

Developments in Milling Practice at the Lead/Zinc Concentrator of Mount Isa Mines Limited from 1990.

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ABSTRACT

The Lead/Zinc Concentrator of Mount Isa Mines Limited processes complex fine grained ore from the Isa and Hilton silver-lead-zinc orebodies, producing lead concentrate, zinc concentrate and (until 1996) a low grade middlings (LGM or bulk) concentrate.

Metallurgical performance declined dramatically during the 1980's because of declining ore quality, as the ore became both finer grained, and contained increasing amounts of refractory pyrite.

Developments in milling practice to restore performance focused on two areas: liberation and separation. Increased mineral liberation was achieved by more than doubling grinding and regrinding capacity to increase sphalerite liberation. This successfully recovered an extra 20 per cent zinc metal to zinc concentrate, which previously reported to final tailing, lead concentrate or LGM concentrate. There was also a small increase in galena liberation, increasing lead recovery to lead concentrate and reducing contamination of the zinc concentrate by lead. The increased sphalerite and galena liberation also significantly reduced the production of LGM concentrate.

The second area of development was to improve the separation of galena and sphalerite from gangue minerals. This was achieved by circuit rationalisation, better understanding of water chemistry leading to an improved reagent scheme, and improved process control. These changes both improved performance and simplified the circuit, giving better and steadier concentrate grades and recoveries.

The combination of increased liberation, improved separation, and circuit simplification has dramatically increased the metallurgical performance of the Lead/Zinc Concentrator when treating very complex fine grained ore.

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INTRODUCTION

Processing of Mt Isa silver-lead-zinc ore commenced in 1931 at the No.1 Concentrator, initially treating a mixture of oxidised and sulphide ore, but by 1935 treating only sulphide ore (Kruttschnitt et al, 1939). Over the years the flowsheet was developed to improve metallurgical performance of the fine grained and difficult to treat Mount Isa ore (Kruttschnitt et al, 1939, Cunningham, 1953, Challen et al, 1968, Davey and Slaughter, 1970). Over the period 1952 to 1960, ore reserves were increased substantially, and the decision was made to increase the treatment rate, firstly by modernising and expanding the existing No.1 Concentrator, and secondly by constructing a new (No. 2) Lead/Zinc Concentrator (Challen et al, 1968).

The No 2 Concentrator was commissioned in June 1966 and total silver-lead-zinc ore treatment was transferred from the No. 1 Concentrator in May 1967. Various improvements in the 1970's (Bartrum et al, 1977) were followed by the installation of larger flotation cells in a single circuit conversion (Johnson et al, 1982). A major increase in throughput occurred in 1982 with the commissioning of a Heavy Medium Preconcentration plant to reject 30 per cent of waste ore before flotation (Fiedler et al, 1984). From 1987, ore from the nearby Hilton mine was introduced to supplement the Mt Isa ore. By 1992, treatment rate had reached 5 Mt/y, of which 30 per cent was from the Hilton Mine and 70 per cent from Isa Mine.

During the 1980's, the increase in throughput and decline in head grade were exacerbated by a significant increase in ore complexity. This resulted in a severe liberation problem, with a finer mineral liberation grain-size than the plant grinding capacity could achieve, and a worsening separation problem caused by increasing amounts of naturally floating carbonaceous pyrite. Metallurgical performance declined dramatically as the plant did not have technologies to deal with either problem. This paper chronicles the change in ore characteristics, along with the highly successful technological changes to return good metallurgical performance with the more complex ore.

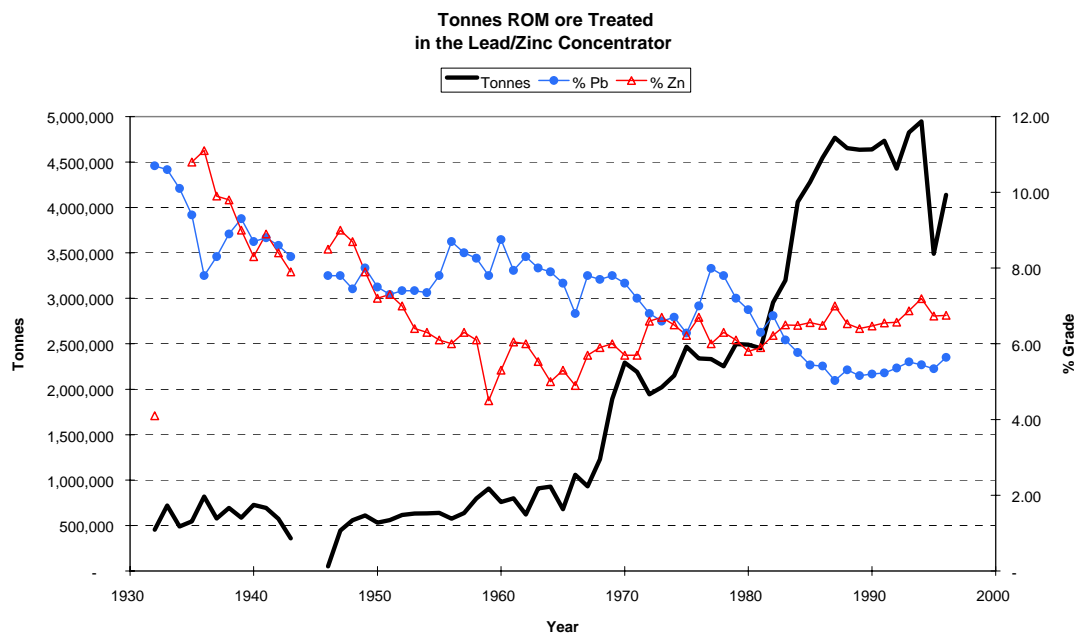


Figure 1.

The history of processing silver-lead-zinc ore at Mount Isa Mines has been one of steadily increasing tonnage and declining head grades, as shown in Figure 1. This was exacerbated by increasing complexity of ore.

MINERALOGY

The mineralogy of the silver-lead-zinc orebodies can be characterised as fine intergrowths of galena and sphalerite with both sulphide and non-sulphide gangue. The main silver material is freibergite which is intimately associated with the galena. The non-sulphide gangue is quartz, dolomite and carbonaceous

shale. Minor amounts of chalcopyrite are present. The sulphide gangue is both pyrite and pyrrhotite, with pyrite predominant. Pyrite is present as two distinct types; firstly, as normal relatively coarse grained euhedral pyrite, and secondly, as fine grained (5 to 30µm) spheroidal pyrite. This second type is refractory and contains elemental carbon, sometimes forming atoll rims on galena. The carbonaceous pyrite is hydrophobic under almost any conditions and is the dominant sulphide diluent in both lead and zinc concentrates (Davey et al, 1970, Munro, 1993).

Feed to the Lead/Zinc Concentrator consists of both Isa and Hilton ores. Isa ores can be classified into two broad categories :

- The upper Isa orebodies, No's 1,2 and 5, referred to as "Black Star" orebodies. They are more massive and are mined by open stoping at lower mining cost, and therefore have a lower cut-off grade. Generally this ore is finer grained and has considerably more fine-grained carbonaceous pyrite, whilst core replacement of pyrite by galena (atolling) is more common and at a more advanced stage (Davey et al, 1970).
- The lower Isa orebodies, No's 7 to 14 and Rio Grande are referred to as "Racecourse" orebodies. These are more narrow and mined by bench stoping with a higher cut-off grade. Generally these orebodies are higher in grade for lead and silver, coarser grained and lower in pyrite. The pyrite is more the euhedral type than the fine-grained carbonaceous type. Though this ore has a higher mining cost per tonne, it is the more profitable ore since it has the highest grade and the best metallurgy. The gradual displacement of this ore by Black Star and Hilton ores is the reason for the continual decline in ore quality experienced in the Concentrator.

The "Racecourse" and "Black Star" categories are used to describe the mineral types in the ore and its metallurgical performance, as well as its geological location.

Hilton ore has been treated through the Isa Lead/Zinc Concentrator since 1987 and is divided into two similar categories. The upper orebodies, No's 1,2 and 3 ("Black Star" type ore), are more massive in size and thus allow for open stoping, contain more fine grained, naturally floating carbonaceous pyrite than the lower orebodies and contain more pyrrhotite than Isa orebodies. The lower orebodies, No's 4 to 7 ("Racecourse" type ore), are narrower and deeper orebodies, mined by bench stoping, with more euhedral pyrite and non-sulphide gangue dilution than the upper orebodies. The silver minerals in the Hilton orebodies are less associated with galena than in the Isa orebodies, hence silver recovery to lead concentrate is lower than from the Isa orebodies. Hilton orebodies also contain a wider range of silver minerals.

