding sites where the mineralogy is suitable and the management is open to evaluating novel innovation.

ProFlote®, incorporating the magnetic conditioning technology is the equipment resulting from solving these practical problems. The ProFlote® technology applies a high strength magnetic field to the slurry and is installed immediately prior to the flotation separation stage. The flotation feed slurry passes through the series of ProFlote® tanks where the central core contains a series of high strength Nd/B/Fe permanent magnets. As the slurry enters the tanks it passes around the magnetic core where the slurry is magnetised and aggregation occurs before the slurry passes to the flotation cells. To overcome the presence of ferromagnetic material in the slurry there is an automatic cleaning cycle where the magnetic field is removed from the slurry for a brief period to allow the ferromagnetic material to be washed from the magnets, before being reinserted in the slurry.

Successful initial evaluation of the technology was undertaken at Triako's Mineral Hill mine, a small operation in NSW. Following this success, testing began on a scaled up ProFlote® at larger operations. The results of this testwork are presented here.

## Plant evaluation of paramagnetic aggregation technology

The paper presents the results from two plant trials, one at the CSA mine owned by Cobar Management Pty Ltd (CMPL) and the other at the Golden Grove Mine (NGGO) owned by Newmont Pty Ltd.

## RESULTS AND DISCUSSION

### Cobar Management Pty Ltd

CMPL own and operate the CSA Mine at Cobar, NSW. The underground mine produces approximately 600 000 tonnes per annum at an average feed grade of 6.0 per cent copper and 20 g/t silver which is treated in a conventional flotation circuit. Average monthly copper recovery has been between 91 per cent and 97 per cent, while average monthly copper concentrate grade has been between 27.9 per cent and 30.5 per cent copper. Feed grade varies from three to ten per cent copper and throughput rates vary from 60 to 130 tonnes per hour.

A review of available historical metallurgical data indicated that the majority of copper losses in tailings were in the  $-38\ \mu\text{m}$  size fraction, with up to 56 per cent of total copper losses reporting to this fraction. Changes to operating parameters and improved operator training resulted in minor improvements to overall recovery but these were not consistently achieved. Mineralogical studies indicated that the particles were predominantly liberated chalcopyrite with no evidence of oxidation or other obvious reasons for poor recovery. Further studies confirmed that increases in mill throughput rates and copper feed grades were resulting in either reduced copper recovery and/or lower concentrate grades.

CMPL agreed to perform a trial with a ProFlote® unit to be installed in the rougher feed stream. The unit arrived on site in March 2003 and was installed in April 2003. Installation was simple with minor pipe modifications required to sit the unit in the flotation feed stream after the flotation feed pump. The only service required was compressed air at 80 - 100 psi.

The first trial commenced in late April 2003 and ran for approximately one month. The unit was trialled with various on/off combinations on 12-hour shifts. Initial indications suggested an improvement in copper recovery of up to two per cent with similar or slightly lower concentrate grades. The only operational effect was poor level control of the flotation feed pump hopper. The initial ProFlote® unit operated with a cleaning cycle incorporating an air 'flush' at periodic intervals to remove build up of coarse sediments. This appeared to change the pump discharge head resulting in periodic loss of level control. The air flush has been removed in later versions and no significant build up has been observed.

Further on/off trials were performed with a range of throughput rates and feed grades. Conclusions at the end of these trials were:

- no discernible difference in copper recovery or concentrate grade when treating low throughput rates (65 t/hr) was detected;
- approximately three per cent improvement in copper recovery was achieved using the ProFlote® device when treating high throughput rates (130 t/hr) although concentrate grades dropped slightly (29.7 per cent to 29.4 per cent); and
- standard deviation of copper recovery reduced during ProFlote® operation supporting the observation of reduced 'spiking' in flotation tailings grades.

