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Improving Fine Lead and Silver Flotation Recovery at BHP-Billiton's Cannington Mine

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ABSTRACT

BHP-Billiton's Cannington mine in Northwest Queensland produces a silver-rich lead concentrate and a zinc concentrate. The plant has the capacity to treat 3.1 mta and for the year to June 2006 mean feed grade was 10.3% Pb, 461 g/t Ag and 3.7% Zn. Recovery losses of lead and silver are primarily in the fine fractions. In 2001, the plant was modified when it was determined that splitting the lead feed into two size fractions, and floating the two fractions separately increased fine lead and silver recovery. The fine lead fraction (called the split float) is 80% <16µm and about a quarter of the lead and silver is in the finest fractions. In 2006 to further improve lead and silver recovery a statistical plant trial of magnetic aggregation conditioning of the split float feed was undertaken. While pure galena is diamagnetic the Cannington galena contains iron within the galena mineral resulting in the galena having a paramagnetic, magnetic susceptibility. The trial demonstrated to greater than 98% statistical significance an increase in lead recovery, and to 99% statistical significance an increase in silver recovery, and to greater than 97% significance an increase in the lead in concentrate in the split float circuit. These economic recovery benefits with magnetic aggregation of the split float feed, has led to the incorporation of this technology in the plant.

INTRODUCTION

The BHP-Billiton owned Cannington silver, lead and zinc mine is located in the Australian state of Queensland about 800 kilometres west of the coastal city of Townsville. This is a pastoral area characterised by large cattle farms, low population density and very little infrastructure. The area has two seasons, a wet, hot summer and a dry, mild winter. Because of its isolation and lack of infrastructure the mine has always operated on a fly in fly out basis.

Geological drilling defined the resource as 43.8 million tonnes grading 11.6% lead, 4.4% zinc and 538 ppm silver (Walters and Bailey, 1998). At times during its history Cannington has held the crown as the single largest lead and silver mine in the world (Torrisi and Smith 2003). Another unique characteristic of the operation's history is the rapid progress from discovery to operation. The first drill to intersect the ore body was in 1990 and the mine and plant was commissioned in 1997.

The history of the Cannington mineral processing plant has been one of ongoing and continual process improvements. Table 1 shows some of the operational statistics for 1998, the first full year of design operation, and the current data for the year to June 2006. The increase in plant performance and quality and quantity of product over those 7 years is substantial. These changes have come about through Cannington's dedicated program of targeting and solving processing problems. A steady increase in plant throughput has been seen since start-up due to plant upgrades and a number of capital projects. A reduction in plant throughput occurred in FY06-07 due to an extensive mine rehabilitation program which was undertaken to improve ground stability and employee safety. It is expected that processing rates will return to normal levels during FY07-08.

Table 1: Cannington Plant Production

Year	Feed (wmt)	Pb Conc (wmt)	Zn Conc (wmt)
FY98-99*	1,426,823	220,173	96,515
FY05-06	3,114,153	430,458	182,674

*FY is the Australian Financial year, July 1 to June 30.

Mineralogy

The Cannington deposit is a silver rich Broken Hill type deposit. The sulphide minerals are galena, sphalerite and pyrrhotite with little pyrite present. The gangue minerals are either silicious rich or iron rich. The ore body is divided into ten different ore types, based on the gangue mineralisation and the Pb:Zn ratio.

In the iron-rich zones the galena and sphalerite are mainly associated with pyrrhotite, magnetite and fluorite (Walters and Bailey, 1998), whereas in the silicious zone there is little iron mineralisation and quartz is the main gangue mineral.

The Cannington sphalerite is an iron rich zinc sulphide with an average iron content of 8.8% iron in the sphalerite mineral (French et al, 1994). The sphalerite can also have chalcopyrite inclusions.

The galena is generally coarse grained, and contains on average 1200ppm silver, around 1300ppm antimony and from 400 ppm to 4000 ppm iron (Walters, 1994). While the iron content of the galena averages 400-4000 ppm, depending on the source, in some areas the iron content can be over 10,000 ppm (French et al 1994).

The primary economic deportment of silver is either contained in solid solution in the galena, or as the mineral freibergite, a copper, antimony, iron and silver sulphide (French et al, 1994). The freibergite occurs as anhedral inclusions in the galena (Walters and Bailey, 1998). The freibergite is on average 32% silver, 27% antimony and 5% iron. There are also small amounts as complex silver sulfides and as antimonial silver.

The freibergite and galena are generally found together with galena-freibergite composites common.

