Magnetic Conditioning at the Hellyer Tailings Retreatment Plant

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ABSTRACT

During periods of high metal prices and strong metal demand tailings retreatment can be economic. Most tailings retreatment operations are technically challenging, low margin operations, because the easily recovered mineral has already been removed, therefore, process improvements can potentially yield large economic benefits. The Hellyer tailings dam was owned by Intec Ltd and contained 10.9 million tonnes grading 2.8 per cent zinc, 3.0 per cent lead, 88 g/t of silver, 0.16 per cent copper and 2.6 g/t gold. In November 2006 the Polymetals Group, as operator, restarted the Hellyer plant to retreat these tailings. Within a few months the operation had stabilised and Polymetals was meeting budgeted targets of throughput, metal recovery and concentrate quality. Polymetals and Intec then investigated methods that would improve the metallurgical result for this difficult to treat tailings. Since the tailings are fine-grained with 80 per cent <45 µm, magnetic aggregation of the paramagnetic sphalerite was evaluated. Magnetic aggregation of an oxidised tailings feed has not been tested before, nor had the technology been available when Hellyer ore was originally processed. Extensive statistical test work showed magnetic aggregation of the sphalerite fines in the zinc cleaner circuit increased total zinc recovery. The increase in the mean zinc recovery with magnetic conditioning was 3.63 per cent, at a statistical level of confidence of 97.8 per cent, and at the same mean zinc concentrate grade.

INTRODUCTION

Hellyer history

The Hellyer deposit is located about 80 km south of Burnie in north-western Tasmania and was discovered in 1983 by the Australian mining company Aberfoyle. The deposit was 15 million tonnes of massive sulfides, grading 13 per cent zinc, seven per cent lead, 0.35 per cent copper with 160 g/t of silver and 2.3 g/t of gold (Lane and Richmond, 1993). Concentrate production commenced in 1989 and processing continued until 2000 when the mine was closed and the plant placed on care and maintenance.

The ore was a fine-grained massive pyritic sulfide orebody, assaying about 85 per cent sulfides, with the sphalerite and galena finely disseminated throughout (Lane and Richmond, 1993). The mineral texture is variable and the inter-mineral associations also vary. The primary grind required for liberation was 80 per cent <35 - 45 µm, with P80 of the lead and zinc regrinds being around 20 µm. Sulfide particles greater than 30 µm in the flotation feed were frequently sulfide composites (Lane and Richmond, 1993).

While a number of studies investigated retreating the tailings, including a pilot plant study in 1999/2000 and various hydrometallurgy studies, it was not until the 2006 increase in metal prices that tailings retreatment became economic, with regrinding and flotation selected as the most economic process. The original Aberfoyle plant was modified to suit the tailings feed (Lawry, Platts and Bott, 2008). The Hellyer Zinc Concentrate Joint Venture, established in 2006 was a 50:50 joint venture between the ASX listed Intec Ltd and the privately owned Polymetals Group. Intec owned the Hellyer tailings resource and Polymetals Group, with the operational and processing skills, was the operator of the project.

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Hellyer tailings deposit

The tailings dam contained 10.9 million tonnes of tailings that graded 2.8 per cent Zn, 3.0 per cent Pb, 88 g/t Ag, 0.16 per cent Cu and 2.6 g/t Au. The copper is present as chalcopyrite, the lead as galena and the zinc as sphalerite. One feature of the tailings is its fine particle size. The P80 of the dam tailings is <45 µm, but more significantly a plant survey showed that 30 per cent of sphalerite in plant feed is <7 µm and about 80 per cent is <38 µm. The galena is also fine with 50 per cent <7 µm. The sulfide minerals are generally oxidised having been in the tailings dam at least seven years and up to 15 years, with most of the galena in particular very heavily oxidised and effectively unfloatable (Lawry, Platts and Bott, 2008).