The two microphotographs show the difference in complexity between coarse grained, high grade ore (Figure 2) and the fine grained ore with refractory fine pyrite dilution (Figure 3) (Riley and McKay, 1976). Both of these samples were taken from the No 5 orebody at Mt Isa. Over the years, as the tonnage has increased and the head grade declined, more of the ore feeding the Concentrator has been of the Figure 3 type and less has been of Figure 2 type.

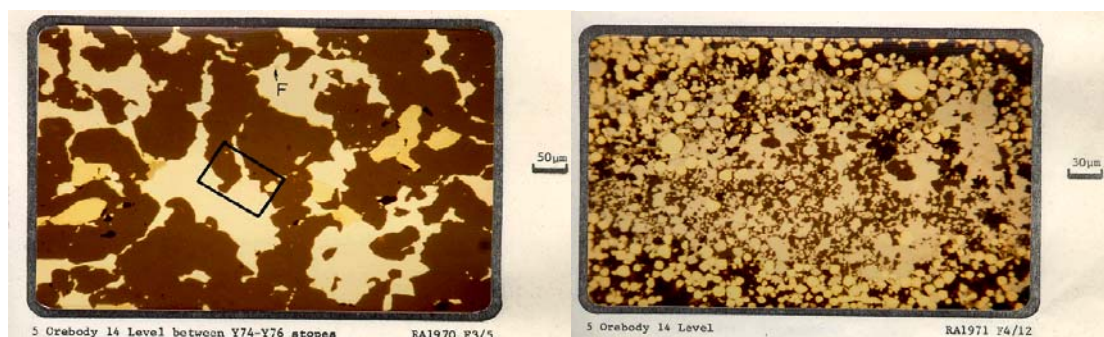


Figure 2.

Figure 3.

Microphotographs of Mount Isa No 5 orebody showing the different grain sizes and complexities that occur in the orebodies at Mount Isa (Riley and McKay, 1976).

The mineralogy at Mount Isa presents two distinct problems that affect metallurgical performance – achieving adequate mineral liberation during grinding, followed by separation in flotation. Clearly the ore in Figure 3 needs much finer grinding to achieve equivalent mineral liberation. While the main separation problem is the increase in refractory pyrite, finer grinding to solve the liberation problem increases the difficulty of separation.

Ore Type Performance

The impact of the more difficult separation because of ore type on flotation performance is demonstrated by Figures 4 and 5 : at the same grind size and ideal laboratory conditions, lead performance can vary from 60 per cent Pb concentrate grade at 90 per cent recovery (characteristic of the best “Racecourse” ore) to 15 per cent Pb concentrate grade at 50 per cent recovery (characteristic of the worst “Black Star” ore) (Figure 4). Similarly, at the same (fine) grind size, zinc recovery at target concentrate grade can vary by 20 per cent (Figure 5).

The laboratory flotation tests shown in Figures 4 and 5 were conducted to evaluate ores using a laboratory flowsheet similar to current plant operation. Crushed ore was initially ground to P80 = 37 μm , followed by a lead rougher, rougher concentrate regrinding to P80 = 15 μm and final lead concentrate production by three stages of dilution cleaning. Lead rougher tailing and first cleaner tailing were combined to feed the zinc rougher, with zinc rougher concentrate regrinding to P80 = 15 μm and final zinc concentrate production by three stages of dilution cleaning. The flotation tests were conducted with a very fine regrind size (15 μm) to maximise liberation, since less regrinding gives less liberation, and hence both lower zinc recoveries and more zinc contamination of the lead concentrate.

Lead circuit laboratory flotation performance (Figure 4) varies widely for different ore types depending on the refractory nature of the ore and the type and content of the iron sulphides. Different ore types yield significantly different performance. Since different ore types are being mined at any one time from many sources, so the mixture of ore types feeding the concentrator is continuously changing. This causes the performance of the plant to be continuously changing in the absence of intervention by the control room operator.

This leads to two effects in the short term operation of the concentrator:

- there is a change in performance and the operator is unable to tell if it is an ore change or another input change (eg: reagents, mechanical failure, uncontrolled input); and
- the operator makes a controlled change and the performance changes in an expected or unexpected manner. Was the effect from the operator's change or an ore change ?

These issues also need to be addressed when trying to improve plant performance.

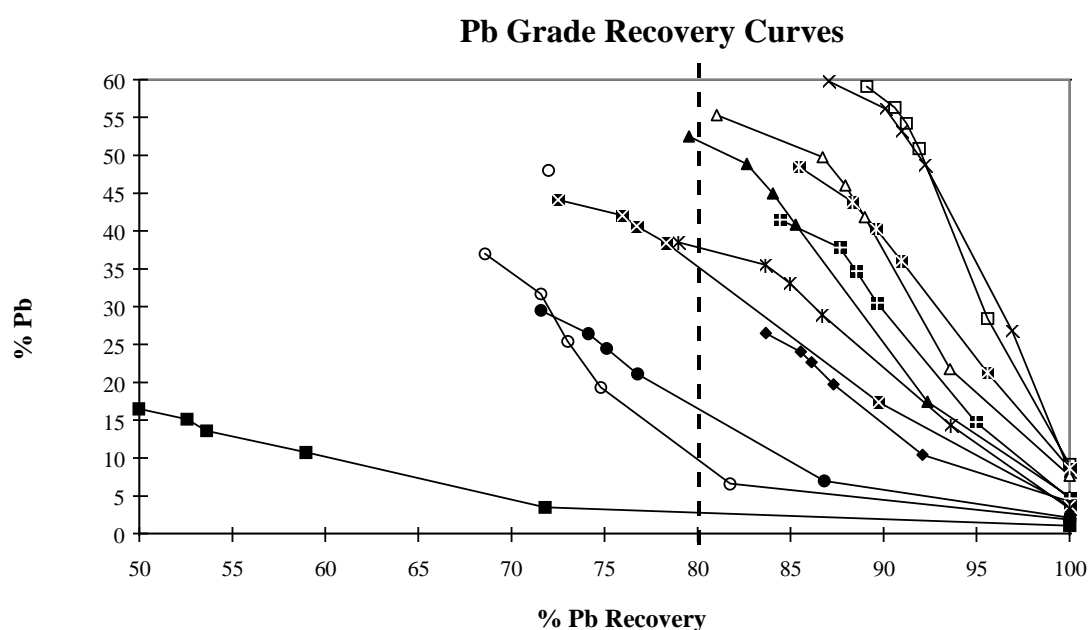


Figure 4 - Lead circuit laboratory flotation performance of different ore types.

Zn Grade Recovery Curves wrt Plant Feed

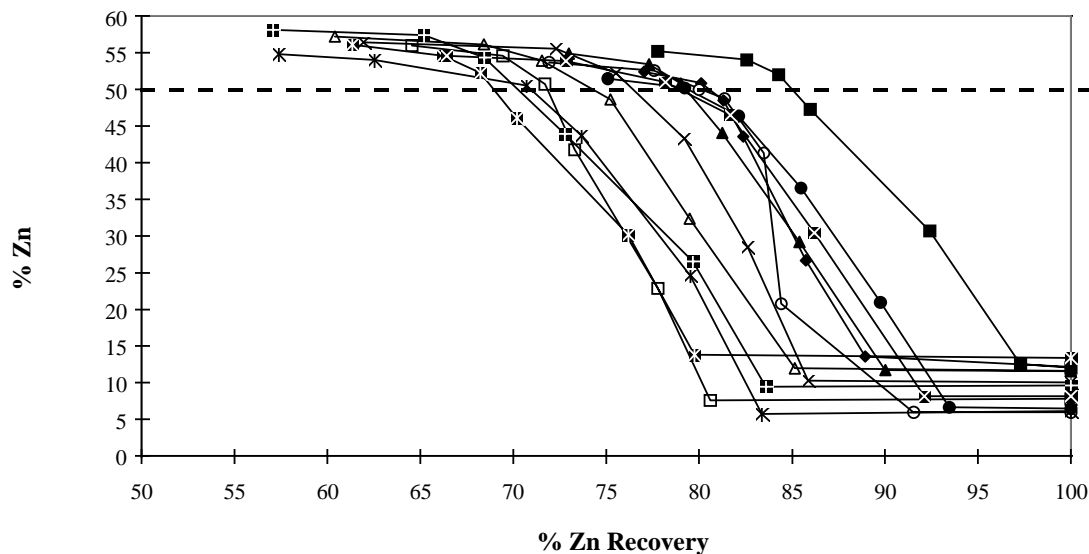


Figure 5 - Zinc circuit laboratory flotation performance of different ore types.

Zinc metallurgical performance (Figure 5) varies over a smaller band, due to low zinc losses in the lead circuit and the ability to be more selective against pyrite.

The methods used in the laboratory have shown good agreement with plant performance and provide confidence in using laboratory testwork to predict plant performance.