The 2005 mill feed mineral composition is given in Table 2.

Table 2: Mineral Composition in Cannington Feed

Mineral	Composition %
Galena	11.7
Sphalerite	5.9
Pyrrhotite	2.2
Pyrite	0.8
Other Sulfides	0.3
Fluorite	5.9
Talc	1.5
Quartz	29.5
Magnetite	7.3
OtherNSG	34.9

Mining

The Cannington mine is an underground mine that uses the open stoping method. Ore is trucked and hoisted to the surface where it is stockpiled for treatment. Stockpiling allows for ore blending of the different ore types to minimise plant feed variability.

Ore Processing

An excellent and detailed review of the history, expansions and modifications of the Cannington mineral processing operation is published by Alford and Clarke (2007). In general ore processing follows the conventional flow sheet for lead and zinc sulphide ore bodies. Processing includes grinding, sequential flotation to produce a silver rich lead concentrate and then a zinc concentrate, fluoride leaching and dewatering. The concentrates are trucked to the railhead near Cloncurry and then by train to Townsville for shipping to the smelters, mainly in Japan.

Grinding

The current grinding circuit is an AG mill followed by a vertimill both in closed circuits with cyclones. The oversized material from the mill discharge can be directed to a pebble crusher for further size reduction or sent directly back into the mill. The circuit is shown in Figure 1. The original target p80 was 90 microns, however due to increases in feed rates the p80 exceeds this size. The current p80 averages between 100 – 120 microns with a continued focus being placed upon reducing this further.

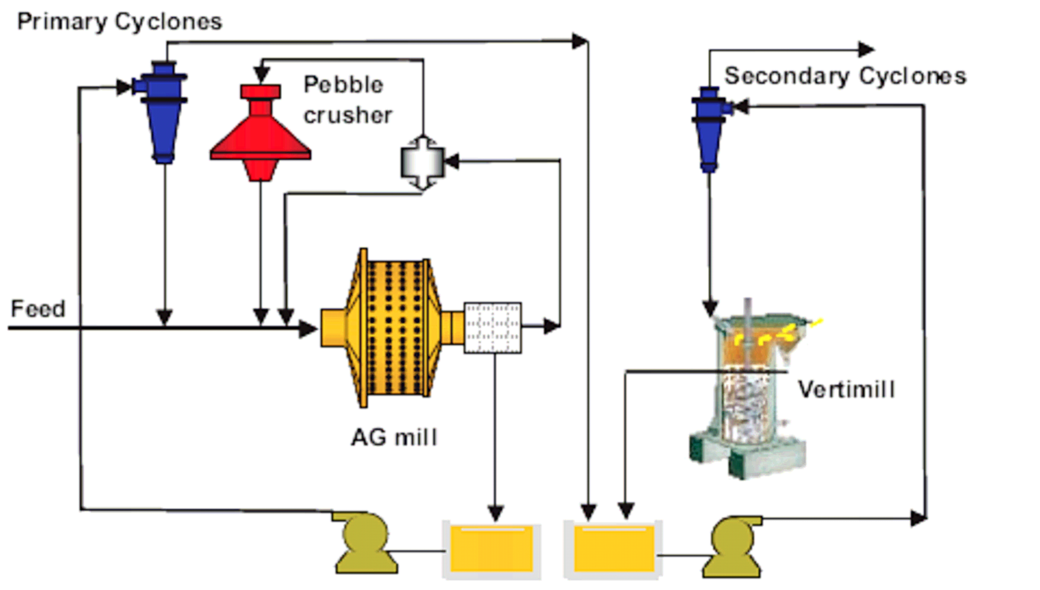

Figure 1: Cannington Grinding Circuit

Flotation

The plant initially used a pre-float circuit to remove fluorine-bearing talc but this is rarely used at present due to the low talc levels in the ore. The sulphide flotation circuit consist of three circuits, a fine lead flotation circuit (called the split float circuit) a lead flotation circuit and a zinc flotation circuit. Each of the three flotation circuits has its own multistage cleaning circuit. The flotation circuit is shown in Figure 2. The overflow from the secondary cyclone classification that is undertaken in the grinding circuit is pumped to another bank of cyclones where the cyclone overflow (P80 of 16 µm) reports to the split float circuit. The underflow reports to the lead flotation circuit. The split float tails returns to the lead flotation feed and the split float cleaned concentrate combines with the lead concentrate to make a combined lead concentrate. The lead circuit tail is zinc circuit feed and the zinc circuit tail is final tail.