Overall the trials confirmed that the ProFlote® unit provided improved copper recovery when treating high metal loadings, due to either high feed rates or feed grades, presumably, due to

improved flotation kinetics. In addition, improvements in silver recovery and concentrate grade were also observed. There were no apparent benefits when treating lower metal loadings as increased circuit residence time appeared to be sufficient alone to achieve satisfactory recovery.

A comparison of flotation performance is shown in Table 2 comparing the results of three months just prior to the introduction of the ProFlote® unit and three months after the initial trials. Improvements in copper and silver recovery were clearly observed with similar or improved concentrate grades.

**TABLE 2**  
Plant results, before and after ProFlote® installation.

	Copper recovery (%)	Silver recovery (%)	Concentrate grade (Cu %)	Concentrate grade (Ag g/t)
ProFlote On	97.1	66.9	29.3	64.3
ProFlote Off	94.0	54.9	29.5	53.0
Difference	+3.1	+12.0	-0.2	+11.3

Separate studies indicated that the 'bottleneck' in the CSA flotation circuit was the lack of sufficient cleaning capacity. A second ProFlote® unit was trialled in the Cleaner Feed stream in late 2003 and trialled through to May 2004. There did not appear to be any benefit with the additional unit. Presumably this is because that once the fine particles are agglomerated, they remain so, and additional agglomeration is not achieved.

The ProFlote® unit has been in continuous operation since early May 2003. Copper losses in the  $-38\ \mu\text{m}$  size fraction in flotation tailings have reduced significantly as shown in Figure 1.

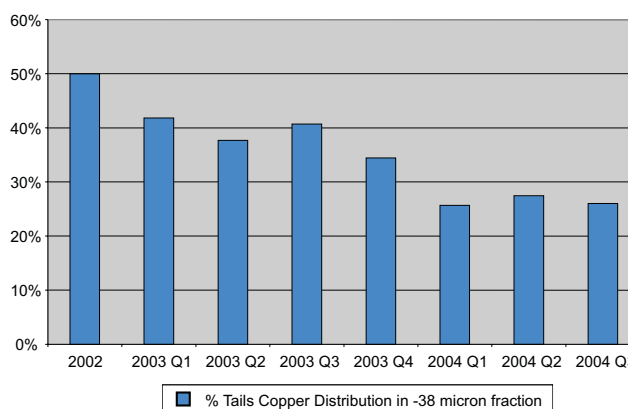


FIG 1 - CSA mine flotation tailings, per cent of Cu in tails  $<38\ \mu\text{m}$ .

Copper recoveries have been consistently above 97 per cent and concentrate grades  $>29$  per cent copper. The economic benefit of the unit is clearly recognised with additional revenue from improvements in copper and silver recovery significantly exceeding the unit's rental fee.

### Newmont Golden Grove Operations

Newmont's Golden Grove Operation (NGGO) is located north-north-east of Perth, approximately 370 km direct. The operation processes 1.3 million tonnes per annum of copper and lead/zinc ore campaigned separately, producing four different concentrates that are trucked approximately 275 km by road to the port at Geraldton. The copper is present as chalcopyrite, lead