Aberfoyle's Hellyer plant

The original Hellyer plant is described in detail in the literature (Richmond and Campbell, 1992; Lane and Richmond, 1993). Like most copper, lead and zinc ores the Hellyer mineralogy was complex and the sulfides difficult to separate. Four concentrates were produced, a copper/silver/gold concentrate, a lead concentrate, a zinc concentrate and a lead/zinc concentrate. Because mineral separation was difficult, production of these saleable concentrates meant that some valuable minerals, particularly sphalerite, were lost to tailings. The plant management evaluated a range of technologies to improve the mineral separation and extensive studies of various technologies, both physical and chemical were undertaken. One of the technologies that the Hellyer mill pioneered was high intensity conditioning (HIC). Holder (1994) reports that the introduction of HIC in the copper cleaner circuit increased copper recovery by 13 per cent and silver recovery by seven per cent. Before HIC, surface oxidation limited the copper and silver recovery in the cleaner, and HIC removed these oxidation products from the mineral surface. Nevertheless, a significant percentage of the difficult to recover minerals were lost to tailings.

Table 1 shows the Hellyer targets for concentrate grade and recovery, from the first few years of operation (Lane and Richmond, 1993). After substantial improvements in the plant Hellyer metallurgical outcomes exceeded the best expectations of the feasibility study.

Lane and Richmond (1993) summarise the laboratory and plant studies undertaken by Aberfoyle to investigate the metallurgical performance in the original plant. Their paper details the modifications that were made to the plant and to the plants operating procedures to improve plant metallurgy.

Lead recovery for the different size fractions for the Hellyer lead circuit is shown in Figure 1 (Lane and Richmond, 1993). Lead rougher recovery is better than lead cleaner recovery and for both; recovery is best between 7 - 30 µm.

Zinc recovery for the different size fractions for the Hellyer zinc circuit is shown in Figure 2 (Lane and Richmond, 1993). The key to improving sphalerite metallurgy was found to be improving the metallurgy of the sub 5 µm sphalerite. Of particular importance was improving the flotation rate of the sphalerite relative to the pyrite, especially in the cleaner circuit. The rate of flotation of sphalerite was slow because of its fineness and because of rapid surface coating. The sphalerite's flotation rate was comparable to pyrite, so selective flotation was difficult. Analysis of the final zinc concentrate showed the presence of fine liberated pyrite, but attempting to reduce the recovery of this fine liberated pyrite invariably led to lower sphalerite recovery. About 55 per cent of

TABLE 1
Target metal grades and recovery for the original Aberfoyle plant (Lane and Richmond, 1993).

Stream	Copper		Lead		Zinc		Silver	
	Grade %	Recovery %	Grade %	Recovery %	Grade %	Recovery %	Grade ppm	Recovery %
Feed	0.35	100	7.5	100	13.5	100	160	100
Cu/Ag concentration	14	25					5000	20
Pb concentration			60	40			600	19
Zn concentration					50	63		
Bulk			15	13	35	17	250	10

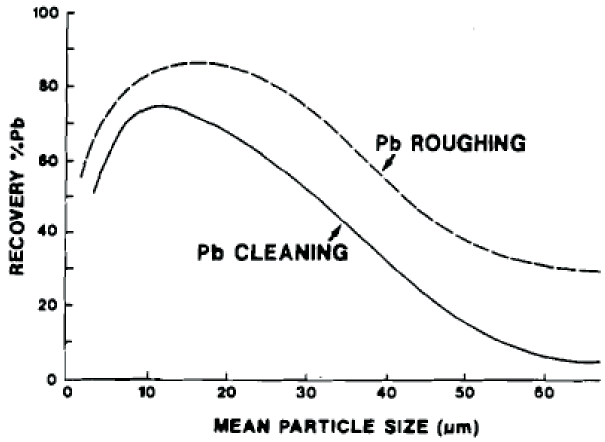

FIG 1 - Lead recovery by particle size for original Hellyer operation (Lane and Richmond, 1993).

the pyrite in final sphalerite concentrate was liberated, and only 45 per cent composited with sphalerite. Figure 2 shows that sphalerite recovery in the rougher was better than in the cleaner across the size fractions. The work of Lane and Richmond (1993) showed that the sphalerite recovery decreased in the coarser fractions because of composites, and in the fine fractions because of poor selectivity against free pyrite. It can be seen that the real zinc recovery problem at Hellyer was in the cleaner circuit where Lane and Richmond (1993) showed it was selectivity against fine pyrite that was the main challenge. Another factor that also affected sphalerite recovery was high levels of fine galena in the zinc rougher feed.