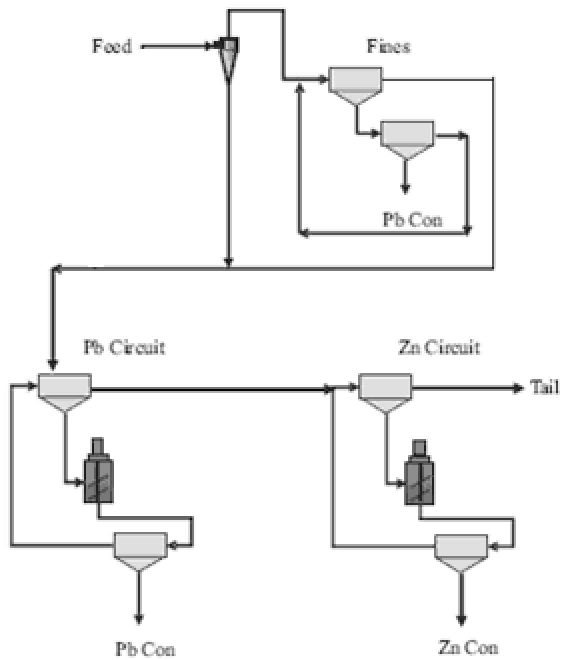

Figure 2: Outline of the Cannington Flotation Circuit

Table 3 shows a simplified flotation circuit mass balance looking at the combined split float and lead flotation circuit concentrate and the zinc concentrate grades and recoveries for FY05-06 (Alford and Clarke, 2007).

Table 3: Metallurgical Grades and Recoveries

Stream		Grades			Recoveries %		
	P80 µm	Ag (ppm)	Pb %	Zn %	Ag %	Pb %	Zn %
Feed	104	500	10.0	3.5			
Pb Conc	19	3400	70.0	4.2	86.5	88.9	15.2
Zn Conc	19	310	4.2	49.0	3.1	2.1	70.8
Tail		63	1.09	0.60			

The reagents used throughout the Cannington flotation circuits are common to most lead/zinc mines in Australia.

Methyl Iso-Butyl Carbonal (MIBC) is the standard frother used in all three circuits, however it is usually only required at the conditioning stages of the split float and lead circuits.

Sodium Ethyl Xanthate (SEX) is the general collector reagent used in all three circuits. Aerophine 3418A is also used in the split float and lead circuits as a silver/lead selective collector.

Sodium Meta-bisulphite is used in the split float and lead circuit cleaners as a sulphide depressant with the aim of reducing zinc and iron mineral recovery to the lead concentrate. Copper sulphate is then required to re-activate the zinc minerals in the zinc circuit.

Sulphuric acid and aluminium sulphate are used in both leaching circuits to remove fluorine from the final concentrates and control pH in the leaching circuits. Lime is used in the process water to maintain the required slurry pH throughout the flotation circuit which has been found to be vital in both lead and zinc recoveries.

The reagent consumptions for FY05-06 are given by Alford and Clarke (2007) and are in Table 4.

Table 4: The Flotation Circuit Reagent Consumption for FY05-06.

Section	Reagent	Dosage (g/t)
Total Circuit	MIBC	15
	Lime	300
	Acid	3200
	Al_2O_3	700
Pb Flotation (including spilt float)	SEX	150
	3418A	30
	MBS	150
Zn Flotation	SEX	60
	$CuSO_4$	70

Concentrate Treatment

The high fluoride in the concentrate is reduced by a fluoride leach of the concentrates. Aluminium sulfate is used to leach the calcium fluoride from both the lead and the zinc concentrate. This leaching is carried out at a pH of 3.5 and at greater than or equal to 50 degrees centigrade.

The leached concentrate is then thickened, pressure filtered and stored before shipment.

The Split Float Circuit

The lead split float circuit was the outcome of a detailed research program by Cannington to reduce the valuable mineral losses from the circuit. The underlying research, the design of the circuit and the installation and commissioning of the circuit is well detailed by Torrisi and Smith (2003).

In summary, Torrisi and Smith (2003) show that the historical data from the Cannington plant had shown that over 50% of the lead and silver losses in tails were lead and silver in the <5 µm fraction, and that recovery of this ultra fine liberated galena and silver was a problem in many silver-lead flotation circuits. The problem of losses of ultrafine lead and silver was a major revenue drain on the project that was critical at a time of historically low metal prices. Testwork had shown that the low recovery of the ultrafine particles was due to their low flotation kinetics. Splitting the fine mineral and floating this stream separately at a pH of around 5 significantly increased the flotation kinetics and therefore the recovery of the ultrafine lead and silver. The research led to the design of a split float lead-silver flotation circuit.