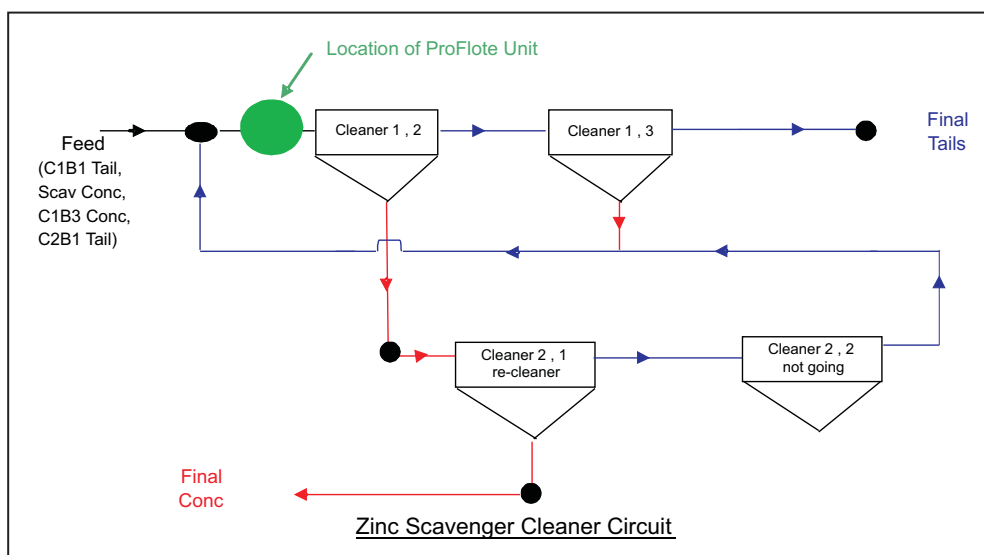


FIG 2 - Location of ProFlote® unit within the zinc flotation circuit flow sheet.

as galena and zinc as sphalerite, and when processed produce a copper low precious metal concentrate, a high precious copper or lead concentrate and a zinc concentrate that are shipped overseas to various smelters.

A major source of valuable mineral losses at NGGO tends to be in the fully liberated fines or slimes particles. Several improvements in the recovery of these fines have been made over the years through procedural and operating refinements, however liberated fines remain the largest single source of loss. The ProFlote® conditioning device was trialed in an attempt to recover the lost slimes values.

The unit was installed in the feed line to the Scavenger Cleaner circuit as that part of the circuit is common to both the treatment of copper and zinc ore. Installing the device here also meant that any slimes produced from the regrind circuit feeding the scavenger cleaner circuit may also be captured. A diagram of the location of the ProFlote® device within the zinc circuit is illustrated in Figure 2 with a circle.

ProFlote® can be turned 'on' or 'off', that is the magnetic field can be inserted in the slurry stream or removed from the slurry stream. On/off trials were conducted over a two-week period on copper and then zinc ore in an attempt to improve the recovery of copper and zinc. Samples of the feed, concentrate and tail were collected after the circuit was allowed to stabilise for 12 hours. The analysis of the results is discussed below.

**Zinc cleaner scavenger circuit**

The results from the two week on/off trial showed that there was a mean difference of 3.84 per cent improvement in zinc recovery when ProFlote® was on and a 1.65 per cent improvement in concentrate grade. A summary of the results is contained in Table 3.

A paired statistical analysis was conducted on the results to have confidence in the achieved results. This was done by comparing the 'on' results with the previous and following 'off' results. It was found that there was a 97.5 per cent confidence level in the 3.84 per cent increase in zinc recovery reported. The same statistical analysis was applied to the zinc concentrate grade results and that analysis showed that there was a 95 per cent confidence level that there is an improvement in concentrate grade.

A sizing analysis was performed on a couple of the on/off samples to determine what size fraction the ProFlote® unit had an effect on. The results are graphically represented in Figure 3.

