PAPER 3

Updating the Lucky Friday Mill – Experiences in Revamping a Historic Milling Operation

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ABSTRACT

In 1942, commercial mining operations began in what is now the Lucky Friday Ag/Pb/Zn mine in Idaho's Silver Valley. Wholly owned since 1962 by Hecla Mining Company, the mine has produced more than 145 million ounces of silver and has been a mainstay producer in Idaho's Silver Valley for over 70 years.

However, by the early 21st century, the mill had fallen, technologically, seriously behind the rest of the industry. Accordingly, starting in 2005, the current mill management team set to revitalizing the milling operation. This paper will describe how the mill was systematically evaluated and the aging infrastructure replaced with more current technology. Driven by internal mill review studies supported by QEMSCAN mineralogical balancing, the mill has in recent years been through waves of upgrading – from the installation of new mechanical and column flotation equipment to the introduction of on-stream analysis, a technology well-accepted around most of the industry but alien to this technologically conservative mining district.

The experience at Lucky Friday is a testament to the need for ongoing structured mill optimisation and technology upgrading in established mining operations. This paper will describe engineering experiences associated with upgrading equipment in an operating mill, contained in a confined building. Experiences in teaching the team new ways of operating a mill, and finally the metallurgical rewards that have been reaped will be described.

INTRODUCTION

Mining in Idaho's Silver Valley

Idaho's Silver Valley is a 64-km long valley flanked by the US Northern Rocky mountains, seldom more the 3-km wide, and stretching across much of Idaho's northern panhandle, roughly 150 km south of the Canadian border. Mining has been continuously active in the area for more than 140 years, during which time, the Silver Valley has become one of the richest silver producing districts in the world, and as of 1985 had out-produced the 500 years of production from the famed Potosi district in Bolivia. Since 1885, almost 100 deposits have been discovered and worked in the district (Chapman, 2000).

Legend has it that silver was first discovered in September 1885 by Noah Kellogg when his jackass wandered off and kicked over a tetrahedrite stone in the side of what is now Silver Mountain, Kellogg, ID (Chapman, 2000). The site turned into the Bunker Hill mine, one of the largest Ag/Pb/Zn mines in US history, a mine that operated as a major integrated mine/mill/smelter/refinery complex until 1981, and has continued to operate as a minor producer since.

Shortly after the Bunker Hill mine was opened, other mines followed – including the legendary Sunshine Mine, the single largest recorded producer of silver in US history, and one of the largest in the world.

The Lucky Friday Mine

The Lucky Friday Mine (Figure 1) is located adjacent to the town of Mullan, on the eastern end of the Silver Valley close to the border with Montana (Figure 2). Original claims were made on a modest quartz vein outcrop, as early as 1889 but it was only in 1939 that intensive exploration began which, by 1941, had delineated enough commercial ore to justify construction of the mine. The mine was constructed soon after and started production in 1942. It has since produced over 145 million ounces of silver. In 1958, Hecla Mining Company bought 38% of the mine, which was merged into Hecla in 1964. Lucky Friday is a deep underground mine, mining 1,000 tons per day of ore from the Gold Hunter deposit. The mine presently employs 300 people.

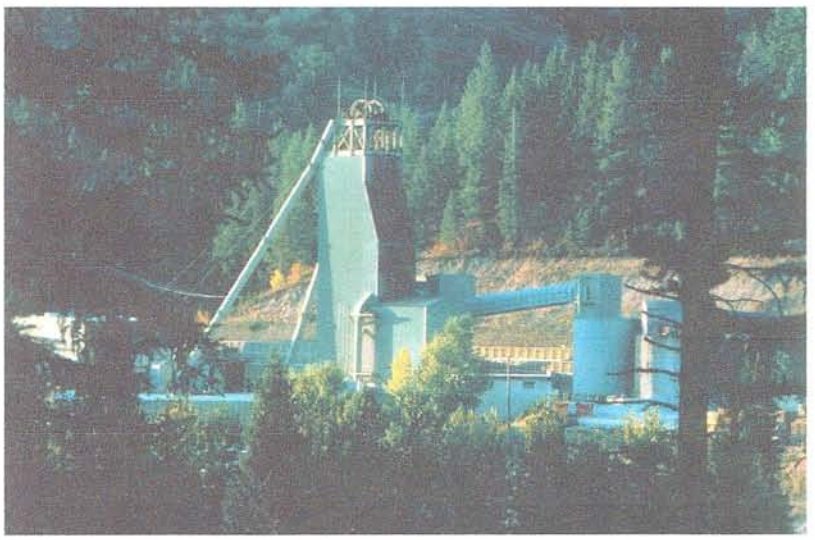

Figure 1: Lucky Friday mine

Silver production in 2009 was 3.5 million ounces, in concentrates that contained 22,000 tons of lead and 10,600 tons of zinc.

Figure 2: Lucky Friday mine location

Mullan is a mining town, owing its existence almost entirely to the mine, and was impacted accordingly during the period of very low metal prices (the silver price in the 1990's and early 2000's seldom rose above $6/oz and dropped as low as $3.80/oz). Consequently, in 2003, at the bottom of the last cycle, with production costs at over $4/oz, the mine was struggling to survive. At that time, Hecla, owners of the mine were faced with the decision to either shut the mine down or take the braver path of investing in the mine to improve its production efficiency, improve the quality of the concentrates created, increase silver production and reduce costs. They chose the latter option, a decision that has brought rich dividends through the success of the mine's efficiency improvement programs and the rise in the silver, lead and zinc prices through to the present day. At the time of writing, the price of silver is over $23/oz.

This paper describes how the mill was running prior to this improvement program, what improvements have been made, how they were implemented and what benefits have been seen.

Current Mining

Current mining is from the Gold Hunter vein system, a deep narrow vein epithermal system (Anon., 2004). The Silver Shaft, the main working shaft has been sunk to 6200 feet (1,9 km) though the present working level is 5900 feet (1,8 km). Mining is by overhand and underhand cut and fill, using bio-diesel powered rubber tired equipment, and overhand timbered raises using slushers. There are typically six to seven active headings producing ore.

Ore Mineralogy

Typical composition of the rougher feed, from QEMSCAN analyses of samples taken from the circuit during October 2009, is as shown in the first (left) column in Table 1. The remaining data describe how the mineral content is distributed by size, describing their propensity to slime or conversely resist breakage. Silver is mostly contained in tetrahedrite, with lesser amounts in the galena and other sulphosalts such as boulangerite and bournonite. Lead is most predominantly contained as galena and zinc as a relatively high grade sphalerite. The pyrite content is relatively low for Pb/Zn ores, at about 2%.

The host rock mineralisation is mostly quartz and siderite, both acting as benign floaters in Lucky Friday flotation. In milling, galena has a strong tendency to slime, tetrahedrite somewhat less so, and sphalerite and pyrite less than galena or tetrahedrite. Quartz tends to resist breakage.