A SCIENTIFIC APPROACH TO THE PROBLEM

The rapidly deteriorating metallurgical performance in the 1980's was attributed to continual changes in ore mineralogy. Until the nature of these changes were fully understood, the response consisted of an endless circle of circuit changes, reagent changes, operator changes, metallurgist changes and so on. Fortunately, this was a brief, (though tense) period. It was clear that the solutions could only come from a rigorous scientific approach based on the mineralogy.

Fortunately, good scientific tools were in place to understand the nature of the changes. The two fundamental tools were size-by-size mineralogical analysis and liberation analysis. These two tools were applied to routine plant inventory samples, detailed plant surveys and laboratory and pilot plant testwork. The data combined to provide a unique mineralogical profile of plant performance, which captured both the decade-long decline in performance, and the results of the step changes in improvement.

Routine Analysis of Plant Inventory Samples

Inventory samples of plant products are taken and assayed every shift for metallurgical accounting purposes. Great care has been taken with the design and operation of inventory samplers to ensure there is no size or assay bias. Shift samples are combined into weighted daily composites, which are further combined into weighted monthly composites. In addition to chemical assays, the monthly composite samples are subjected to the following analyses:

- Screen sizing to 37 μm , followed by fine sizing (infrasizing until 1992/93, then cyclosizing after 1992/93. The cyclonizer part of the sizing is extended to C7 by a precyclone, followed by collection of the normal C1 to C5 cyclosizer fractions and then to C6 by a centrifuging of the minus C5 fraction. This procedure allows extension of the size analysis to finer sizes, as well as collection of the finest sample. The C6 fraction is typically 4 to 7 μm for sphalerite (Johnson, 1992).
- Chemical assay of all size fractions, providing a fully sized mass balance for the plant each month.

- Liberation analysis of size fractions. Until 1992/93 this was done by manual point counting and afterwards by QEM*SEM (Quantitative Evaluation of Materials by Scanning Electron Microscope). This provided a full size-by-size monthly mineralogical mass balance of plant operations.

Plant Surveys

In addition to monthly balances, more detailed information was obtained from occasional full or part plant surveys. The surveys are carefully designed to provide a complete snapshot of the operation, with a full mass balance including cyclone splits and down-the-bank flotation performance. All samples are assayed, and selected samples sized and analysed mineralogically.

Laboratory and pilot plant testwork

Laboratory and pilot plant testwork was used to test and identify solutions to problems, the size and quantity of potential performance improvements and the flowsheets required to achieve performance gains. As before, all samples were assayed, and selected samples sized and analysed mineralogically.

THE LIBERATION PROBLEM

The first step change in sphalerite liberation occurred in July 1980 when target zinc concentrate grade was dropped from 52 per cent Zn to 50.5 per cent Zn to maintain recovery above 70 per cent (Johnson et al, 1982). This change caused the adoption of the rigorous mineralogical approach to quantify future ore changes. Figure 6 shows a graphic summary of the changes to sphalerite liberation after 1980.

In Figure 6, 'sphalerite liberation' represents the percentage of sphalerite in plant feed which has been liberated before exiting the plant in either concentrate or tailings. This is achieved by the tonnage-weighted mathematical combination of all plant products to form a recalculated plant feed, which represents the total effect of all grinding and regrinding in the plant. A sphalerite grain is considered liberated if it is more than 90 per cent sphalerite in two-dimensional analysis (Gottlieb et al, 1994).

From 1984 to 1991, sphalerite liberation dropped from almost 70 per cent to just over 50 per cent. This was attributed to finer grained ore, although the recalculated feed sizing coarsened from P80 = 50 μ m to P80 = 80 μ m because of increases in throughput with no extra grinding power. A decrease in sphalerite liberation causes a drop in zinc recovery, since the maintenance of zinc concentrate grade at 50.5 per cent Zn allows no additional lower grade composites in the concentrate.

SPHALERITE LIBERATION IN RECALCULATED FEED VS ZINC RECOVERY

Smoothed Data: 3 period rolling average

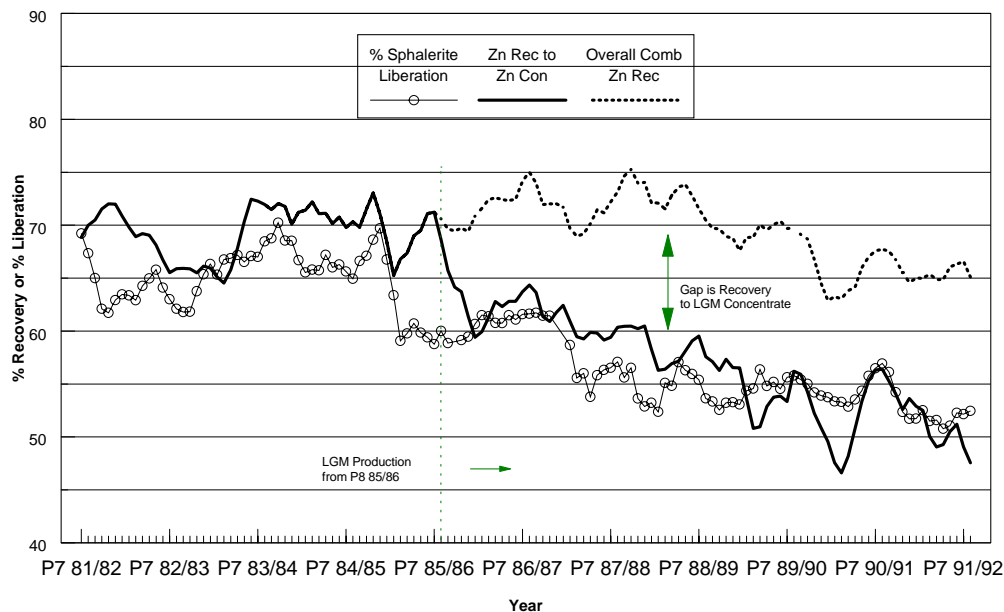


Figure 6.

There were two possible responses to the liberation problem; either grind finer to increase liberation, or place low-grade middlings into a new, lower grade concentrate. Because of the high capital cost of additional grinding, production of a Low Grade Middlings (LGM), or bulk concentrate became necessary in 1985/86, to maintain total zinc recovery. This concentrate typically assayed 13 per cent Pb and 34 per cent Zn, compared with zinc concentrate which had to contain 50 per cent Zn and less than 3 per cent Pb. Figure 6 shows the effect of the LGM Concentrate on total zinc recovery. The difference between the zinc recovery to zinc concentrate and overall combined zinc recovery is the zinc recovery to the LGM concentrate.

As liberation continued to drop, recovery to zinc concentrate fell with concomitant increases in LGM concentrate production. By 1988, total zinc recovery had to decline further as the target 34 per cent zinc in LGM concentrate was unattainable and the LGM concentrate market had become saturated.

In hindsight, production of LGM concentrate distracted management from the true severity of the problem, since zinc recovery was still quoted as over 70 per cent until 1989. This was really a misrepresentation, since only 55 per cent was recovered to zinc concentrate, with 15 per cent to LGM concentrate. It should also be noted that at the beginning of LGM production, revenue was high because of good contract terms for LGM concentrate. As production rose, contract terms declined until zinc in LGM concentrate was worth less than half that of zinc in zinc concentrate.

Size by size analysis

During the 1980's the sphalerite liberation declined in three major steps (as shown in Figure 6):

- Mid 1985 from 70 per cent to 60 per cent,
- Mid 1987 from 60 per cent to 55 per cent, and
- Early 1991 from 55 per cent to 50 per cent.

Figures 7 and 8 show the recalculated plant feed sphalerite liberation (by size fraction) and the zinc recovery to zinc concentrate (by size fraction) for selected months over the period 1984/1985 to 1991/1992. Size fractions displayed in Figures 7 and 8 are infrasizer fractions (F7, F6 and F5) and sieve size fraction +400# (+37um). The unsized sample is shown as ALL.

It can be seen from the sphalerite liberation by size (Figure 7) that sphalerite liberation in all size fractions decreased almost uniformly. Consequently, zinc recovery by size also decreased uniformly

(Figure 8). Figures 7 and 8 combine to show that all size fractions were becoming more complex and difficult to liberate, not just the coarse size fractions.