While the circuit implementation was successful, the dual pH operating strategy encountered operational problems in the rest of the plant. This included poorer flotation of the coarse lead when the split float tail recombined with the lead flotation feed, and increased sphalerite flotation in the lead circuit. The final outcome was to maintain the split float circuit to float fine lead and silver, but operate this circuit at the natural pH. In effect the split float circuit increased the flotation residence time for fine galena and silver flotation.

Initially the feed to the split flotation circuit was targeted at an F80 of 8 micron and was achieved using three stages of particle size classification. In recent years however only one stage of classification has been used with the F80 seen to increase to <16 micron. Analysis has found that there has been no loss of fine lead or silver due to this change.

An assessment of the impact of the split float circuit installation was that overall lead recovery in the <9µm fraction increased by 7-9%, lead grade in final lead concentrate by 4-6% and silver recovery in the lead concentrate by 2-4%. This was a major economic benefit to Cannington.

The percentage of feed reporting to the split float circuit is strongly dependant on the mill feed rate and the flotation feed size. It can vary from 25% at 300 tph mill feed to 15% at 450 tph mill feed. On average during the trial and under normal operating conditions (400 tph mill feed) approximately 20% or 80 tph of the mill feed reports to the split flotation circuit.

The split flotation reagent consumption in grams per tonne of circuit feed is significantly higher than the main lead flotation circuit due to the slower flotation kinetics of the finer particles.

Lead Zinc and Silver Losses at Cannington

Torrisi and Smith (2003) showed that even after the introduction of the split float circuit and the extra residence time for fines flotation recovery of lead and silver in the finest fraction was still only 78.7% for lead and 75.5% for silver. Yet most of the fine sulfides were liberated. The only reason for their low recovery was the slow flotation kinetics of the minerals. The slow flotation kinetics was a result of their size.

Due to the fine nature of these particles it has always been very difficult to maximise their recovery. The split flotation circuit has seen significant improvements in the finer lead size fractions as discussed earlier however it is still the –C5 fraction that poses the most difficulty to recover due to the required residence time necessary to achieve flotation.

Magnetic Aggregation of Minerals.

The aggregation of magnetic minerals in a magnetic field has been studied extensively. The application of magnetic aggregation to paramagnetic minerals has been reported in the literature since the 1970's. An excellent discussion and review of the magnetic aggregation of paramagnetic minerals is published by Svoboda (1987). It details the particular conditions under which magnetic aggregation of paramagnetic particles will occur. Svoboda (1987) shows the conditions of particle size and solution chemistry under which magnetic aggregation will occur are met in flotation feed stream. Some examples of the application of magnetic aggregation as a conditioning step in the flotation of fine sulphide minerals was published by Engelhardt et al (2005). Their work showed that the flotation recovery of <38um paramagnetic mineral, and more particularly <20um paramagnetic mineral could be significantly improved by prior conditioning in a magnetic field. Rivett et al (2007) also showed that this technology was applicable to <38um paramagnetic minerals especially <20um minerals. Interestingly, Rivett et al (2007) showed that even gold, a diamagnetic mineral was effected by magnetic conditioning, with an increase in flotation recovery of <20um gold. The possible explanations for the increased recovery of diamagnetic gold was discussed by Rivett et al (2007), with reference to the quite extensive literature showing that gold depending on its mineralogical deportment could be recovered and concentrated magnetically.

Both Engelhardt et al (2005) and Rivett et al (2007) show that the use of magnetic aggregation is mineral specific, and depends on the magnetic susceptibility of the minerals. Gaudin and Rush-Spedden (1943) shows that the magnetic susceptibility of sulphide minerals is not simple. There magnetic susceptibility is dependent on the minerals their associations and inclusions. This is particularly important in sulphide ore bodies with significant iron present.

Gaudin and Rush-Spedden (1943) carried out magnetic separation studies on a number of sulphide minerals from existing operations and found that galena was completely non-magnetic, except when composited with magnetic minerals, but found that freibergite was as strongly paramagnetic as chalcopyrite. Using a concentrate sample from the Sunshine mill in Idaho they were able to separate galena into the non-magnetic fraction, and the freibergite into the magnetic fraction. Gaudin and Rush-Spedden (1943) also found in the case of concentrate from the Sullivan concentrate in British Columbia that some of the <38 μm galena reported to the magnetic fractions but this was locked with magnetic minerals. For sphalerite Gaudin and Rush-Spedden (1943) found that it reported to the magnetic fraction if it had magnetic inclusions. These magnetic inclusions were very fine and dispersed and hard to identify.