The ore is quite consistent, QEMSCAN analyses have been conducted periodically on Lucky Friday samples through the duration of the mill upgrade program and the modal composition is largely the same as shown below:

Table 1: Modal composition of Lucky Friday rougher feed, November 2009

		Distribution, %		
	Abundance, %	+75	+38	-38
Pyrite	1.8	23.9	35.3	40.8
Tetrahedrite	0.1	22.1	21	56.9
Galena	7.8	7.6	19.1	73.3
Boulangerite	0.1	22.4	30.9	46.7
Sphalerite	4.8	23.1	31.8	45.1
Quartz	24.4	52.6	21.3	26.1
Siderite	47.4	37.1	31.2	31.7
Micas/Clays	2.3	50.5	25.7	23.8
Others	2.3	n/a	n/a	n/a

The ore is coarse-grained, both the lead and zinc sulphides being roughly 85% liberated at 106 microns. Although the data on the silver mineralisation is not adequate to confidently determine the liberation characteristics, it seems likely that the tetrahedrite is similarly sized.

The ore has an average work index of 12.9 kWh/ton, and is very consistent in hardness, the mean deviation being less than 1 kWh/ton.

CRUSHER PLANT

In 2004, mill throughput was nominally 1,000 tons per day – however the mill produced a substantial amount of coarse "gravel", roughly 100 tons per day, which had been accumulated on the mine site for an extended period of time. Though this was relatively low grade and hard, it represented a serious loss in production capacity. North American deep underground mines have a very high mining cost, and with all the cost already sunk in delivering this material to the surface, the inability to mill this material represented a substantial cost inefficiency. As the gravel was typically ½ to ¾ inch in size, it was concluded that the crusher plant needed to be modified to ensure a -½ inch product.

Prior to 2004, the crusher plant consisted of a two-stage circuit ((Figure 3). Primary crushing employed a Kue Ken 20 x 36 (inch) jaw crusher, which was set to create a product with a 3-inch top-size, from a feed delivered from the mine with a 12-inch top size. This product was delivered to a single deck 4 x 7 (foot) vibrating screen with ¾ inch openings. Screen oversize was delivered to a Symons three-foot standard cone crusher, set to deliver a product sized at ¾ inch. This latter crusher and screen were deemed to be operating at full capacity and it was concluded that an attempt to run the circuit in two stages to a product size of ½ inch would fail.

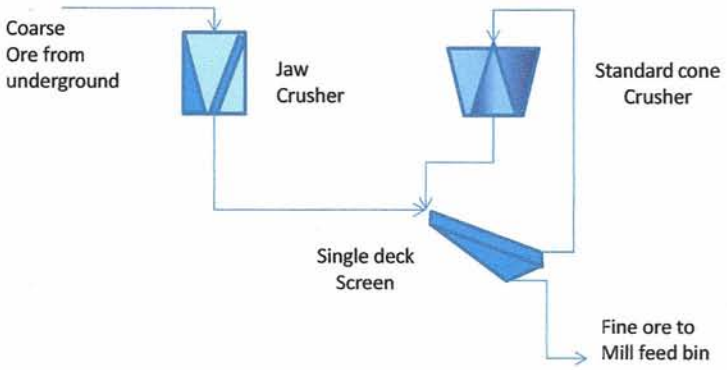

Figure 3: Two-stage crusher circuit operating prior to 2004

Accordingly, in 2004, the circuit was modified to include a third stage of crushing, employing a Symons three-foot short head cone crusher (Figure 4). Further, the screen was replaced with a larger 5 x 12 (foot) triple deck screen initially with one-inch, half-inch and three eighths-inch apertures. The initial design called for the finest screen to consist of oval-shaped self-cleaning apertures but these proved too easily blinded, so this deck was retrofitted with conventional ½ inch square apertures, the middle deck apertures were changed to ¾ inch. The flowsheet is as shown in Figure 5.

The expanded crusher circuit eliminated the troublesome coarser material, and gravel production stopped. The actual benefit was very difficult to quantify, however, as so many other changes were being made to the mill at the same time.

GRINDING CIRCUIT:

Before 2004, the grinding circuit was the ultimate in simplicity (Figure 6) – the 9½ x 12 (foot) ball mill running in closed circuit with a single 20-inch Krebs cyclone. The mill operated with a mix of 3-inch and 4-inch mild steel balls, in an overflow mill fitted with manganese steel liners. The mill feed rate was typically 42 tons per hour, though with 3-4 tph of gravel being removed through the trommel screen oversize, the grinding rate (tonnage to the cyclone overflow) was actually nearer 38-39 tph. Water was added to the mill feed to control the discharge density at about 70% solids, and to the cyclone feed pump box with the cyclone feed density being targeted at 63% solids. All density measurements were (and continue to be) made by hand. Cyclone overflow runs at 40% solids, underflow at about 80% solids. Grinding circulating load was about 200-300%.

(A) (B)

Figure 4: New tertiary Symons short-head crusher.
(A) Installation adjacent to the old building. (B) Crusher in operation

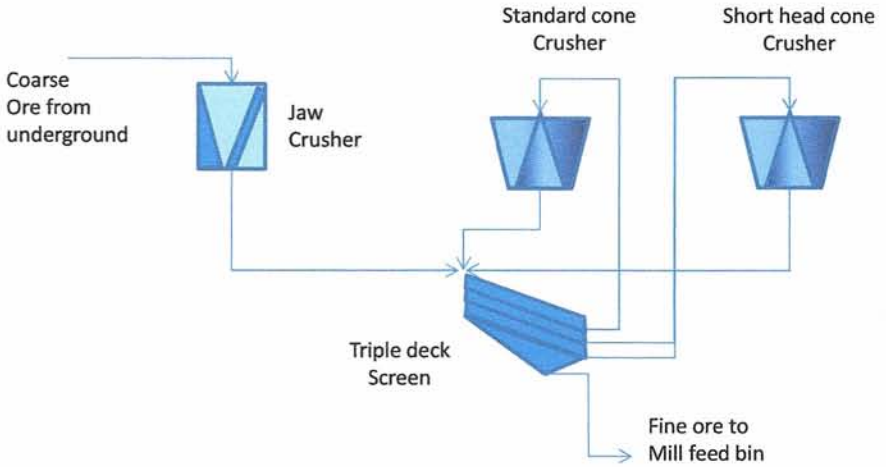

Figure 5: Three-stage crusher circuit operating since 2004

Size-by-size analysis of the cyclone overflow pointed to excessive Pb and Ag grades in the slime sizes and subsequent QEMSCAN analyses showed that sliming was a far more pervasive problem in the grinding circuit than liberation. This is a particularly significant problem for galena, as its density is so much higher that it has a greater tendency to report to the cyclone underflow. Accordingly, in addition to attempting to increase throughput through the circuit, changes were made to the circuit to reduce the degree of sliming of, especially, the lead and silver values. The following most significant changes were made:

1. The 20-inch cyclone was replaced with a nest of 4 x 10-inch cyclones. This allowed for far greater flexibility in the circuit, allowing for throughput to be pushed to maximum, and in subsequent years fine-tuned as needed.