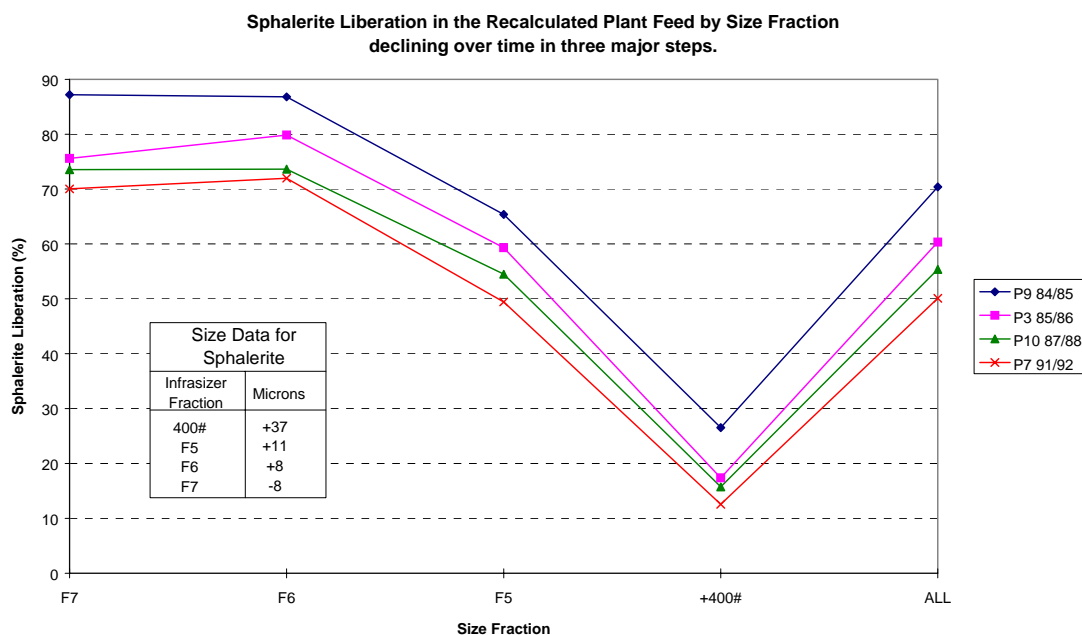


Figure 7.

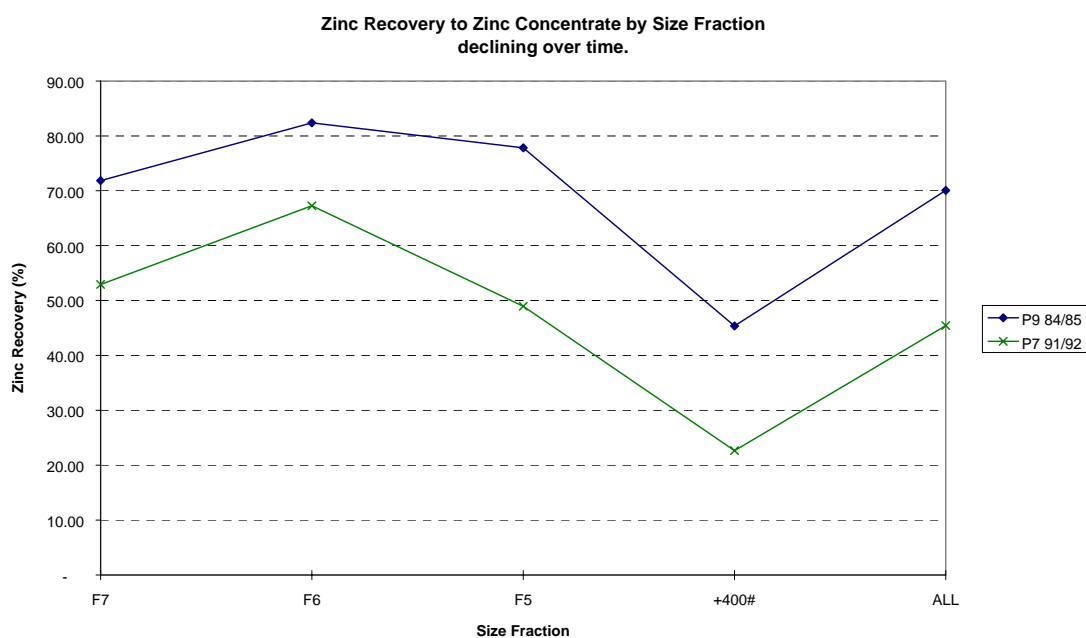


Figure 8.

THE SEPARATION PROBLEM.

While declining liberation posed the most serious problem for zinc metallurgy, decreasing separation efficiency posed the greatest problem for lead metallurgy, and was a secondary issue for zinc metallurgy. The separation problem was caused by increasing amounts of fine grained, carbonaceous

pyrite. As shown by Figure 9, lead head grade declined during the 1980's with a concomitant increase in iron sulphides. This was exacerbated by an increasing proportion of the pyrite occurring as the naturally floating carbonaceous type, rather than the "well behaved" euhedral pyrite.

Carbonaceous pyrite is hydrophobic under almost any flotation conditions and consumes large quantities of reagent, making flotation difficult to control. Figure 10 shows the changes to lead metallurgy from 1973 to 1990, with lead concentrate grade and recovery both decreasing and the high viability of lead concentrate grade. Although the natural floatability of the carbonaceous pyrite results in a greater impact in the lead circuit, Figure 11 shows that there was also some impact on the zinc circuit. For the same concentrate grade 1980-1992, iron sulphide content rose from 4 per cent to 6.5 per cent. This also contributed to falling zinc recovery, since the higher iron sulphides content in zinc concentrate left less room for composite particles containing sphalerite.

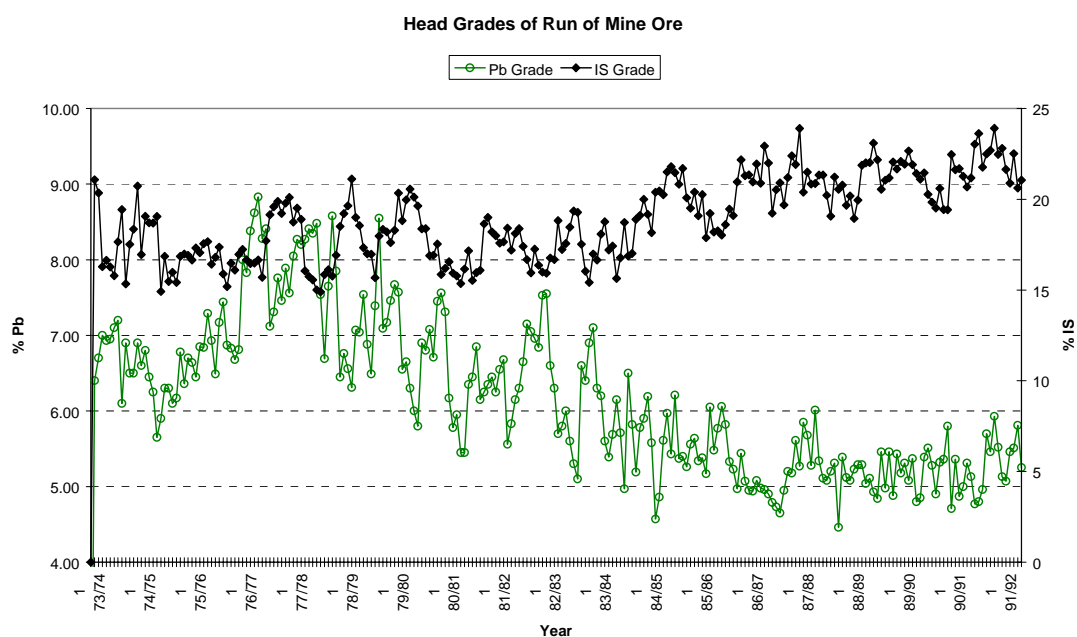


Figure 9.

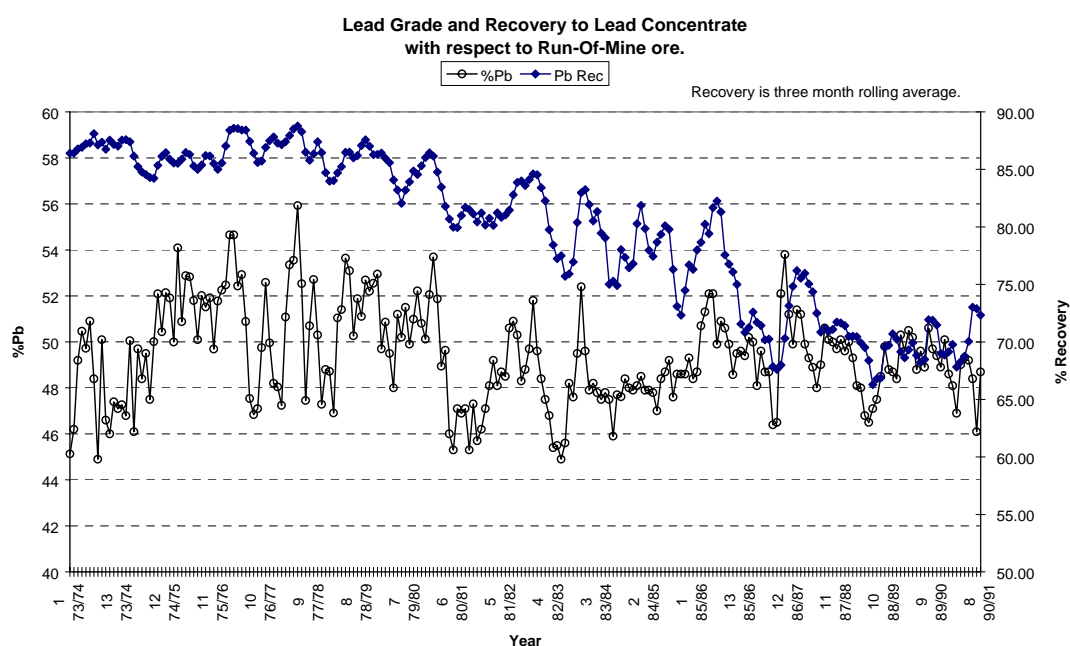


Figure 10.