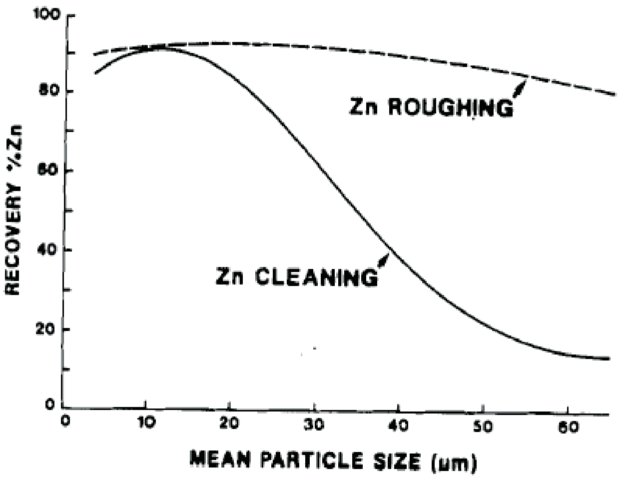

FIG 2 - Zinc recovery by particle size for original Hellyer operation (Lane and Richmond, 1993).

Lane and Richmond (1993) showed that some of the selectivity against pyrite could be improved by split conditioning in the cleaning circuit and by redox control in the regrind mill.

Hellyer retreatment plant

Retreatment of tailings always poses technical problems. If the original plant flow sheet and plant conditions are used to retreat the tailings then it is an attempt to selectively recover mineral that the plant has already been unable to recover selectively. Some tailings mineral would be recoverable, because of regrinding and retreating, but retreating low-grade tailings profitably is a technical challenge. If the mineral was not selectively recoverable in the original processing, then its residence in the tailings dam for a decade is unlikely to enhance its selective recovery. Indeed, the galena at Hellyer is almost unrecoverable because it has oxidised while in the tailings dam. The technical challenge for Polymetals and Intec was to efficiently and selectively recover in a saleable concentrate the minerals in the tailings that had originally been too difficult to recover selectively. Because of the complexity of this fine grained ore Polymetals and Intec has been open to using technology, either optimising existing technologies, or seeking technologies that were not available to the original operation to enhance the difficult mineral separation, and hence the profitability of the operation.

The metallurgical balance for the retreatment plant is given in Table 2 and the plant flow sheet in Figure 3.

The size distribution in the rougher feed and the size-by-size recovery in the rougher circuit are given in Table 3 and the same information for the cleaner circuit is in Table 4. The zinc recovery by particle size for the rougher and cleaner circuit is graphed in Figure 4.

It can be seen from Figure 4 that the particle size recovery graph has some similarities to the graph (Figure 2) from the original plant, with similar relative zinc recoveries in the mid- range particles and better rougher recovery in the coarser fractions, though less pronounced – reflecting that only the more difficult minerals are left. In the original plant the rougher recovery was superior for the finest fractions, whereas in the retreatment plant the zinc recovery was higher in the cleaner for the finest fraction. Clearly, the finest fractions are where the sphalerite is distributed, with 80 per cent of the zinc in cleaner feed less than 15 µm and 56 per cent of the zinc in rougher less than 15 µm. Importantly, as Lane and Richmond (1993) have shown it wasn't zinc recovery as such that was the challenge at Hellyer, rather it was selective zinc recovery relative to pyrite that was critical. Any technology that would assist the process must be selective against pyrite. The sphalerite because of its decade of residence in the tailings dam would no doubt have some surface oxidation but, unlike the galena, is still recoverable.

IMPROVING THE RECOVERY OF FINE SPHALERITE

It is commonly believed that flotation like most mineral separation processes is size limited. The efficiency of separation