2. An Outokumpu flash flotation unit was installed on the ball mill discharge to recover Pb and Ag before they can return to the mill through the cyclone underflow. This unit has proved to be capable of producing a final concentrate grade product, assaying ~63% Pb and 90-95 oz/ton Ag. Typically roughly 40% of the values are recovered to final concentrate through this route. As will be discussed later in the paper, this also had the benefit of freeing up flotation capacity in the lead circuit – a benefit that proves very useful at times of feeding high grade ore, and also when throughput is pushed harder through the mill.

In addition, the ball size was changed, initially to 3-inch and 2-inch, but in more recent times to 2-inch. Further, by modifying the cyclone geometry, the circulating load has risen to ~350%.

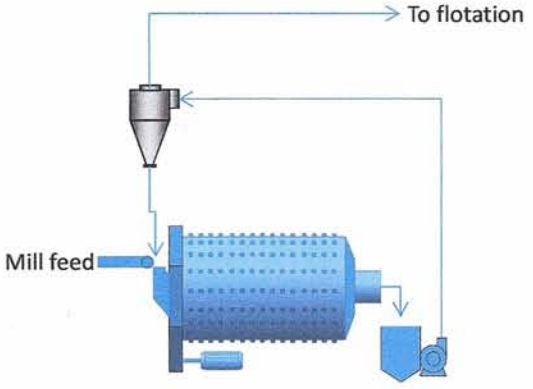

Figure 6: Grinding circuit in 2004

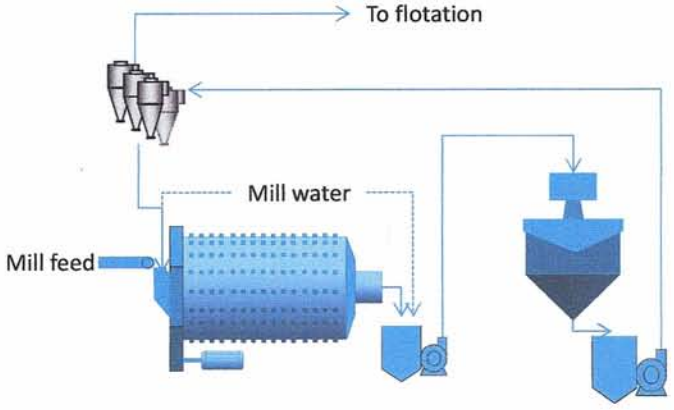

Figure 7: Present grinding circuit

Following the changes, and a period of continual fine-tuning and optimisation (such as changes in the cyclone geometry and dropping the ball sizes), the mill throughput has risen to 43 tons per hour, with no gravel now being produced – representing, overall, a 13% increase in throughput. The product size, at 73-76% passing 106 microns, has coarsened slightly from the previous 78% passing 106 microns – however the mineralogy has clearly indicated that this would not have an adverse effect on recovery, and overall metallurgy seems to be bearing this out.

Installation of flash flotation has been particularly beneficial. While the simultaneous coarsening of the grind and the installation of the flash flotation unit has made it quite difficult to truly establish the benefit in terms of reduced Pb/Ag sliming, the additional flotation capacity has been key to improvements in overall plant performance. Indeed when the flash flotation unit has to be taken off-line, the lead circuit can quickly become overwhelmed resulting in the need to drop tonnage. The flash flotation unit has also proved successful from a maintenance perspective. It in fact seldom needs to be taken out of service for maintenance reasons and, after three years of near-continuous service, the agitator is only now reaching the point where it is sufficiently worn to be replaced. This may in part be the result of the relatively non-abrasive ore however concerns of excessive wear that had been cited to the authors by other operators, have not been borne out in the case of the Lucky Friday installation.

SILVER/LEAD FLOTATION CIRCUIT

The whole process optimisation initiative started innocently enough early in 2003 with a QEMSCAN analysis of the silver/lead concentrate. The initial objective of the study was to determine if and how the lead concentrate could be upgraded. The following conclusions arose from this study:

- Galena was well-liberated, with 81.2% of the galena being greater than 90% free.
- Pyrite contained in the concentrate was also more free than expected, at 45% free.
- The concentrate contained 9% siderite, of which 46% was free and presumably present through entrainment.
- Silicates also accounted for 9% of the concentrate mass, and these were 68% liberated.

In total, some 27% of the concentrate was either pyrite or non-sulphide gangue, almost half of which was liberated and likely present largely through entrainment. The potential economic payback from rejecting just some of this material for a marginal mine was tantalising (Figure 8).

Achieving such upgrading was, conceivably, possible by a combination of finding a more effective circuit configuration and introduction of some form of froth washing capability. In the case of the latter, froth washing trays were installed in 2004 as a trial, into the existing pre-rougher cells, to see if even this crude approach to gangue rejection could yield some improvements. The results from this initial study were promising, indicating a 2% improvement in lead concentrate grade from 67.3% to 69.7% - however a follow-up study yielded somewhat inconclusive results. In the authors' experience, this is not unusual, as froth washing a conventional cell concentrate, where positive bias is hard to establish and the froth depths very shallow is usually only partially effective. Follow-up work using column flotation, described later, yielded more reliable data.

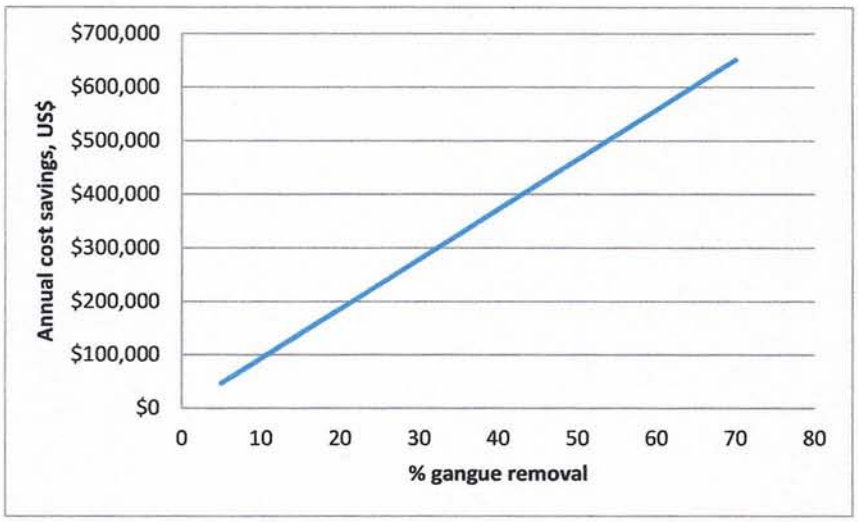

Figure 8: Cost savings associated with gangue rejection from the 2003 Ag/Pb concentrate