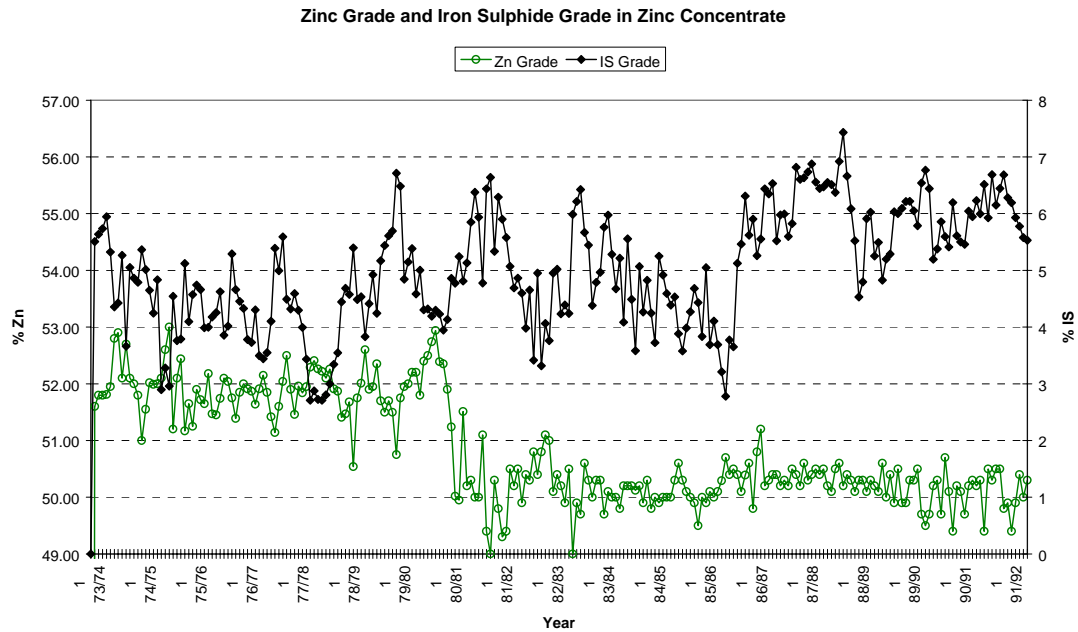


Figure 11.

The falling lead concentrate grade also posed a serious problem for the lead smelter. Smelter throughput was limited by sinter plant sulphur removal capacity. Increased pyrite in lead concentrate caused lead grade to decrease and sulphur to increase (Figure 12), reducing smelter throughput.

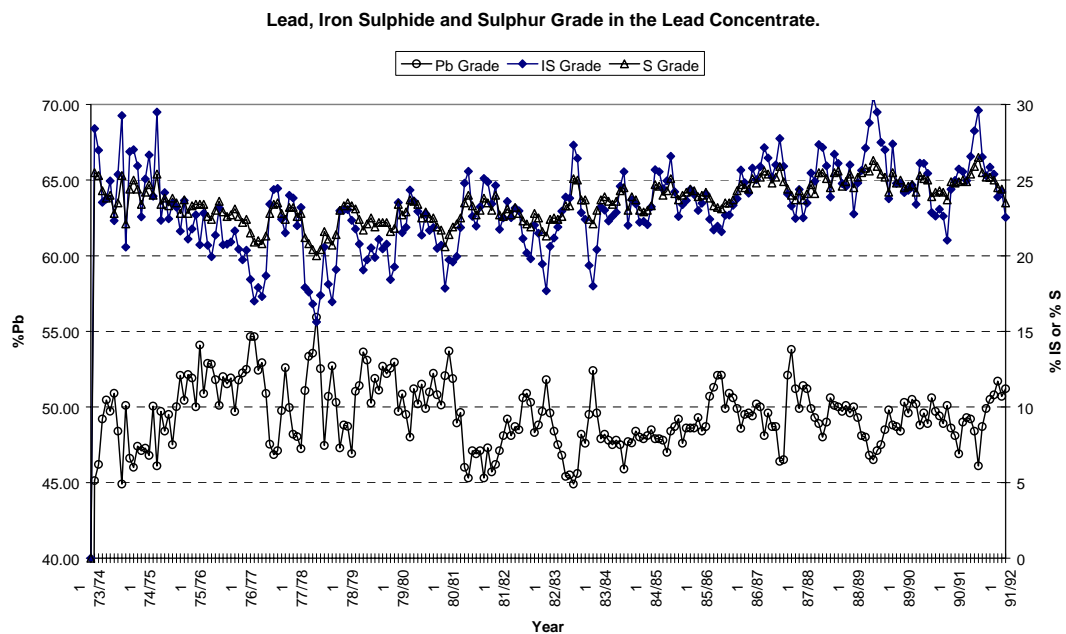


Figure 12.

Numerous circuit and reagent alternatives were trialed to minimise the impact of the carbonaceous pyrite. After several decades of work, the most effective response remains a dextrin depressant at natural pH (7.5 to 8). Other changes instituted in the 1980's to improve lead circuit separation included:

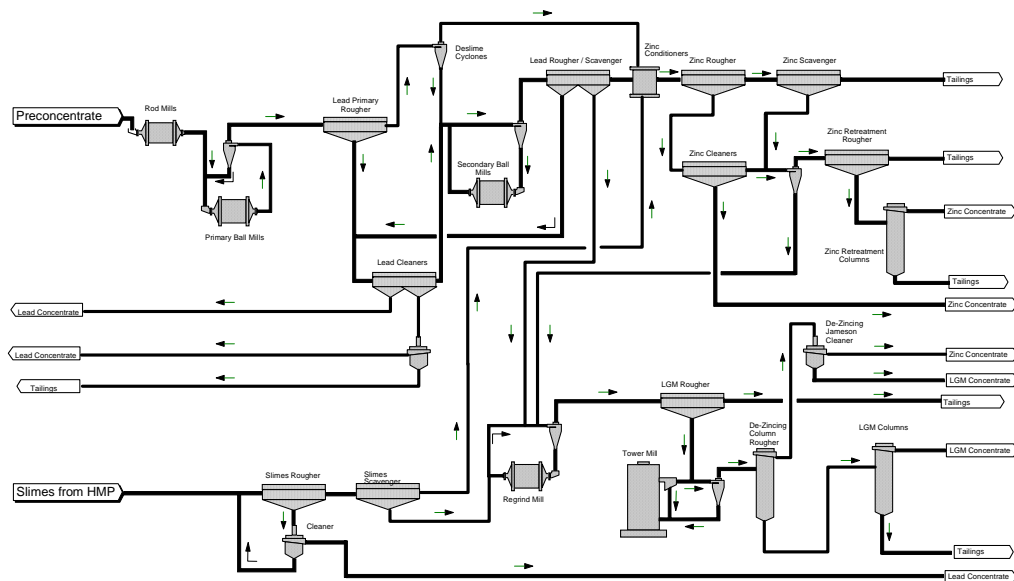
- Installation of preflotation before lead roughing which operated between December 1986 and February 1987. Preflotation concentrate was cleaned before discarding at an assay slightly higher than head grade for lead, zinc and silver. Preflotation operation was stopped partially because of recovery loss and partially due to the resultant unstable operation of subsequent lead roughing and cleaning. Without preflotation, the majority of lead circuit xanthate addition is added to mill feed, but with preflotation the addition is made to rougher feed. Addition of the xanthate to rougher feed appeared to cause instability in the lead circuit, particularly in summer with high pulp temperatures (+45 deg C).
- Use of a “high-low split” lead cleaning circuit during 1991 and 1992 (Figure 13). This circuit collected a high grade concentrate from the first part of the lead cleaners, whilst a low-grade concentrate from the second part of the cleaners was sent to further cleaning in a Jameson cell, the latter producing a high-grade concentrate and a low-grade tail. High-grade concentrates were sent to the lead smelter whilst the low-grade Jameson cell tailing was stored until smelter capacity was available.
- Installation of a Heavy Medium Plant (HMP) slimes roughing and cleaning circuit in 1988. These “slimes” are generated in the mining process, represent about 15 per cent of the lead in the flotation feed, and have different reagent requirements to normal feed (Grano et al, 1988). A flowsheet including roughing at pH 9 (with lime), zinc sulphate, and cleaning in a Jameson cell was developed, producing a lead concentrate from HMP slimes that averaged over 60 per cent Pb, compared with an estimated 45 per cent Pb when included in conventional feed. The separate HMP slimes roughing and cleaning contributed a two per cent increase in overall lead concentrate grade.
- The LGM concentrate circuit from 1986 assisted by directing difficult lead middling streams into a low grade concentrate. This raised lead concentrate grade, and helped match concentrator and smelter capacity by directing some metal away from the lead concentrate.