The magnetic aggregation equipment for flotation conditioning is being marketed by Ausmetec of Australia and Dorr-Oliver Eimco under the name ProFlote. It consists of a series of high strength magnetic permanent magnetic cores in a stainless steel shell, the equipment is designed so that the magnetic material in the slurry does not blind the magnets. The magnetic aggregation devices are installed as a flotation conditioning step.

With Cannington's continued commitment to processing improvements and to maximise its financial returns a decision was made to evaluate magnetic aggregation in the plant to improve the recovery of fine mineral.

RESULTS AND DISCUSSION

Magnetic Aggregation on the Zinc Circuit at Cannington.

Sphalerite that has iron in the mineral matrix even at only a few percent is known to be paramagnetic (Svoboda, 1987). The Cannington sphalerite with 8.8% iron in the mineral was tested and as expected found to be strongly paramagnetic. Therefore, the initial proposal was to test the magnetic aggregation technology in the zinc circuit. The zinc cleaner circuit after regrinding was the best installation position because in this stream all the sphalerite was fine. The results of the tests were a surprise. The statistical On/Off test work carried out over about 2 months with 106 data points showed a statistically significant increase in lead and silver recovery in the zinc circuit. The results are normalised and given in Table 5.

There was no significant increase in zinc recovery with magnetic aggregation.

As researchers these results were a puzzle. Why is the galena affected by this process, yet the sphalerite not affected?

Table 5: Results of Magnetic Conditioning on the Zinc Circuit

Normalised Results	Magnetic Aggregation ON	Magnetic Aggregation OFF	Difference	T Test Statistical Significance
Pb Recovery in Zinc Circuit	110.2	100	10.2	>99%
Ag Recovery in Zinc Circuit	110.6	100	10.6	>99%
Zn Recovery in Zinc Circuit	100.3	100	0.3	NS

A composite Cannington galena sample was collected and its magnetic susceptibility was measured. The galena was found to be strongly paramagnetic. Table 6 compares the measured magnetic susceptibility of the Cannington galena with some other common paramagnetic minerals found in Svoboda (1987).

Table 6: Magnetic Susceptibility of Paramagnetic Minerals and Cannington Galena

Mineral	Magnetic Susceptibility (m^3kg^{-1})x10^{-9}
Chalcopyrite	1596
Bornite	101
Cassiterite	2136
Cannington Galena	820

This measurement showed the galena to be strongly paramagnetic. This was a surprising result given the literature's reference to galena being diamagnetic (Svoboda, 1987). However, others have also reported paramagnetic galena. Shvets and Shaposhnikov (1971) found that some galena from the Krivoi Rog basin was paramagnetic with a magnetic susceptibility as high as 21 (m^3kg^{-1}) $x10^{-9}$. They explain that the paramagnetism of the galena as due to the presence of iron and manganese in the galena crystal. The Cannington galena has a substantially higher magnetic susceptibility than that reported by Shvets and Shaposhnikov (1971), but the iron content of the Cannington galena is relatively high. The high magnetic susceptibility of the Cannington galena may be due to the form of the iron in the galena. It is possible that the iron is present as magnetite that has a magnetic susceptibility some 1000 to 100,000 times that of paramagnetic minerals, so that even a 1% or 0.1% magnetite in the galena could account for the high magnetic susceptibility of the Cannington galena. Alternatively, the iron could be present as a ferromagnetic iron sulphide, or as native iron, though native iron is less likely. Iron could also be substituted for lead in the galena crystal. It is known that iron substitution for zinc in sphalerite even at a few percent creates a strongly paramagnetic sphalerite mineral. The iron content in the galena at Cannington averages less than 1 percent and so from the sphalerite comparison, substitution alone would not be expected to give the strong magnetic susceptibility measured, however, the difference may be due to the differences in structure between galena and sphalerite and so the substituted iron could affect the magnetic susceptibility in a more significant way. There are then a number of alternate explanations for the unusual paramagnetism of the Cannington galena. Investigation is ongoing to identify the reason for this very interesting and unusual characteristic of the Cannington galena.

The next stage of testing was to test the ProFlote in the split float circuit.

Testing Magnetic Aggregation Conditioning on the Lead Split Float Circuit.