Lucky Friday's mill is a small plant. With roughly 40 tons per hour throughput, the cleaner circuits in particular are amenable to relatively easy modification, albeit on a temporary basis. Further, the Lucky Friday ore is forgiving, so configuration changes could confidently be made without risk of a metallurgical "catastrophe". Accordingly, with an open-minded management and a motivated maintenance crew, full plant trails can become a relatively easy way to investigate different circuit configurations so, rather than running through a lengthy laboratory test program, the decision was made to test various flowsheet configurations in the mill. In addition to the froth washing studies, the following exercise tested five different circuit configurations which was completed in just two months near the end of 2004 – during which time (including the zinc flowsheet testing conducted soon after), some 500 meters of piping was installed and removed – a considerable amount in a mill that is less than 100-meter long. The baseline circuit in 2004 was as shown in Figure 9.

The first modification tested involved re-directing the scavenger concentrate to the 1st rougher feed, so allowing the pre-roughers to float a super-high grade concentrate without risk of dilution from the reground scavenger concentrate material (Figure 10). This also maximised the residence time in the pre-roughers for optimal recovery of this super-high grade lead product, directly to final concentrate. This circuit was run continuously for 4 days.

The beneficial effect of removing the circulating load from the pre-rougher feed on pre-rougher concentrate was substantial. Pre-rougher concentrates averaged 72-76% lead, much higher than the 66-69% grade typical when the scavenger tailings were redirected to the pre-rougher feed. Iron grades dropped substantially, as did zinc grades. This was an interesting result. The recirculating stream consistently assays higher than the head grade – but it had a substantial detrimental effect on the quality of the product floated from the pre-roughers. A lesson is learned here: that streams should not necessarily be matched in circuit design, by grade, as much as by quality. The new feed contained significant coarse, fast-floating values in a benign ore matrix –

ideal for producing top-grade product, while the circulating load contained considerable slimed values and gangue from regrinding, the separation of which is considerably more challenging.

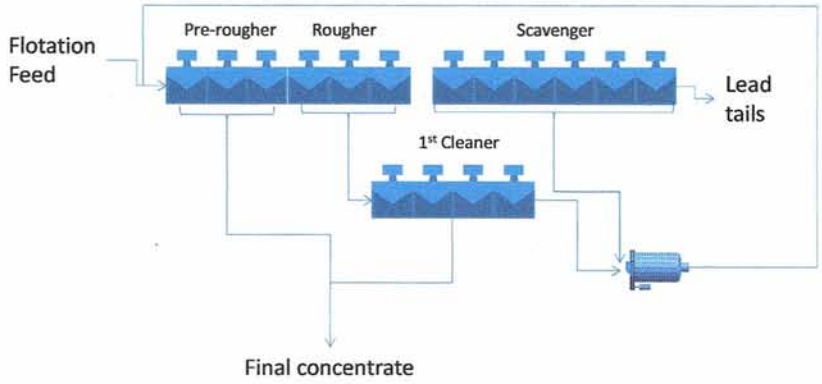

Figure 9: 2004 baseline Pb/Ag circuit

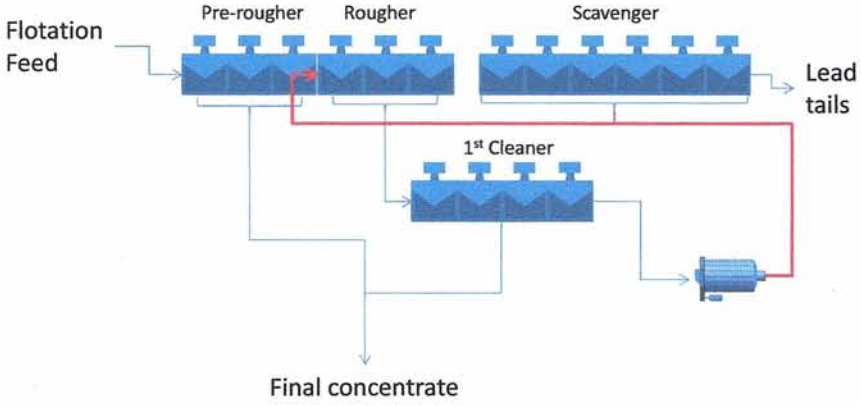

Figure 10: Redirecting the scavenger concentrate to the rougher feed.

This improvement was so stark, that the change remained in place for the rest of the program. Some 40-50% of the Pb and Ag were floated to a high-quality product, meaning half the job was done – so attention was now focused on the other half – the slow-floating Pb and Ag in the rougher and scavenger concentrates, and those values in the circulating load – which was now floated into a relatively low-grade product assaying about 40% Pb.

The second circuit modification studied involved commissioning some additional cleaner flotation capacity, previously off-line. This effectively increased the overall 1st cleaner capacity, while retaining the style of the original circuit as a single cleaner – essentially therefore testing if additional cleaner capacity alone would be beneficial. The easiest way to do this at the time was

to commission some spare cells, and to split the rougher concentrate between the two cleaners (Figure 11). However without adequate flow instrumentation, it was not possible to ensure that the concentrate was indeed equally split. This may have detracted from the effectiveness of this particular plant trial.

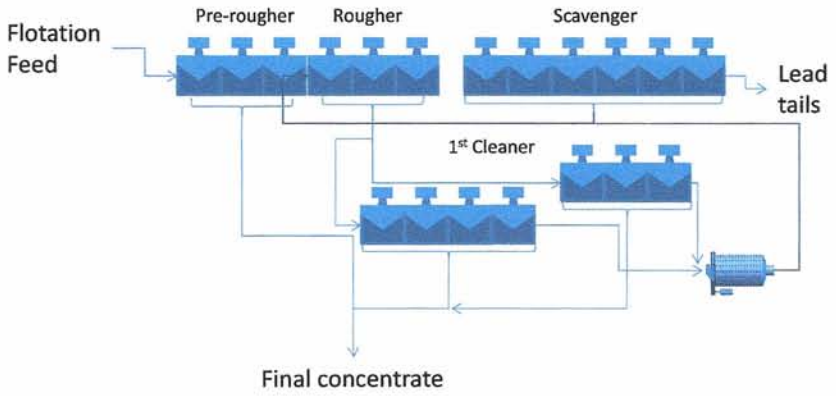

Figure 11: Additional first cleaner capacity

Running this circuit offered no benefits over the baseline. The roughers pulled a concentrate assaying 32% Pb, and the cleaner cells yielded little upgrading at all – with the cleaners upgrading the rougher concentrate by just 4% to about 36%.