**TABLE 3**  
Results from two-week trial campaigning zinc ore.

On/Off	Feed %Zn	Conc %Zn	Tail %Zn	Recoveries %Zn
Off	29.4	45.6	5.11	93.0
On	27.4	39.1	6.22	91.9
Off	16.2	36.3	3.83	85.4
On	10.8	32.6	1.56	89.9
On	26.0	40.1	3.50	94.8
Off	25.2	38.9	3.83	94.1
On	18.4	31.2	2.63	93.6
On	22.0	39.8	2.46	94.7
Off	23.0	37.4	6.68	86.4
On	33.0	46.0	5.96	94.1
Off	30.8	42.4	8.64	90.4
On	32.4	45.6	9.84	88.8
Off	33.4	46.4	7.94	92.0
On	35.2	47.2	8.39	92.6
On	30.3	44.2	6.04	92.7
Off	29.2	40.3	5.84	93.6
On	32.0	44.0	8.20	91.4
Off	32.3	44.6	8.84	90.6
On	34.3	47.4	9.61	90.3
Off	17.8	40.7	4.68	83.3
On	26.8	45.4	5.90	89.6
Off	31.0	46.3	11.00	84.6
On	30.3	46.2	7.54	89.8
Off	25.0	44.9	5.98	87.8
Off	33.9	46.8	8.30	91.8
On	28.4	43.4	3.36	95.6
Off	23.6	40.6	10.90	73.6
On	31.6	46.8	5.26	93.9
<b>Mean Off</b>	<b>27.39</b>	<b>42.50</b>	<b>7.16</b>	<b>88.34</b>
<b>Mean On</b>	<b>29.57</b>	<b>44.15</b>	<b>6.21</b>	<b>92.18</b>
<b>Difference</b>	<b>2.18</b>	<b>1.65</b>	<b>-0.95</b>	<b>3.84</b>

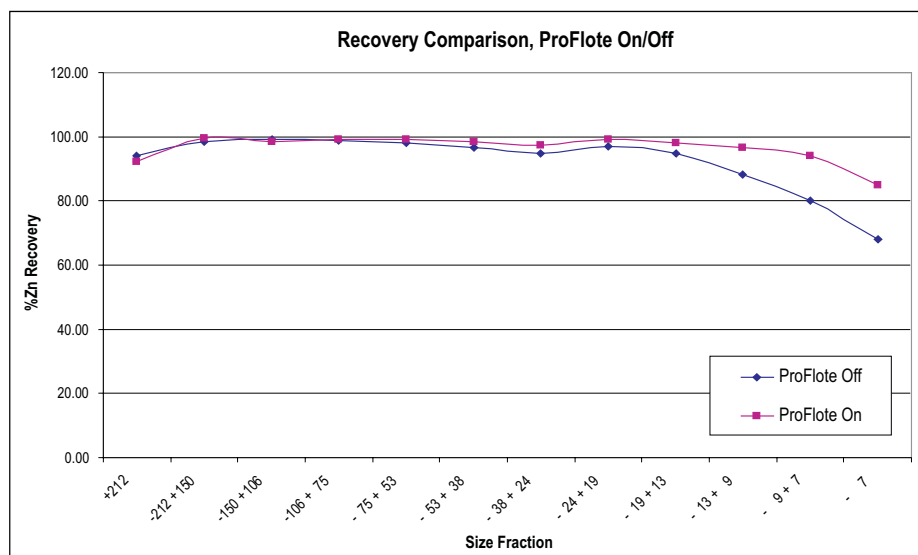


FIG 3 - Zinc recoveries by size fractions for zinc trial samples.