In the case of zinc separation, little improvement was possible because of the poor liberation. Use of traditional iron sulphide depressants (eg. lime and dextrin) became severely restricted because of uncontrollable circulating loads of composites and fine free sphalerite (Johnson et al, 1982). In turn, the circulating loads consumed limited cleaning and retreatment flotation capacity. Attempts to improve zinc circuit selectivity included :

- Operating the LGM circuit to provide an exit for the most difficult composite streams, eg zinc scavenger concentrate and zinc cleaner tailing;
- installation of new column cleaning capacity in the zinc retreatment and LGM circuits (Espinosa-Gomez et al, 1989 and Espinosa-Gomez and Johnson, 1991);
- substitution of lime with dextrin in zinc cleaning, with later restriction of dextrin additions to minimise circulating loads;
- trials of a number of supplementary collectors which promised, but did not provide, increased selectivity; and
- use of a hot reverse cleaning circuit developed in the pilot plant. However, by the time this approach was developed it was clear that circuit simplification was the priority rather than the addition of new equipment.

The changes made during the 1980's were individually effective in achieving performance improvements, however the rate of improvement did not match the ore type rate of decline. Further, the succession of small changes had dramatically increased the complexity of the combined lead, zinc and LGM circuits as the changes had treated symptoms rather than the underlying mineralogical cause. By 1992, the Concentrator had 14 exit streams: 8 concentrates and 6 final tailings (Figure 13) and suffered operational difficulties with respect to stable operation.

June 1992



SOLVING THE UNDERLYING PROBLEMS - THE 1990'S

The projects were;

- Tower mill regrinding installed in the LGM circuit (1991).
- The “Fine Grinding Project,” which doubled grinding and flotation capacity and instituted a “cold” lead reverse cleaning circuit (1992). This project addressed both key issues, ie liberation in the zinc circuit and separation (of carbonaceous pyrite) in the lead circuit.
- Improvement in liberation allowed circuit simplification, the increased use of conventional tools (eg high pH zinc circuit cleaning) and relocation of regrinding duties from the LGM circuit to the zinc retreat circuit.
- New ultrafine regrinding technology at Mt Isa was introduced for both zinc and lead regrinding (1994).
- Generally, the application of process control became more effective with the improvements because of more achievable targets.

While it was well-known prior to 1992 that finer grinding was required, the high capital cost prevented the acquisition of the requisite additional grinding equipment. Conversion of the Copper Concentrator to SAG milling in 1991 solved this problem by releasing two 5m by 6.1m, 2.6 MW ball mills. A project was approved to install these mills for secondary grinding in the Lead/Zinc Concentrator, along with a 520 kW Tower Mill for additional regrinding in the LGM circuit. The finer feed sizing and more dilute pulps necessitated extra flotation capacity, which resulted in the installation of two banks of nine Dorr Oliver DO600 cells for lead secondary roughing and scavenging, and three banks of 12 Dorr Oliver DO600 cells for zinc roughing and scavenging. Existing flotation cells were used to provide additional roughing, retreatment, or cleaning capacity. In summary, the new equipment provided the following changes :

- An increase in primary and secondary grinding power from 6.3 MW to 11.5 MW. Recalculated feed sizing dropped from P80 = 80um P80 = 37um microns;
- increase in regrinding power from 0.75 MW to 1.27 MW. Regrinding size in the LGM circuit dropped to P80 = 12um; and
- a doubling of flotation capacity.

The Tower mill was commissioned in December 1991, and the two secondary mills in October and November 1992. Figure 14 shows the effect on sphalerite liberation and zinc recovery. The key features are:

- Sphalerite liberation increased by 25 per cent by 1993;
- an increase in total zinc recovery of eight per cent. More importantly, the amount of zinc reporting to the low value LGM concentrate was reduced, yielding an increase in zinc recovery to zinc concentrate of over 15 per cent. These recovery gains not only achieved feasibility estimates almost immediately after commissioning, but also quickly exceeded the same estimates.

SPHALERITE LIBERATION IN RECALCULATED FEED VS ZINC RECOVERY

Smoothed Data: 3 period rolling average

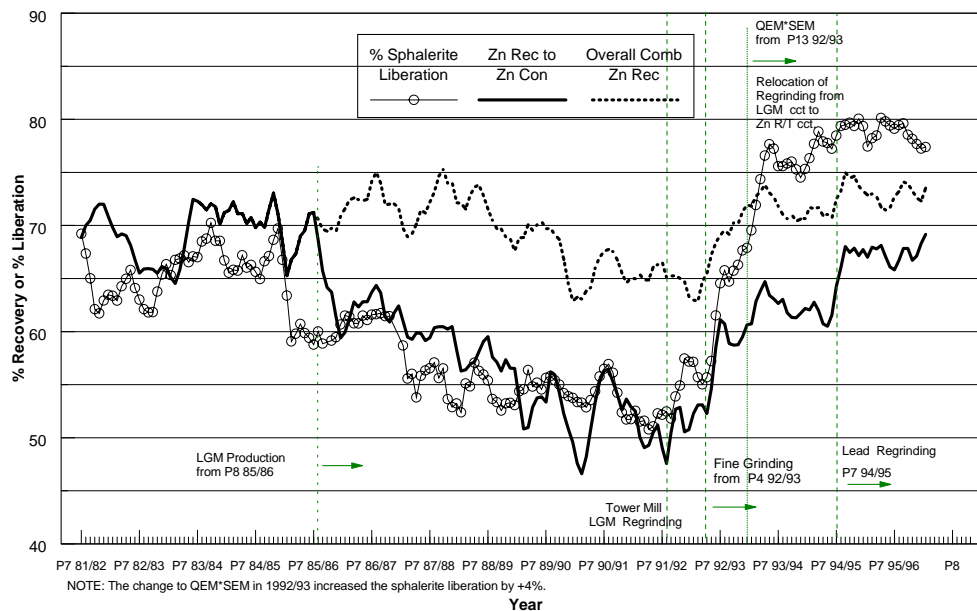


Figure 14.

MOUNT ISA MINES LIMITED - LEAD/ZINC CONCENTRATOR FLOTATION FLOWSHEET

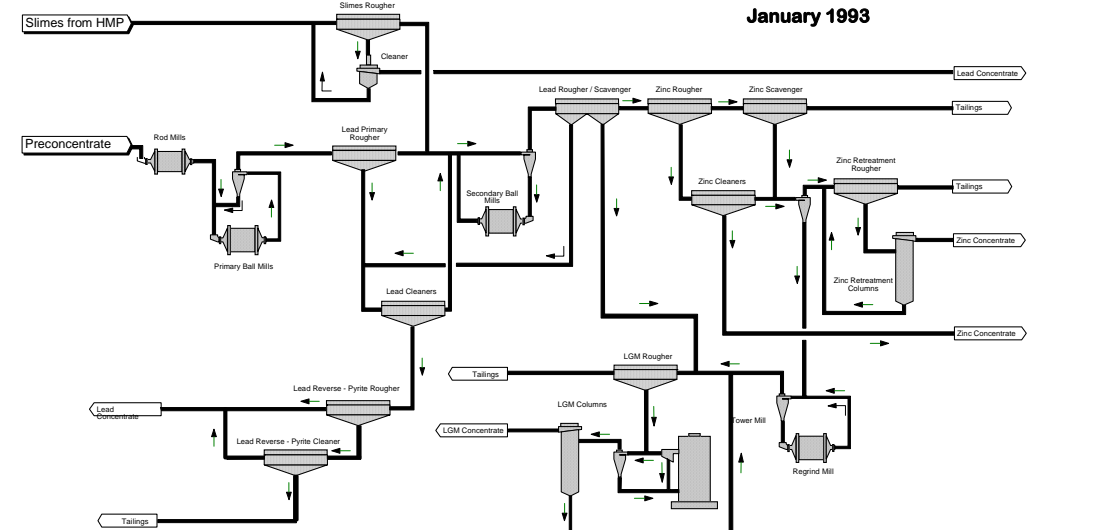


Figure 15. - Flowsheet after the installation of increased grinding and flotation capacity (nine products).