The next circuit tested involved using these same cleaner banks but now in a two-stage cleaner configuration (Figure 12):

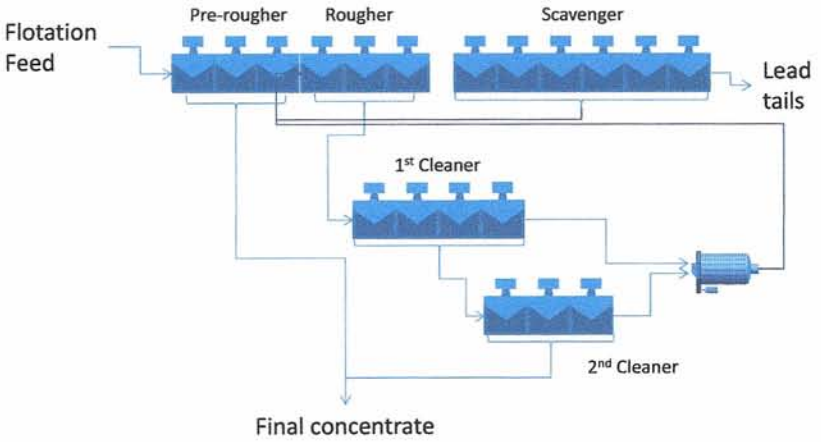

Figure 12: Introducing a second lead cleaner

Running the circuit in cleaner-recleaner configuration was more successful, yielding a final concentrate roughly 15% higher in lead grade than the rougher. The rougher concentrate, itself, rose somewhat likely reflecting a reduced circulating load of poor-acting material from the cleaner tailings. The rougher concentrate grade rose to 46% lead, while the recleaner concentrate rose to 54% lead. This result demonstrated the value of two-stage cleaning – however even a concentrate assaying 54% lead contains substantial gangue (about 38%) so considerable room for improvement remained.

Indeed it was surmised from this study that while pre-roughing (or flash flotation) was an excellent means of creating some high grade concentrate, the existing, old and poorly performing mechanical cells were incapable of upgrading the poorer concentrates now arising from the roughers.

To test this, a column cell from the zinc circuit was borrowed to replace the mechanical second cleaner, while the column tailings were scavenged in two mechanical column scavenger cells. The resulting plant performance now showed a further substantial improvement over previous circuits, with the column concentrate lead grade averaging 65% - a full 24% above the rougher concentrate grade at the time.

The overall concentrate grade (including the pre-rougher concentrate) assayed 69%, and the lead recovery just under 91%. This represented a 3% improvement in overall lead concentrate grade, at similar recoveries.

Some important lessons had been learned from this exercise, including:

- An outlet was needed for the fast-floating lead and silver to be removed quickly from the circuit, unhindered by competition from circulating loads. This conclusion formed a seed of justification for the unit flash flotation cell installed in the grinding circuit.
- Multi-stage cleaning was beneficial
- Froth washing and especially column flotation was beneficial.

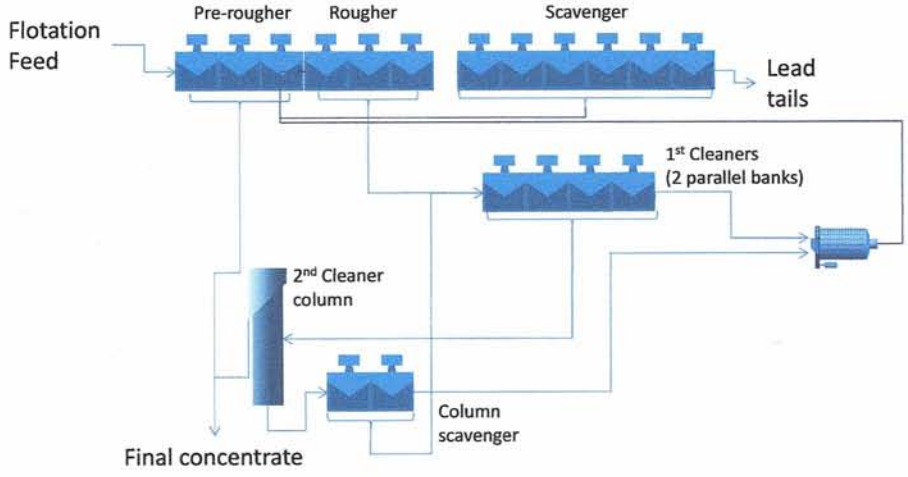

Figure 13: Recleaning using column flotation

These findings were incorporated in the new circuit, designed and constructed during 2005. The circuit was somewhat similar to that shown in Figure 13 but on the supposition that if one column cleaner is better than none, then two must be better than one, the lead cleaner circuit as installed in late 2005 included two large flotation columns representing the first and second cleaners. The 1st cleaner column was 72 inches in diameter, the second 48 inches in diameter. (With funds still tight in 2005, the columns were taken out of the mill from the newly acquired and long mothballed Starr Mine – hence their large size). The columns were installed with CPT "Slamjet" spargers. The new design also incorporated replacement of the rotten 50-year old rougher and scavenger cells with refurbished cells from the Starr Mine and bought commercially.

With all the replacement equipment on site prior to US Thanksgiving weekend in 2005, the mill was shut down on Thanksgiving Day and, through the following week the entire new circuit, including the roughers and scavenger, was retrofitted. This included tearing out the old cells, building in the supporting infrastructure for the new cells, installing them and associated access points and walkways, hooking them up and commissioning them – all in eight days.

The new circuit started successfully on the Friday after Thanksgiving. Pictures depicting the different stages of the main flotation floor retrofit are included in Figure 15 below

Accordingly, by the end of 2005, the Ag/Pb circuit was as shown in Figure 14.