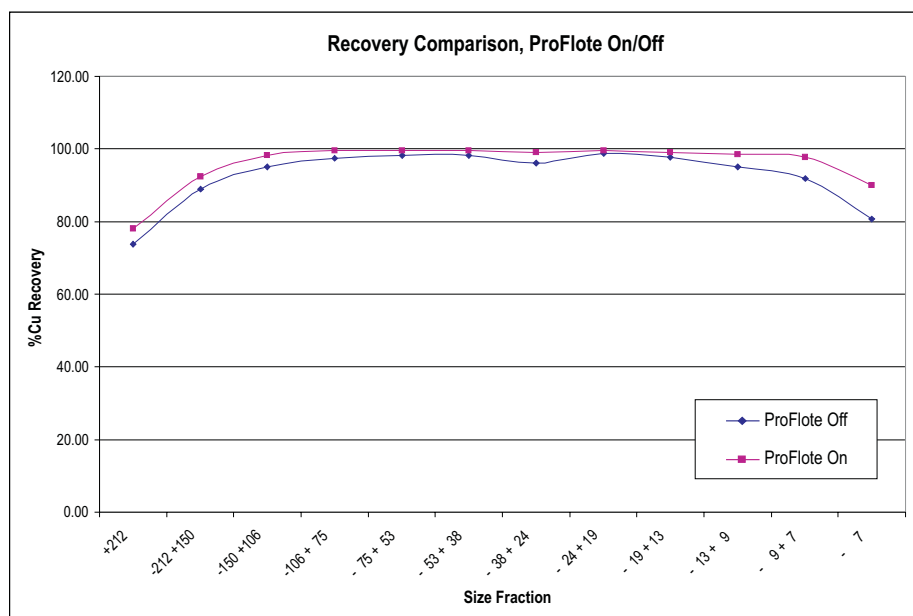


FIG 4 - Copper recoveries by size fractions for copper trial samples.

It can be seen from the figure that the scale of the recovery improvement increases as the size decreases below 19 µm with a difference of 16.88 per cent being established for the true slime fraction below 7 µm.

**Copper cleaner scavenger circuit**

The results from the two week on/off trial showed that there was a mean difference of 3.9 per cent improvement in copper recovery when ProFlote® was on and a 1.85 per cent reduction in concentrate grade. A summary of these results is presented in Table 4.

A similar style statistical analysis to that undertaken on the zinc results was performed on the copper results as well. It was found that there was a 97.5 per cent confidence level in the 3.9 per cent increase in copper recovery reported and up to 95 per cent confidence that there is a reduction in concentrate grade. The probable cause of the lower concentrate grade

produced is the lower circuit feed grades due to a lower grade recycle product from the Cleaner 1 Bank 3 tailings and the likelihood of not adequately compensating for this in plant operations.

A sizing analysis was performed on a pair of on/off data sets from the copper trial and the results of this assessment are graphically illustrated in Figure 4. It can be seen from the figure that the scale of the recovery improvement is relatively consistent at each size fraction with a slight increase as the size decreases below 13 µm with a difference of 9.08 per cent being established for the true slime fraction below 7 µm.

A survey was conducted on the entire circuit while treating both copper and zinc ore to establish the extent of the loss of valuable slimes. Based on the results from the survey and the results achieved through the use of the trial unit a second ProFlote® unit is now being installed onto the feed to the Scavenger circuit in an attempt to capture the unfloted slimes that report to tails.

**TABLE 4**  
Results from two-week trial campaigning copper ore.

On/Off	Feed %Cu	Conc %Cu	Tail %Cu	Recoveries %Cu
On	10.8	18.1	1.30	94.8
Off	20.8	26.9	3.10	96.2
On	11.5	16.8	1.44	95.7
Off	13.0	20.7	4.47	83.7
On	6.8	11.1	1.66	88.9
Off	9.7	19.7	2.10	87.6
On	9.8	19.4	1.20	93.5
Off	7.5	14.2	1.20	91.8
On	14.1	21.5	1.54	96.0
Off	12.5	20.8	2.65	90.3
On	14.2	23.3	1.34	96.1
Off	10.9	18.6	1.01	95.9
On	15.5	23.5	0.87	98.0
Off	14.2	25.4	1.65	94.5
On	12.5	25.3	1.66	92.8
Off	12.6	21.9	2.02	92.5
Off	9.2	17.6	2.87	82.1
Off	8.6	17.6	2.74	80.5
On	8.0	15.9	2.99	76.8
Off	12.0	21.6	5.06	75.5
On	6.7	12.3	2.58	77.6
Off	9.7	18.6	3.83	76.3
On	8.8	19.6	1.26	91.6
Off	9.0	24.9	1.87	85.8
<b>Mean Off</b>	<b>11.51</b>	<b>20.65</b>	<b>2.66</b>	<b>87.1</b>
<b>Mean On</b>	<b>10.79</b>	<b>18.80</b>	<b>1.62</b>	<b>91.1</b>
<b>Difference</b>	<b>-0.73</b>	<b>-1.85</b>	<b>-1.04</b>	<b>3.9</b>

## CONCLUSION

The data gathered at both sites shows that magnetic treatment of the flotation feed gave an improvement in chalcopyrite and sphalerite recovery. The plant testwork has shown the improvement in recovery is in the fine size fractions, and that there appears to be an improvement in flotation kinetics. The results are consistent with magnetic aggregation of fine paramagnetic minerals.