The Fine Grinding Project and Separation Improvements

The 'cold' lead reverse cleaning circuit was installed at the same time as the Fine Grinding Project to remove carbonaceous pyrite from lead concentrate. Conventional lead cleaner concentrate is raised to pH 12 with lime to depress galena but not carbonaceous pyrite. A pyrite concentrate is floated, cleaned, and discarded (Figure 15). The pyrite concentrate assays around 30 per cent Fe, 32 per cent S, and 19 per cent Pb. Typically, the reverse cleaning trades off one per cent Pb and 1.2 per cent Ag recovery for each one per cent increase in lead concentrate grade (and accompanying 0.5 per cent lower Fe and 0.3 per cent lower S). The maximum upgrading capacity of the circuit (because of physical constraints) is 4 per cent Pb. Operation of this circuit is intermittent, depending on current ore type, metal prices, and smelter performance. The circuit's major advantage is the provision of independent control of lead grade/recovery decisions. In the conventional cleaners there is very little ability to trade lead recovery for grade, since the lead cleaner tailing assay has to be kept low to keep galena out of the zinc circuit. A process control system varies the reverse cleaning circuit air addition to control a setpoint lead concentrate grade. This gives the lead smelter a much steadier grade concentrate, while minimising the recovery loss of galena. The circuit is shut down when better ores are encountered. The improved lead concentrate quality resulting from this circuit (by decreasing the iron sulphides) is shown in Figure 16.

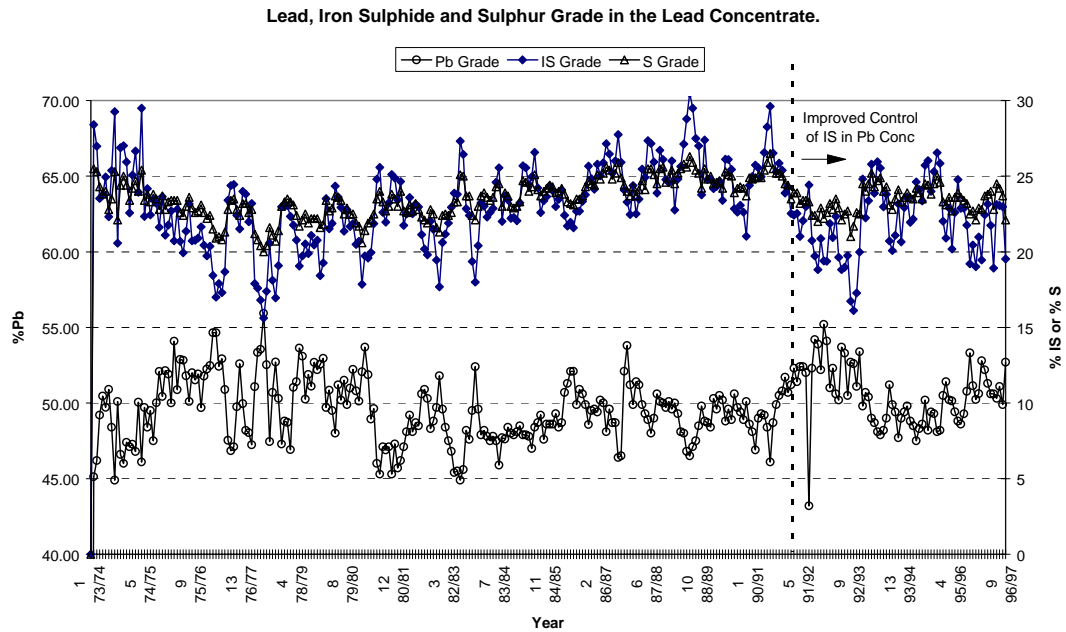


Figure 16.

FURTHER IMPROVEMENTS

In the two years after the implementation of the Fine Grinding Project, further performance gains were made as the circuit was adjusted and simplified. Effectively, operating personnel had to “unlearn” many of the circuit rules essential when poor liberation and insufficient flotation capacity were the root of many problems. The most significant of these were:

- Reintroduction of high pH zinc cleaning using lime. This had been abandoned prior to the Fine Grinding Project because of unmanageable circulating loads of composites.
- Relocation of some LGM circuit regrinding capacity into the zinc circuit (Figure 17). Together with the reintroduction of lime, this helped increase recovery to zinc concentrate by a further 5 per cent.

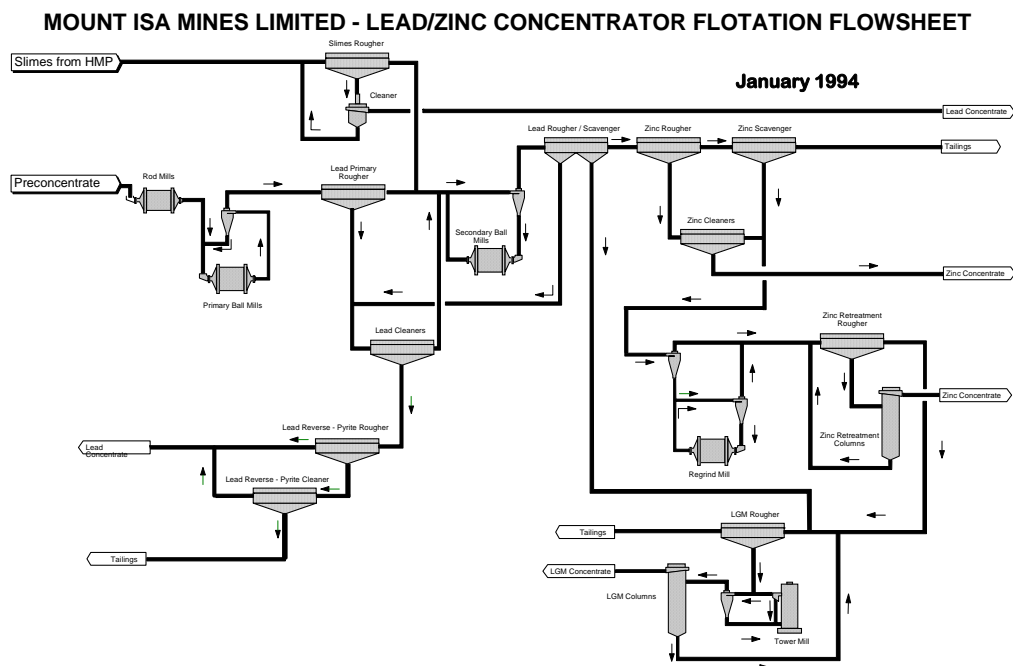


Figure 17. - Flowsheet after the relocation of regrinding capacity. (eight products).

- Use of some fresh water, instead of process (recycle) water, as dilution water in both zinc and lead cleaning. This reduced the impact of salt deposition (especially gypsum) on fine minerals surfaces. It also reduced the lime requirement in zinc cleaning as the slurry was previously supersaturated in calcium, and helped reduce frothing problems in both lead and zinc cleaning.
- Reintroduction of basic process control loops. Enormous efforts in advanced process control had previously yielded little gain, as the process was inherently unstable. Supervisory loops have been gradually introduced as the circuit has been simplified and stabilised. Tonnage/size/load grinding loops are used by operators over 85 per cent of the time on all grinding lines, and 18 flotation loops are used 70 per cent of the time. Tonnage based feed forward reagent ratio controllers are used for cyanide, copper sulphate and xanthate additions in the lead and zinc circuits. Adaptive controllers are used in the cleaners, adjusting both air and xanthate additions.

New Regrinding Technology

- Mount Isa Mines Limited developed revolutionary new ultrafine grinding technology for the McArthur River deposit, with prototypes developed in the Lead/Zinc Concentrator. The circuit has had two 1.1 MW mills regrinding lead concentrate, since 1995 (Figure 18) (Enderle et al, 1997). These mills have further increased liberation and recovery and simplified the circuit. The lead regrinding mills increased zinc recovery by 5 per cent by liberating sphalerite from composites that previously reported to lead concentrate. These mills regrind lead rougher concentrate to P80 = 15 μm .
- The regrinding mills also eliminated the bleed stream of difficult lead middling particles to the LGM circuit, leaving the zinc treatment as the only remaining feed to the LGM circuit.