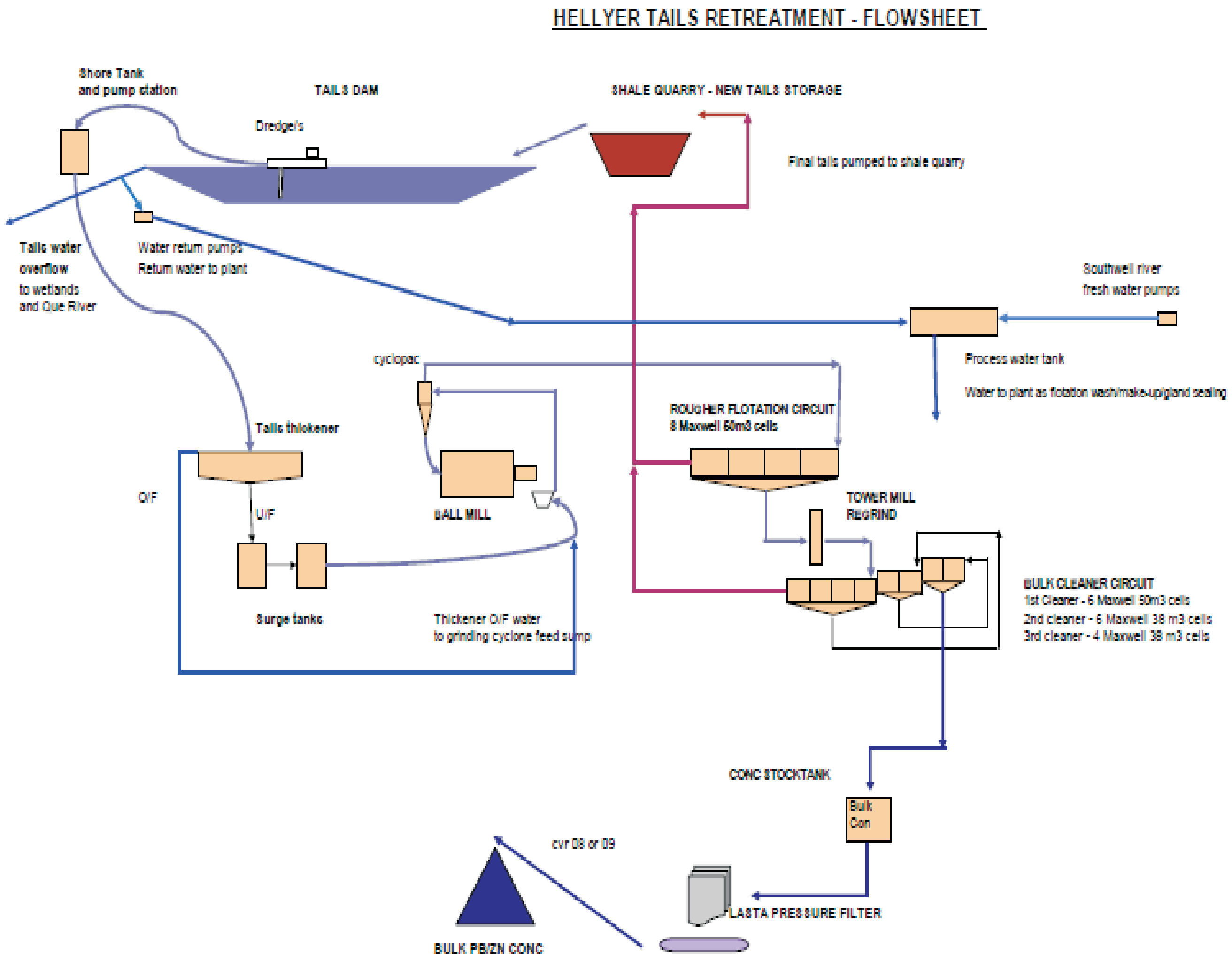

FIG 3 - Hellyer tailings retreatment plant flow sheet (Lawry, Platts and Bott, 2008).

decreases as particle size decreases below a certain particle size. It is argued by some, that the poor selective recovery of fine minerals against coarser mildly hydrophobic particles is due to the poor collision efficiency of fines with bubbles (Duan, Fornasario and Ralston, 2003; Jameson, Nguyen and Ata, 2007). The particle size where selective recovery deteriorates varies from mineral to mineral and operation to operation but Trahar (1981) gives an excellent review of the difficulty of selectively recovering fine sulfide particles. This fine mineral recovery problem Aberfoyle faced at the original Hellyer operation and employed various strategies to ameliorate the problem included long flotation residence times, split conditioning in the regrind circuit, and producing a bulk concentrate (Lane and Richmond, 1993). Nevertheless, while Aberfoyle made substantial improvements in zinc recovery (Richmond and Campbell, 1992) the tailings dam still averaged 2.8 per cent zinc. In fact Polymetals faced a more difficult problem in that they were attempting to selectively recover mineral that had already not been recovered

TABLE 2

Metallurgical balance for the retreatment plant.