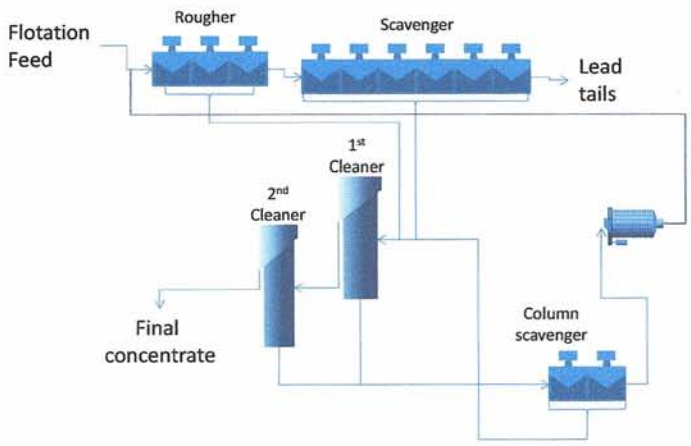

Figure 14: Lucky Friday Pb/Ag circuit, late 2005 to 2007

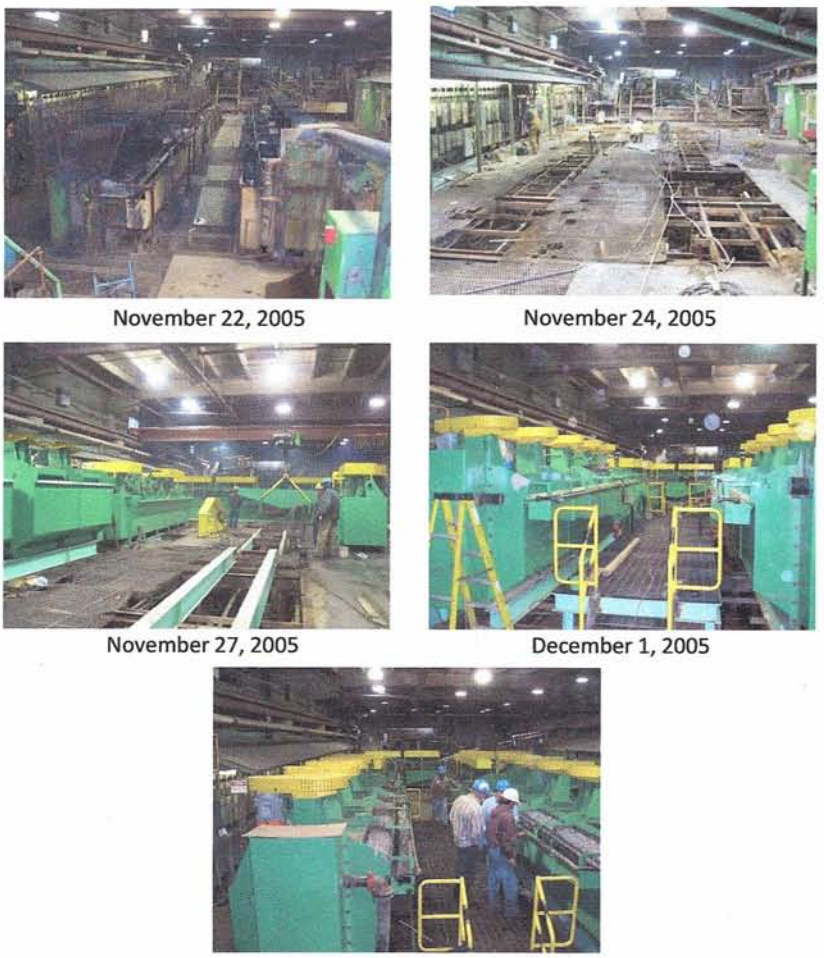

Figure 15: Retrofitting the main flotation floor

The final finding from the 2004 QEMSCAN study, an apparent lack of any need for regrinding within the Lucky Friday Ag/Pb cleaner, was not acted on during this plant trial exercise, or the changes that were made immediately afterwards. The practice of regrinding was very entrenched at Lucky Friday and removing the regrind, as well as changing the configuration and installing column technology was seen as one "risk too many". Accordingly, the regrind mill remained a key player in the circuit. However, as the operators grew more comfortable with the new circuit, confidence to reduce the use and eliminate regrinding became the eventuality – first ball addition was stopped, then finally regrinding was removed in the changes made in 2007. There was no adverse effect on cleaner performance.

Also, in 2007, the former 42-inch diameter zinc 1st cleaner flotation column was commissioned for cleaner duty on the column scavenger concentrate, with the aim of reducing the load and increasing the feed grade into the 1st column cleaner. This circuit (shown in Figure 16) has operated to the present day.

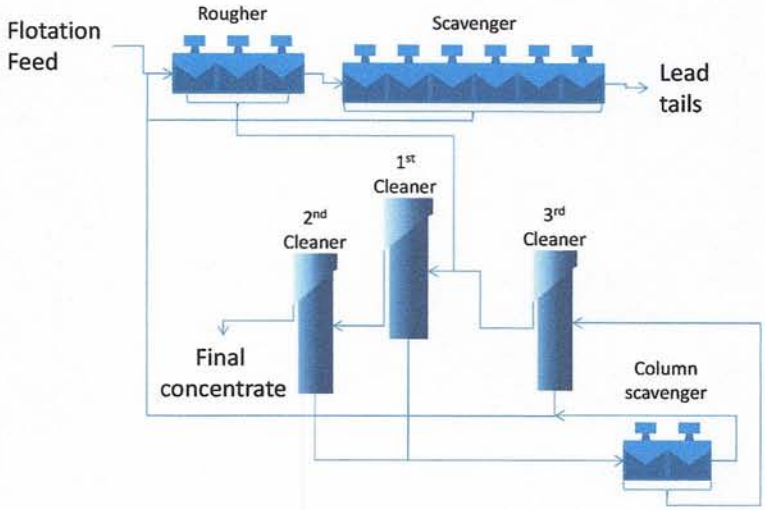

Figure 16: Lucky Friday Pb/Ag circuit, 2007 to present

The resulting circuit performance provided a dramatic response improvement over the 2004 plant circuit performance. Not only had flotation feed tonnages risen by 23%, but silver and lead recoveries had risen by 4%, averaging 92% to the Ag/Pb concentrate in 2009 as opposed to 88% in 2004 – and final concentrate grade was 3-5% higher. These substantial improvements were achieved with, for the most part, minimal capital and virtually no interruption to production.

ZINC FLOTATION CIRCUIT

The change from the traditional Lucky Friday vein system to the Gold Hunter vein system in the early 2000's created specific challenges in the zinc flotation circuit. While the Lucky Friday ore traditionally had only minor contents of Zn, the Gold Hunter vein was a much more Zn-rich material. Accordingly, far greater zinc circuit metallurgical focus was required to maximize the performance of this circuit.

In 2004, zinc recovery averaged about 67% (75% based on the zinc circuit feed). The zinc circuit (shown in Figure 17) was operating in a poor state of repair and had been the focus of very little technical attention for a long time. As the zinc grade (and the zinc price) rose, corporate interest in the zinc product grew. Accordingly, when in 2004 a mineralogical study revealed the potential for zinc recoveries of up to 90% (based on zinc circuit feed), considerable technical attention started to be applied to the circuit. Initially, work started in the laboratory, a small program of 13 tests showing that with reagent optimisation, recoveries could be raised to about 83-85% (based on zinc circuit feed) – however, reagent over-dosing could have a substantial effect on zinc grade – presumably with the additional reagent promoting the flotation of the pyrite in the ore. Further, the effect of circulating loads could only be established in the laboratory through locked cycle testing, which the Lucky Friday mill laboratory was not well geared to do (nor were the human resources readily available for such work), and the shallow froths in the laboratory could not replicate the cleaning capability of the column employed as a zinc cleaner in the plant. For these reasons, and as a result of the successful experience from running plant trails in the Ag/Pb circuit, further work moved to plant trials in the zinc circuit.