The magnetic units were installed in the split float conditioning tanks just prior to the split float flotation cells. The slurry, gravity feeds from the tanks into the flotation bank.

The test was run on a 3 day on 3 day off cycle. Only when there was a shutdown or other known plant disruptions in the grinding circuit was data eliminated from the data set. The trial was run over about 6 months, and resulted in around 150 sets of data. The split float circuit has an excellent automatic sampling system and so the shift composite samples were used. At other sites (Rivett et al 2007) the shift samples were sized to measure the effect of the ProFlote on the fine size fraction. However, the split float feed is already a finely sized stream and so sizing of the shift samples was not carried out.

Lead Results

The results show a statistically significant increase in lead recovery with magnetic aggregation conditioning. As expected there was no difference in the lead in feed but there was a significant difference in lead in concentrate, lead in tail and lead recovery. The results and their statistical confidence are shown in Table 7.

Table 7: Lead Results From the Lead Split Float Circuit

Normalised Results	% Pb Feed	% Pb Conc	% Pb Tail	% Pb Rec
ON	100.2	101.3	81.8	104.0
OFF	100	100	100	100
Difference	0.2	1.3	-18.2	4.0
Statistical Confidence	None	97.5%	>99.9%	99.85%

The normalised 4% increase in lead recovery, and normalised 1.3% increase in lead grade is a very good result. It should be noted that the much larger, more complex, much more capital expensive (approximately AUD$10 million capital cost) and more higher operating cost split float circuit investment delivered only 2-3 times this benefit in recovery and 4-6 times the grade benefit. However, since the ProFlote installation was not a large capital investment the investment risk was also very small. This research and test work has reduced the lead losses in the split float tail by around 18%. The large increases in recovery meant that the results were statistically significant to a very high level of confidence.

These results are consistent with the increased recovery of lead that was measured when magnetic aggregation was applied in the zinc circuit. It is very interesting to note that not only was there an increase in lead and silver recovery but there was a concurrent increase in lead grade. This clearly shows a different grade recovery curve when magnetic aggregation is used and an increase in selectivity.

Silver Results

The results show a statistically significant increase in silver recovery with magnetic aggregation conditioning. As expected there was no difference in the silver in feed but a significant difference in silver in tail and silver recovery. The normalised results are summarized in Table 8.

The increase in silver recovery normalised is 3.5%, to a very high level of statistical confidence. This increase in silver is comparable with the increase in silver recovery from the installation of the split float circuit.

Table 8: Silver Results From the Lead Split Float Circuit

	Ag Feed ppm	Ag Conc ppm	Ag Tail ppm	% Ag Rec
ON	99.3	100.3	83.9	103.5
OFF	100	100	100	100
Difference	-0.7	0.3	-16.1	3.5
Statistical Confidence	None	None	>99.9%	99.5%

The results are also consistent with the magnetic aggregation results that were achieved previously in the zinc circuit.

There was no significant increase in zinc recovery in the split float circuit with the magnetic aggregation technology.

CONCLUSION

Cannington's commitment to process improvement since start-up has resulted in significant advances in the plant's mineral processing. The current high metal prices mean these improvements are delivering substantial benefits to shareholders. The latest instalment in this process improvement campaign is the use of the ProFlote magnetic aggregation technology to increase the recovery of fine lead and silver. With about a quarter of the lead and silver in the split float feed the improvement in split float lead recovery of 4.0% normalised and silver recovery of 3.5% normalised demonstrate a further improvement in the overall metallurgical performance at the Cannington Plant. The ProFlote has substantially increased fine lead and silver recovery to give further improved returns to BHP-Billiton shareholders.

Of the three valuable metals in the Cannington ore, silver, lead and zinc, silver is by far the most valuable revenue stream to Cannington. Even at today's very high metal prices the dollar value of silver in feed is about 40% higher than lead and about 3 times the value of zinc in feed. The major project to install the split float gave substantial financial benefits to the Cannington project. The relatively minor, low-risk project of installing the ProFlote in the split float feed conditioners has delivered a similar benefit in terms of silver recovery, the most valuable product from Cannington as installing the split float circuit, a major upgrade. Its return on lead recovery and grade is lower than the split float project but substantial nevertheless.

Because the ProFlote project was so low-risk, the research required and the economic justification to test the technology was relatively minor compared with the major commitment require for the split float project.

This work at BHP-Billiton's Cannington mine demonstrates that new technology can short cut the pathway to improved financial returns often at lower risk and lower inputs.

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