	% Mass flow	Assay		% Distribution	
		% Pb	% Zn	Pb	Zn
Feed	100.0	3.2	3.0	100.0	100.0
Rougher concentrate	25.1	8.6	9.1	69.5	80.2
Rougher tail	74.9	1.3	0.7	30.5	19.8
Final concentrate	4.3	13.0	37.3	19.0	56.2
Cleaner tail	20.8	7.4	2.7	50.5	24.0
Final tail	95.7	2.8	1.3	81.0	43.8

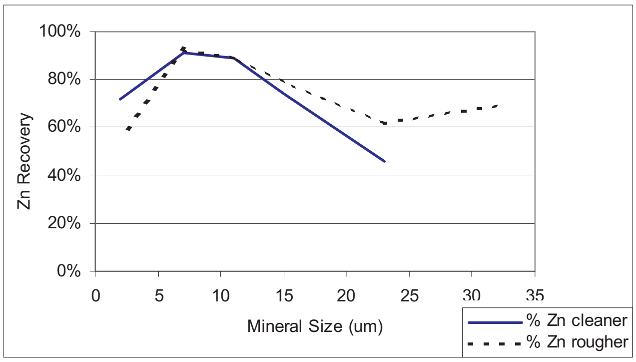

FIG 4 - Zinc recovery by size, cleaner and rougher circuit for tailings retreatment plant survey in 2007.

TABLE 3

Zinc distribution and recovery by size in the rougher circuit for the retreatment plant.

Size	% Zn distribution rougher feed	% Zn rougher recovery
+32 μm	16	69
+23 μm	14	62
+15 μm	14	79
+11 μm	9	89
+7 μm	6	92
-7 μm	41	56
Total	100	67

TABLE 4

Zinc distribution and recovery by size in the cleaner circuit for the retreatment plant.

Size	% Zn distribution cleaner feed	% Zn cleaner recovery
+23 μm	6	46
+15 μm	14	74
+11 μm	15	89
+7 μm	10	91
-7 μm	54	72
Total	100	77

in the initial plant and that had been impounded in a tailings dam for about ten years with resultant deterioration in surfaces.

The literature contains a range of processes that have been evaluated to improve fine mineral recovery. Furstenau, Chander and Abouzeid (1979) published a comprehensive review of the various methods. These methods range from chemical methods such as flocculation or coagulation to physical methods such as shear flocculation or methods to reduce bubble size. Many of these studies have been laboratory studies, and there is little technology in plant use specifically designed to target fine mineral. Split flotation, where the flotation feed is split by particle size and the two sizes are floated separately under different flotation conditions, has been one of the more popular methods utilised in Australia to increase fine particle recovery.

Magnetic aggregation

Recently, there have been a number of papers published that show that fine paramagnetic mineral's flotation recovery can be selectively increased in the plant by magnetic conditioning of the flotation feed (Engelhardt, Ellis and Lumsden, 2005; Rivett, Wood and Lumsden, 2007; Holloway, Clarke and Lumsden, 2008). These studies are statistical plant studies, not laboratory based studies. The magnetic conditioning selectively aggregates the fine paramagnetic mineral increasing the effective size of the mineral. Fine sphalerite that contains small amounts of iron substituted for zinc in the sphalerite mineral matrix has been shown to be paramagnetic (Svoboda, 1987). Pyrite has a low magnetic susceptibility. This difference in magnetic susceptibility between sphalerite and pyrite can be exploited to increase fine sphalerite recovery but not fine pyrite recovery. The range of measured magnetic susceptibilities for pyrite and sphalerite as well as the measured magnetic susceptibility of the Hellyer sphalerite are given in Table 5. Engelhardt, Ellis and Lumsden (2005) have shown that magnetic aggregation could selectively increase recovery for sphalerite particles finer than 38 μm. Moreover, the increases in recovery were selective with no loss in concentrate grade. In effect an improved grade recovery curve resulted.

TABLE 5

Magnetic susceptibilities.