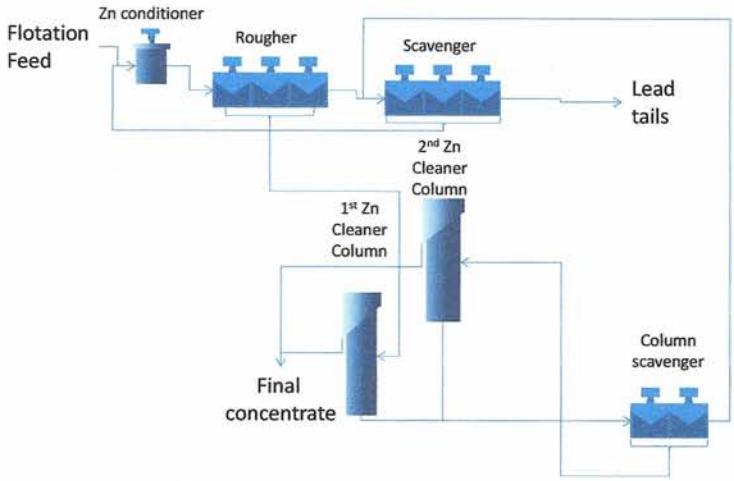

Figure 17: Zinc circuit as operated in 2005

The baseline plant performance, based on a three month statistical average is shown in Table 2. The circuit as operating at the time could yield a 52% Zn concentrate, at 75% stage recovery (based on the zinc plant feed).

Table 2: Zinc assays and recovery from existing and proposed flowsheets

	Assays, % Zn	
	Current	Proposed
Zinc circuit feed	2.0	2.1
Zinc concentrate	52.4	55.1
Zinc tails	0.58	0.44
Recovery, %	**75.4**	**80.0**

The proposed circuit employed more reagent to boost zinc recovery, while changing the cleaner configuration to two stages to enhance concentrate quality (Figure 18). Copper sulphate dosage was boosted to 0.6 lb/ton, xanthate and the dithiophosphinate, Cytec 3418A, were each boosted to 0.068 lb/ton, and MIBC to 0.084 lb/ton. Laboratory testing had shown conclusively that dithiophosphinate was needed in conjunction with the xanthate to maximise zinc recovery and these were used together initially, but at the time of writing only xanthate is being used in zinc flotation. The changes were made during a maintenance shutdown in December 2004, and restarted in the new configuration on December 9th. Sampling was conducted during December 9th to 11th, the average results from which are summarised in Table 2. Considerable improvements in both concentrate and recovery were achieved, although the plant did not "look" totally right. The second column, in particular, was "burping", suggesting excessive air or poor water chemistry for bubble generation. Reducing the air failed to totally resolve the problem. More frother (MIBC) was added and column froth depths were dropped to ensure ongoing concentrate production. The problem became particularly acute on December 11th, which could

somewhat be ascribed to a change in ore quality as similar problems were uncharacteristically observed in the Ag/Pb circuit at the time. Interestingly, when the December 11th data are eliminated, a zinc recovery (from December 9th and 10th) of 83% is calculated - almost identical to what had originally been achieved in the laboratory.

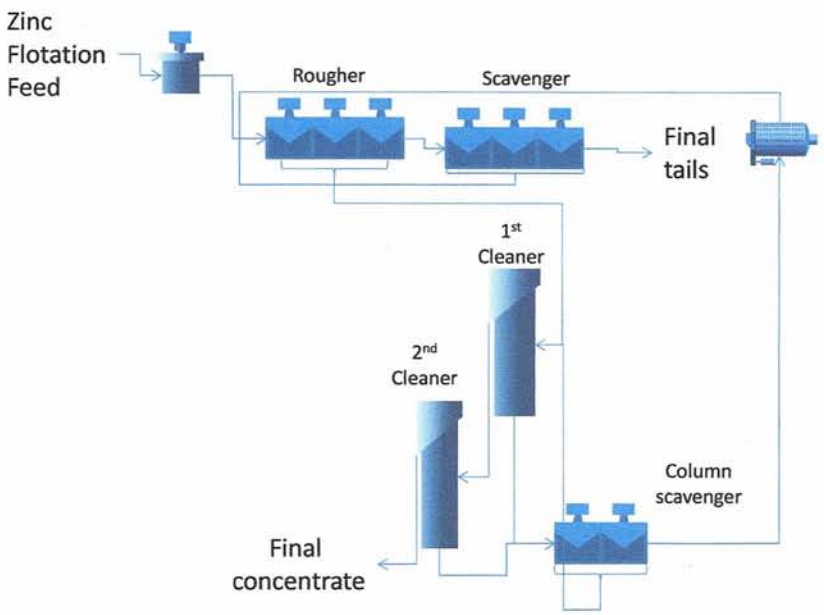

Figure 18: Test zinc circuit configuration, December 2005

The success of the zinc plant trial prompted a re-design of the circuit. The poor state of the zinc roughers and scavengers, now almost 50-year old, needed to be addressed and newer cells were installed for this duty. There was insufficient evidence from the plant trial that regrinding offered any benefit so this was removed from the circuit, and the column scavenger tailings were returned to the zinc conditioner - otherwise the circuit was as described in Figure 18. This circuit operated until the end of 2007, when in a bid to boost grade further, an additional column was installed as a cleaner scavenger cell (Figure 19). Also, following the success of the flash unit in the Pb/Ag circuit, a zinc unit "flash" cell was also installed. While locally called a flash cell, the 10-m^3 capacity of this cell easily matches the entire rougher/scavenger circuit. To date this has been used at times of high zinc feed grades however recent matrix mineralogy studies by Blue Coast Metallurgy Limited have indicated significant loss of zinc through incomplete flotation, so trials are planned to incorporate this tank cell permanently into the circuit.