Improvements have been shown to occur in both a rougher-scavenger circuit and a cleaner circuit. At CSA where there is no concentrate regrind, additional magnetic treatment of rougher concentrate did not appear to give additional benefit. This would suggest that the aggregates are sufficiently resilient to survive flotation and pumping, so that single treatment where there is no regrind is sufficient. CSA's copper recovery after

ProFlote® installation on the rougher averages in excess of 97 per cent, so the potential for additional improvements in the cleaner is limited.

To improve flotation performance an aggregation method must be selective, resilient and cost effective. The testwork shows that magnetic aggregation of paramagnetic minerals gives selective and resilient aggregates that improve flotation performance. The operating costs of the technology are significantly less than the value of the benefit to the mine.

## ACKNOWLEDGEMENTS

Barry Lumsden would like to thank the Department of Industry Tourism and Resources for their financial support of his research, Professor Jameson of Newcastle University for his support and Ashley Kidd for his engineering expertise during the trial.

Kathryn Ellis would like to thank Ashley Kidd for his assistance during the trial of the unit and during the evaluation process.

## REFERENCES

- Finch, J and Dobby, G, 1990. *Column Flotation*, pp 37-49 (Pergamon Press: Oxford).
- Fuerstenau, D, Chander, S and Abouzeid, A, 1979. The recovery of fine particles by physical separation methods, in *Beneficiation of Mineral Fines Problems and Research Needs*, (eds: P Somasundaran and N Arbiter), pp 3-61 (The American Institute of Mining, Metallurgical and Petroleum Engineers, Inc).
- Lu, S, Song, S and Dai, Z, 1988. The hydrophobic and magnetic combined aggregation of paramagnetic minerals – a new way of fine particle separation, in *Proceedings XVI International Mineral Processing Congress* (ed: E Forssberg), pp 999-1009 (Elsevier: Amsterdam).
- Schuhmann, R, 1942. Flotation kinetics. 1. Methods for steady-state study of flotation problems, *J Phys Chem*, 46:891-902.
- Sivamohan, R, 1990. The problem of recovering very fine particles in mineral processing – a review, *International Journal of Mineral Processing*, 28:247-288.
- Svoboda, J, 1981. A theoretical approach to the magnetic flocculation of weakly magnetic minerals, *International Journal of Mineral Processing*, 8:377-390.
- Svoboda, J, 1982. Magnetic flocculation and treatment of fine weakly magnetic minerals, *IEEE Transactions on Magnetics*, Mag-18(2):796-801.
- Svoboda, J, 1987. *Magnetic Methods for the Treatment of Minerals*, pp 304-315 (Elsevier: Amsterdam).
- Svoboda, J and Zofka, J, 1983. Magnetic flocculation in secondary minimum, *Journal of Colloid and Interface Science*, 94(1):37-44.
- Trahar, W J, 1981. A rational interpretation of the role of particle size in flotation, *International Journal of Mineral Processing*, 8:289-327.
- Trahar, W J and Warren, L J, 1976. The floatability of very fine particles – a review, *International Journal of Mineral Processing*, 3:103-131.
- Tsouris, C, Scott, T and Harris, M, 1995. Para- and dia-magnetic particle flocculation in a magnetic field, *Separation Science and Technology*, 30(7-9):1407-1418.
- Wang, Y, Pugh, R and Forssberg, E, 1994. The influence of interparticle surface forces on the coagulation of weakly magnetic mineral ultrafines in a magnetic field, *Colloids and Surfaces, A: Physicochemical and Engineering Aspects*, 90:117-133.