Mineral	Magnetic susceptibility $M^3kg^1 \times 10^{-9}$
Sphalerite (containing Fe)	38 - 5900 (Svoboda, 1987)
Pyrite	1 - 5 (Svoboda, 1987)
Hellyer concentrate	1165

The aggregation of ferromagnetic particles is extensively studied and well understood, but the aggregation of paramagnetic particles was only investigated in the 1980s (Svoboda, 1981, 1982, 1987; Svoboda and Zofka, 1983; Lu, Song and Dai, 1988; Skvarla and Zelenak, 2003). These workers investigated the key parameters that impacted on the magnetic aggregation of paramagnetic particles. They showed that with the new high strength magnetic fields available from rare-earth permanent magnets fine paramagnetic particles could be aggregated.

Svoboda (1987) has shown that the total energy of interaction (Vt) 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

V_a is a relatively small energy component compared to V_m and V_r

Aggregation will then depend on the magnitude of the opposing attractive and repulsive forces, V_m and V_r. Svoboda (1987) derives an expression for V_m:

$$V_m = (8\mu_0 \pi \kappa_1 \kappa_2 b_1{}^3 b_2{}^3 H^2) / (9\{h + b_1 + b_2\}^3)$$

where:

μ_0 is the magnetic permeability of a vacuum

κ_1 and κ_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

V_r, the electrostatic repulsion is proportional to the particle sie and the square of the particle surface charge.

Therefore, if $V_m > V_r$ then aggregation will occur. Aggregation has been shown to occur for paramagnetic particles as fine as 1 μm, even at relatively low magnetic induction (Wang, Pugh and Forssberg, 1994).

Engelhardt, Ellis and Lumsden (2005) showed that sphalerite recovery could be improved through magnetic aggregation, but their work was on fresh ore. The important difference at the Hellyer retreatment plant, compared to other operations where magnetic aggregation has proven successful is that the Hellyer sphalerite was aged in the tailings dam, where some oxidation of the sphalerite and the other minerals would no doubt have occurred. Also as Lane and Richmond (1993) had observed the sphalerite surface rapidly became coated after grinding, and this is important in the Hellyer plant where flotation residence times particularly in the cleaner are substantial. Of interest in this test work is the effect of mineral ageing on magnetic aggregation.

Does oxidation change the repulsion energy (V_r) significantly? It has been shown by a number of workers (Healy and Moignard, 1976; Vergouw, Difeo and Finch, 1998) that the oxidation of sphalerite leads to surface coating of zinc oxide-hydroxides on the sphalerite surface. The effect of sphalerite oxidation is to reduce the magnitude of the surface charge of the sphalerite, and correspondingly to increase the pH at which the surface charge is zero, the iso electric point (iep). Healy and Moignard (1976) showed that the iep increased from pH 2 to pH 8.5. This would suggest that at the Hellyer flotation pH of around 11, V_r is reduced relative to fresh ore, making magnetic aggregation more probable.

However, the sphalerite in the cleaner feed would be expected to be relatively free from surface oxidation because it is reground, then treated by high intensity conditioning, a surface cleaning operation, immediately prior to cleaner flotation. Surface oxidation may not be the major effect of the tailings impoundment on V_r for sphalerite; rather, the V_r for sphalerite may be more affected by the high concentrations of dissolved cations, particularly iron from the oxidation of the pyrite.

Vergouw, Difeo and Finch (1998) studied the zeta potential and agglomeration of <38 μm sphalerite. They showed that as the pH increased above the iep (pH 4) the agglomeration rate also increased, reaching a maximum at pH10. This was a surprising result, because for the other sulfide minerals, the maximum agglomeration rate occurred at the iep. Their explanation was that a hydrophobic force associated with surface speciation was present. Whether this hydrophobic force is present in the flotation environment where the sphalerite has been activated with copper sulfate, or where there is surface oxidation, is another question? Vergouw, Difeo and Finch (1998) also showed that at pH greater than ten the presence of iron ions reduced the rate of sphalerite agglomeration while the presence of lead ions had little effect on the sphalerite agglomeration. For sphalerite- galena mixtures and sphalerite-pyrite mixtures Vergouw, Difeo and Finch (1998) showed that agglomeration decreased significantly at alkaline pH, probably due to galvanic interactions between the minerals.

The complexity of modelling the effect of different parameters on V_r is even greater because Healy and Moignard (1976) have shown that the zeta potential is also affected by the slurry's per cent solids and the iron content of the sphalerite.