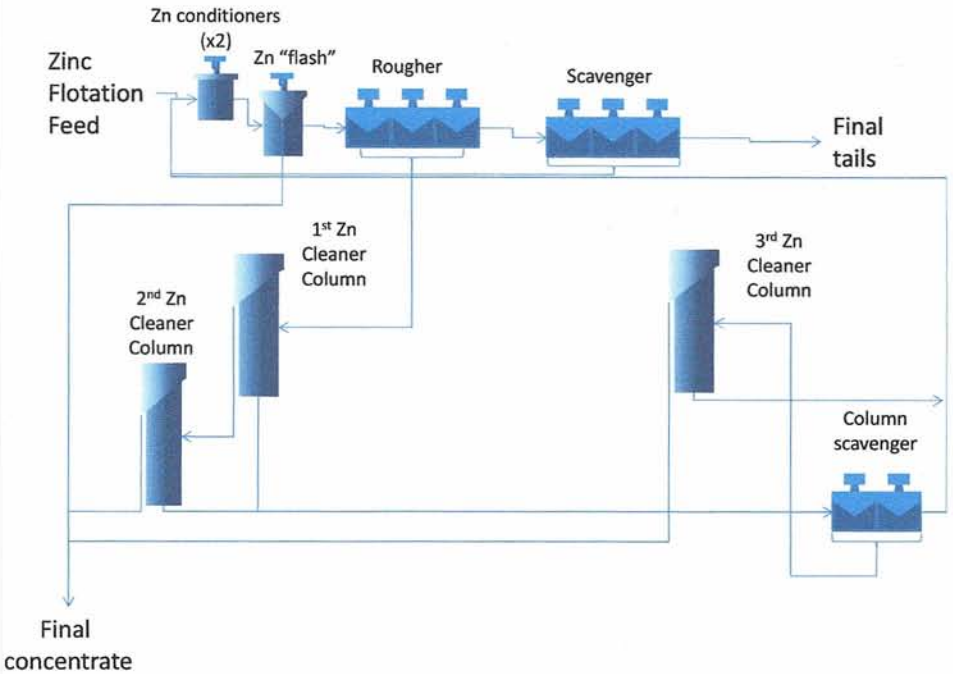

Figure 19: Circuit operating from end of 2007 to present

OTHER CHANGES AND IMPROVEMENTS

The changes to the circuit were not limited to just replacing aging flotation cells, installing columns and changing the flotation configurations. In 2004, other updates were badly needed. The circuit at the time was run totally manually, with a bench-top XRF being the only means of process control available to the operator. Further, reagents were dosed using (at best) Clarkson feeders, samples were taken entirely by hand. In addition to the above changes, metering pumps were installed to improve reagent control, a 12-stream on-stream analyser was installed and a two-stage sampling system installed for the feed and four products. In addition, increasing instrumentation is being installed all the time ultimately leading to expert system control of the flotation circuits.

The existing specialty flotation reagent suite used with xanthate was replaced with a combination of Cheminova's Danafloat 571 and 096 dithiophosphate and dtp blended products, through work performed by Zeroday Enterprises, LLC. This provided improved process control and more consistent resultant concentrate grades. Judging from the relatively low lead-froth mobility and rougher cell froth collapsing, which called for a stronger frother, plant evaluations with Cheminova's Pennol #26 was conducted blended at different proportions in MIBC. This improved lead recoveries approximately by 1-2%, but importantly further improved froth mobility and circuit control. Within the last year, it has been found that Pennol improves zinc scavenger flotation recoveries and a small dose has been added since.

The final change was perhaps the most profound – in-keeping with the trend industry wide, but exacerbated by the exodus of operators from Lucky Friday during the worst of the last down cycle, experienced operators were at a premium and a whole new team had to be trained to run the mill.

PLANT PERFORMANCE SINCE THE UPGRADES

Sustained and continuing improvements have been made to the throughput and metal recoveries since the different changes were implemented (Figure 20 and 21).

Concentrate quality is also improved but with the rise in metal prices in recent years, the focus has been more on recovery and the improvement in concentrate quality has been less significant.

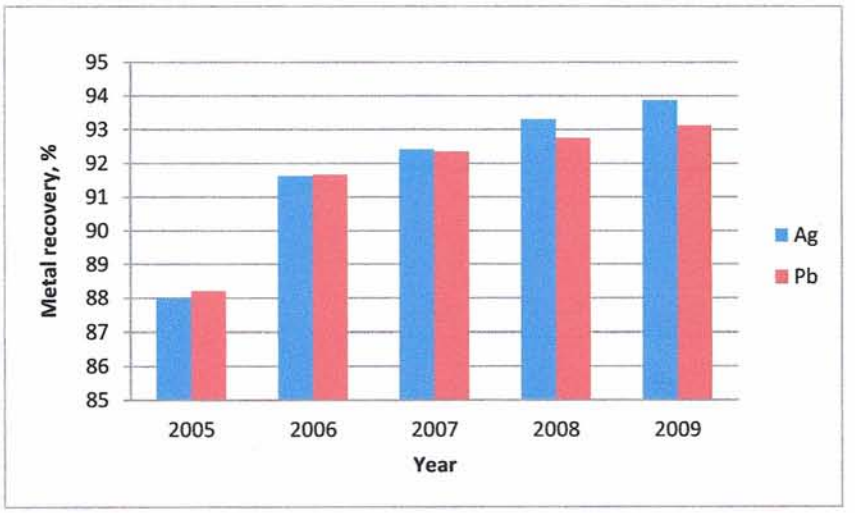

Figure 20: Lead and silver recoveries 2005-2009

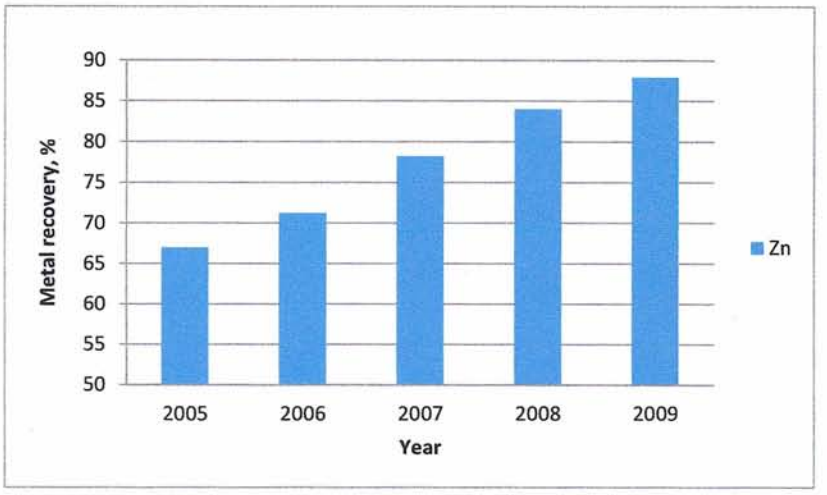

Figure 21: Zinc recoveries 2005-2009

WHAT NEXT?

During 2010, matrix mineralogical studies have been conducted on products from the new circuits, to see what improvements could be implemented next – and the lessons learned from these studies are set to trigger a new wave in plant improvements and optimisation (commissioning of a new tailings impoundment has deflected the attention of the technical team away from the mill in recent months). Optimisation is seen as an ongoing process at Lucky Friday and is far from complete, though perhaps to this point the lowest hanging fruit has been picked.

In the last six years, the Lucky Friday Mine has been transformed from a marginal high cost silver producer to one of the most profitable silver mines in the industry, and Hecla are fully committed to further investments into the future of the mine. An investigation is underway to evaluate sinking a new shaft to 7,800 feet (2,4 km). Along with this investigation, a mill de-bottle necking study will also be performed. A minimum goal of a 15% increase in mill throughput is expected. Lucky Friday may be nearly 70-year old, and the oldest primary precious metal mine in the United States, but her future has perhaps never looked as bright.

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