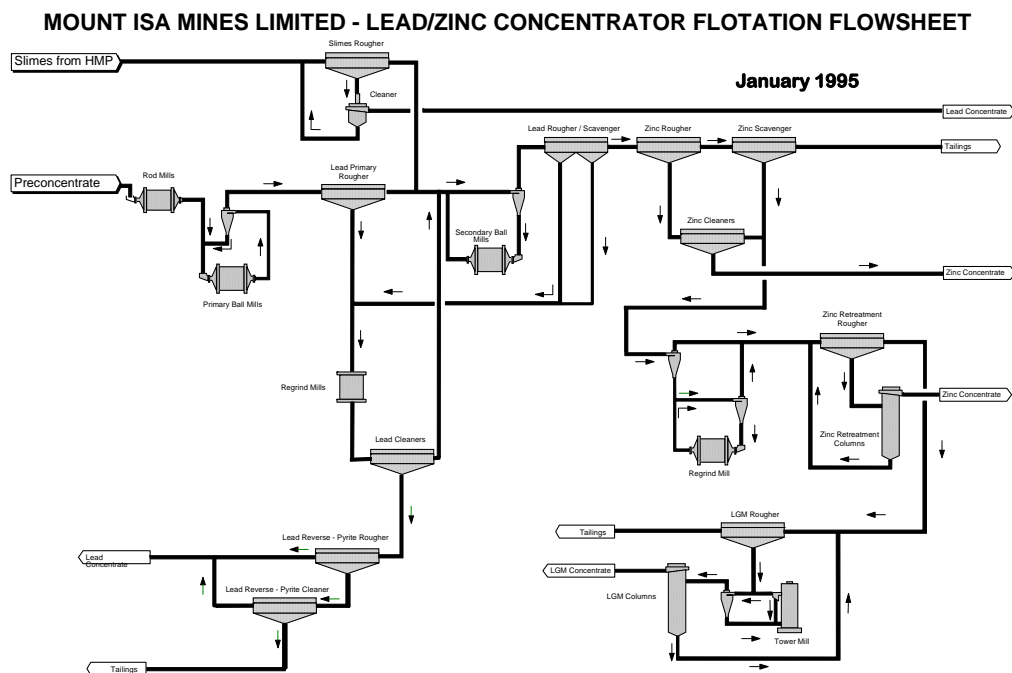


Figure 18 - Flowsheet after the installation of regrinding of lead rougher concentrate.

Size by size analysis

The increase in sphalerite liberation in the recalculated plant feed is shown on a sized basis in Figure 19 for selected months from 1991-1995. Figure 19 shows that liberation increased across all size fractions for each project which increased grinding power or grinding efficiency. The liberation increased in four main stages (Figure 14):

- Installation of Tower Mill in LGM circuit,
- installation of increased primary and secondary grinding power during the Fine Grinding Project,
- relocation of grinding power from LGM circuit to zinc circuit, and
- installation of lead rougher concentrate regrinding.

The zinc recovery by size data increased in a similar manner to the liberation by size data (Figure 20). The figure shows that all size fractions were more liberated with finer grinding and not just the coarse size fractions. The liberation was therefore improved by two methods; firstly by increasing the liberation of each size fraction and secondly, and more importantly, by moving particles from the coarse, less liberated size fraction to the finer, more liberated size fractions.

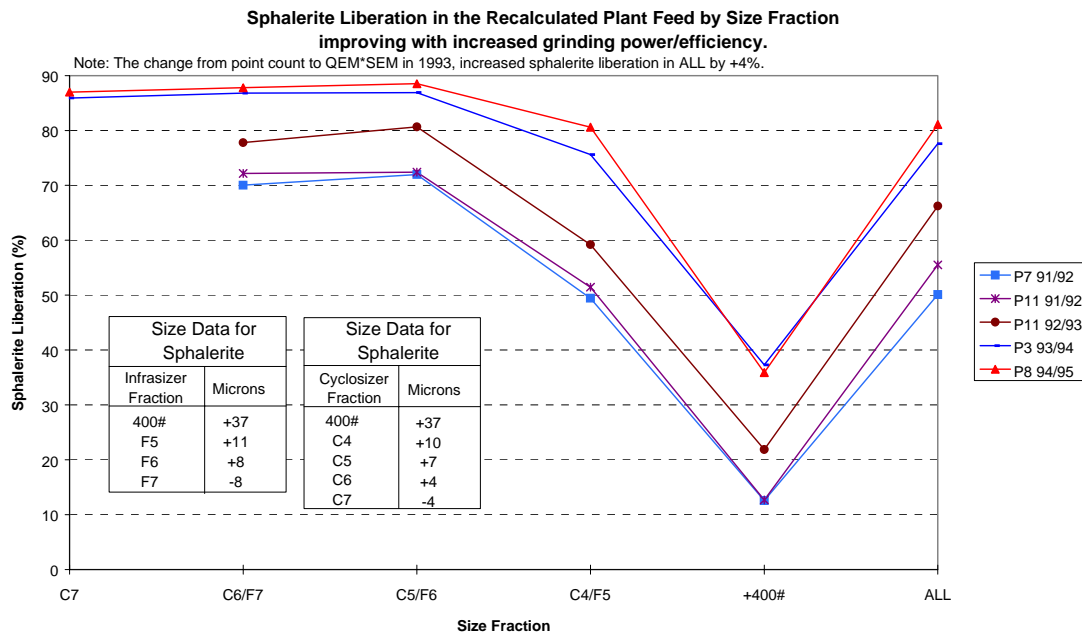


Figure 19.

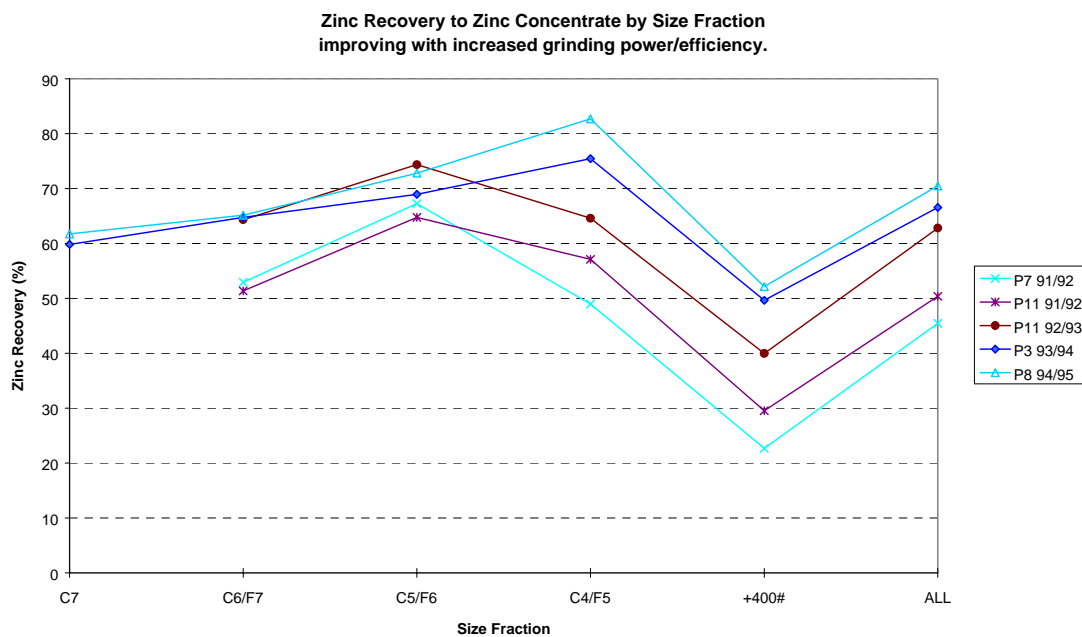


Figure 20.

CONCLUSION

Adoption of a rigorous, size-by-size mineralogical approach to plant operations was crucial to identify and solve the dramatic decline in ore quality and metallurgical performance.

The result was a 20 per cent increase in zinc recovery to zinc concentrate, 5 per cent increase in lead recovery to lead concentrate, improved quality for both lead and zinc concentrates, and 70 per cent reduction in the production of the low value LGM concentrate.

Important also is the simplification of the circuit. From 14 exit streams in 1992, the circuit had eight exit streams by 1995. This produced a dramatic improvement in circuit stability and increased ease of circuit operation. Three main indicators of circuit stability are:

- the willingness of operators to use simple process control loops to assist their decisions;
- the speed of achieving stability after plant start ups, ie. metallurgical results on start-up shifts are now indistinguishable from normal operating shifts; and
- plant spillage and hygiene. High side rubber boots are no longer issued, nor needed !

This case study is an excellent example of the benefits of applying a scientific approach to routine operations over a long period of time.

POSTSCRIPT - RECENT CHANGES

The metallurgical improvements described in this paper were driven by technology changes targeted at the fundamental nature of our fine grained, complex ore. The changes were highly successful and economically essential to business as ore quality declined. However, the improvements came at a price - high capital and operating cost. In 1996, the next improvement came from a comprehensive examination of the mine/mill/smelter business. This led to elimination of the LGM (bulk) concentrate and increased lead and zinc concentrate grades and recoveries, as well as providing considerable circuit simplification. These changes were achieved without capital and without extra operating cost. They will be the subject of future publications.

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