The studies by Healy and Moignard (1976), and Vergouw, Fideo and Finch (1998) show that the parameters affecting V_r are very complex indeed, particularly for an oxidised ore. Predicting the impact of oxidation on V_r for the Hellyer retreatment plant is difficult. But given that the sphalerite in flotation feed is probably not highly coated in oxidised zinc compounds after grinding and high intensity conditioning, but that there are high levels of iron cations it would be expected that V_r is greater for the sphalerite at Hellyer, than it would be for a fresh ore. The test work needed to determine whether the change in magnitude of repulsion energy V_r was sufficient to negate the effect of the magnetic energy of attraction so that magnetic aggregation could not occur.

Experimental

The magnetic conditioning was applied by installing the magnetic conditioners known as ProFlote just prior to the flotation stage. At Hellyer the decision was made to test (separately) in both the rougher and the cleaner circuit, since fines are lost in both circuits. However, as Lane and Richmond (1993) have shown and as Polymetals own plant surveys showed, the major challenge at Hellyer was selective flotation of <7 μm sphalerite in the cleaner circuit. Consequently, test work was initiated in the cleaner circuit, followed by test work in the rougher circuit.

A randomised paired statistical experiment was devised. One of the particular conditions at Hellyer that needed to be addressed in the experimental design was the very long residence times in the cleaner circuit. It was estimated that the residence time in the cleaner could be 13 hours. Since the plant automatic samplers were 24 hour samples the experimental set-up allowed a 24 hour stabilisation time between switching the magnetic conditioners ON or OFF, and the collection of experimental data. The only complication with a randomised paired test, where there is long residence time, is if there is plant disruption due to extensive downtime or other plant issues. If this occurs, then a pair of data is lost, which for Hellyer was four days of testing. Nevertheless, pairing particularly where a tailings dam is being dredged as feed does reduce the variability of the results.

INSTALLATION

Since each plant is unique, retrofitting a circuit with a new piece of equipment at minimal cost and engineering can be very difficult. Often space needs to be made in the circuit. Because it aggregates the minerals for flotation, magnetic conditioning equipment is best installed after the flotation feed pump and before flotation, so that the aggregates are not subjected to pumping before flotation. Ideally ProFlotes are installed either in existing flotation conditioners or in line between the float feed pump and the float cells.

At Hellyer neither option was possible in the cleaner, though the rougher circuit did have a flotation conditioner. There was no geographical footprint available to install the ProFlote equipment in-line after the flotation feed pump in the cleaner circuit. The suggestion came from Hellyer personnel to install the ProFlote in the first cleaner cell, in effect conditioning during flotation. This was a unique installation for the ProFlote and testing this type of installation was then undertaken.

RESULTS AND DISCUSSION

Part 1 – Testing magnetic aggregation in the cleaner circuit

The first stage of the test work was testing magnetic aggregation in the cleaner circuit. The mean total plant results, for the randomised paired test work in the cleaner circuit are given in the Table 6.

TABLE 6

Mean total plant results for magnetic conditioning – cleaner installation.

	Magnetic conditioning ON	Magnetic conditioning OFF
Mean % Zn plant food	2.16	2.19
Mean % Zn final concerntrate	35.65	35.67
Mean % Zn plant tail	0.85	0.94
Mean % Zn recovery	60.59	56.96

The mean of the paired plant zinc recovery differences when magnetic conditioning was ON compared to when magnetic conditioning was OFF, was 3.63 per cent. This mean paired difference was significant at the 97.8 per cent confidence level. The mean of the paired plant zinc tail assay differences when magnetic conditioning was ON relative to magnetic conditioning being OFF was 0.09 per cent zinc. The mean of the paired difference in zinc in tail was significant at the 98.0 per cent confidence level. For the plant zinc recovery and zinc tail assay the statistical results are given in Table 7.

TABLE 7

Statistical analysis of total plant zinc recovery and tail assay results – cleaner installation.

	% Zinc recovery	% Zinc in tail
Mean magnetic conditioning ON	60.59	0.85
Mean magnetic conditioning OFF	56.96	0.94
Mean difference	3.63	0.09
Statistical T for mean paired difference	2.22	2.27

When the magnetic conditioning was applied to the cleaner feed it can be seen that the mean increase in overall plant zinc recovery was 3.63 per cent, but the mean paired difference of the final zinc concentrate grade is negligible when magnetic conditioning is ON. This result is similar to the Golden Grove result (Engelhardt, Ellis and Lumsden, 2005) on fresh sphalerite ore where magnetic conditioning resulted in an improved grade-recovery curve. The improved selectivity was confirmed with no statistically significant difference in the mean paired difference of iron in zinc concentrate. When the magnetic conditioning was ON the mean iron grade in final concentrate was 12.2 per cent, and when magnetic conditioning was OFF the mean iron grade in final concentrate was 11.9 per cent. There was no statistically significant change in any of the other metals or minerals.

Magnetic conditioning, when installed in the first cleaner cell, was effective in selectively increasing the recovery of fine sphalerite in this tailings retreatment operation. The increase in recovery was only for sphalerite, presumably because it is paramagnetic, and not for pyrite.

Part 2 – Testing magnetic aggregation in the rougher circuit

After completing the test work with the magnetic conditioners installed in the cleaner circuit the equipment was then installed in the rougher flotation conditioner.

The total plant recovery results when the magnetic aggregation was installed in the rougher flotation conditioner are in Table 8.

TABLE 8

Mean total plant results for magnetic conditioning ON and OFF – rougher installation.

	Magnetic conditioning ON	Magnetic conditioning OFF
Mean % Zn plant feed	3.51	3.55
Mean % Zn final concerntrate	36.24	36.44
Mean % Zn plant tail	1.04	1.13
Mean % Zn recovery	68.51	66.80

While the results are showing a qualitatively similar effect for magnetic conditioning in the rougher as in the cleaner, the mean paired difference in plant zinc recovery was not statistically significant to the same level of confidence. The reason is two-fold, fewer samples were collected and during this period in the plant there was much greater plant variability. A new cut was started in the tailings dam during the testing and the resulting plant feed grade and plant results were more variable. For the rougher test work, the means of the paired difference in zinc assay in the plant tail of 0.09 per cent zinc was significant at the 97 per cent confidence level. The mean of the paired difference of zinc in tail assay has a much smaller standard deviation compared to the zinc recovery, so the test work did provide sufficient data for the tail result but more data was necessary to evaluate the recovery result.

Because the overall increase in recovery when the magnetic conditioning was installed in the rougher was less than when installed in the cleaner, there seemed to be no benefit in continuing the test work in the rougher, so the equipment was returned to the cleaner.

This test result, showing more benefit in the cleaner than in the rougher is consistent with Polymetals survey data showing a much higher distribution of fine sphalerite in the cleaner feed rather than the rougher feed.

Another reason that the magnetic conditioning showed greater benefit in the cleaner rather than the rougher may well be the greater challenge of fine particle selectively in the cleaner, rather than the rougher (Lane and Richmond, 1993). Perhaps, also, another contributing factor may be the much greater flows in the rougher circuit compared to the cleaner circuit. The same magnetic conditioners were simply transferred from one circuit to another, however, since the flow is three times greater in the rougher circuit compared to the cleaner circuit, residence time in the magnetic field is only about a third in the rougher installation, compared to the cleaner circuit.

CONCLUSION

While magnetic conditioning has been shown to selectively increase fine sphalerite recovery on fresh ore, these results show that even if the sphalerite is somewhat oxidised from internment in a tailings dam for a decade, the magnetic energy of attraction is still sufficient to overcome the possibly increased electrostatic repulsion. This magnetic aggregation will selectively increasing the fine sphalerite recovery.

Also the magnetic conditioning is still effective even if it occurs during the first stage of flotation rather than in a separate stage before flotation. Installation in the Hellyer cleaner circuit was in the first flotation cell.

The greater improvement in the cleaner circuit compared to the rougher circuit, may also support the view that there is an optimum residence time in the magnetic field to maximise the effect of magnetic conditioning. Or the difference between rougher and cleaner results may simply reflect the historical and current metallurgical challenges, which have shown that selective recovery of fine sphalerite at Hellyer is primarily a cleaner challenge, or the fact that the sphalerite in the cleaner is much finer than in the rougher.

But perhaps the most important conclusion is that since magnetic aggregation improved sphalerite flotation in the rougher and cleaner then there would be a cumulative improvement by installing the equipment in both circuits simultaneously.

When strong metal demand returns and metal prices increase the Hellyer tailings dam will no doubt be retreated. The use of magnetic conditioning will enhance the economic returns for the retreatment operation.

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