

The Increased Recovery Project at the Phu Kham Copper-Gold Operation, Laos PDR

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ABSTRACT

The Phu Kham copper-gold operation located in Laos People's Democratic Republic (PDR) is owned and operated by Phu Bia Mining Limited. PanAust Limited, based in Brisbane Australia, holds a 90 per cent interest in Phu Bia Mining Limited, with the remaining 10 per cent held by the Government of Laos PDR.

The 12Mt/y copper-gold concentrator at Phu Kham commenced production in April 2008. Flotation copper and gold recovery from commissioning was consistently poor, due to the high levels of pyrite and problematic secondary and oxide copper species and non-sulfide gangue.

The low recoveries of copper and gold from Phu Kham were recognised during the feasibility study design phase. A bulk sulfide flotation regime in roughing produced high recoveries, however a large capacity cleaning circuit was required and there was risk of not meeting saleable concentrate grade specifications. The installed design offered a capital efficient compromise to ensure that a concentrate grade of 24% copper could be achieved at acceptable recoveries.

Incremental improvements in copper recovery were achieved by 2011 through conversion of conditioning cells to flotation cells in both roughing and first cleaning and increased cleaner capacity through the installation of a Jameson cell. In 2012 the Phu Kham Upgrade Project was commissioned to increase mill throughput, along with additional rougher and cleaner capacity to increase residence times at the nominal 16 Mt/y throughput. The marginal reduction in primary grind size due to the additional ball milling power also provided a small recovery improvement.

This paper presents the development, design, implementation, and performance of the Increased Recovery Project commissioned in April 2013 which targeted the major causes of copper loss from the circuit. Based on fundamental and detailed test work and mineralogical analysis of concentrator streams, an optimised design was developed which has increased copper and gold recoveries by over 5% and 10% respectively. The Increased Recovery Project has positioned the concentrator to process high pyrite ores that were previously considered untreatable, and opened further opportunities for maximising flotation recovery by de-bottlenecking plant regrind and cleaning capacity.

INTRODUCTION

The Phu Kham operation is owned and operated Phu Bia Mining. PanAust Ltd based in Brisbane holds a 90 per cent interest in Phu Bia Mining through its wholly owned subsidiary Pan Mekong Exploration Pty Ltd, with the remaining 10 per cent held by the Government of Laos PDR.

The Phu Kham operation, which is located in the Xaisoumboun Province in Laos PDR, consists of a copper-gold mine using conventional shovel mining and truck haulage to a flotation concentrator. The concentrator was initially designed to achieve 12 Mt/y throughput, treating a high pyrite copper-gold skarn ore with significant clay content (Meka & Lane, 2010).

The design of the Phu Kham plant was a compromise between high copper recovery and the risk of meeting acceptable final concentrate grade. A high recovery bulk rougher flotation design concept was developed during the pre-feasibility study, but difficulty in achieving a 24% copper concentrate specification due to very high pyrite content and complex intergrowth of secondary copper sulfides resulted in a more selective flotation approach being implemented. This design concept produced lower final recovery but required a far lower capital cost due to decreased cleaner capacity. The risk of not achieving saleable concentrate due to high pyrite in the early years' feed blend was also reduced.

The Phu Kham Upgrade (PKU) study commenced in 2010, with the objective of maintaining copper production above 60,000 t/y to counter a scheduled decline in mine copper head grade from 2013. The PKU design was for a nominal 16 Mt/y throughput capacity, and addressed capacity constraints in grinding and flotation circuits as described in Bennett et al, 2012. The PKU was commissioned in the third quarter of 2012 and nameplate capacity was reached by the last quarter of 2012.

The Increased Recovery Project (IRP) study was initiated in 2009 due to the poor copper recovery achieved in early plant operations. Commencing as a tailings retreatment project, the study showed that it was technically feasible to generate a low grade copper-gold concentrate from bulk sulfide flotation of plant tailings, with leaching of the concentrate to recover copper and gold into high grade products. Detailed mineralogical work and metallurgical test work was performed to determine the causes of copper and gold loss to tailings, which was supported by comprehensive routine monthly mineralogical data of plant composite samples extending back to 2008. During the study an opportunity for increasing copper and gold recovery in the existing concentrator at lower technical risk and lower capital cost became apparent, with this opportunity developed into the final IRP design.

The PKU and IRP have been complimentary projects, with significant increases in plant throughput to 20 Mt/y rate and copper and gold recovery increases of over 5% and 10% respectively achieved.

IDENTIFICATION OF PLANT LOSS MECHANISMS

Mineralogical analysis of monthly composite samples has been performed at G&T Metallurgy Services since the post-commissioning period in 2008. Until mid-2010, a major source of copper loss was typically fine secondary copper sulfide minerals (chalcocite and covellite) in cleaner tailings which were easily oxidised and exhibited very slow flotation kinetics under the aggressively selective conditions employed in flotation for pyrite depression using a high lime pH environment. The general association of gold with pyrite in Phu Kham ore was the major reason for gold loss. In mid-2010 the ore feed to the plant changed from mainly supergene secondary copper sulfide mineralisation to mainly hypogene primary sulfide mineralisation, with chalcopyrite typically over 70% of the contained copper sulfide. The change in mineral characteristics in ore feed changed the major copper loss source to low quality copper sulfide and gangue composites in coarse size fractions in the rougher flotation tailings. This loss category typically accounted for over 50% of the total unrecovered copper in final tailings.

The data from a November 2011 (Figure 1) showed that the copper sulfide particles lost were typically composited with non-sulfide gangue and to a lesser extent pyrite.

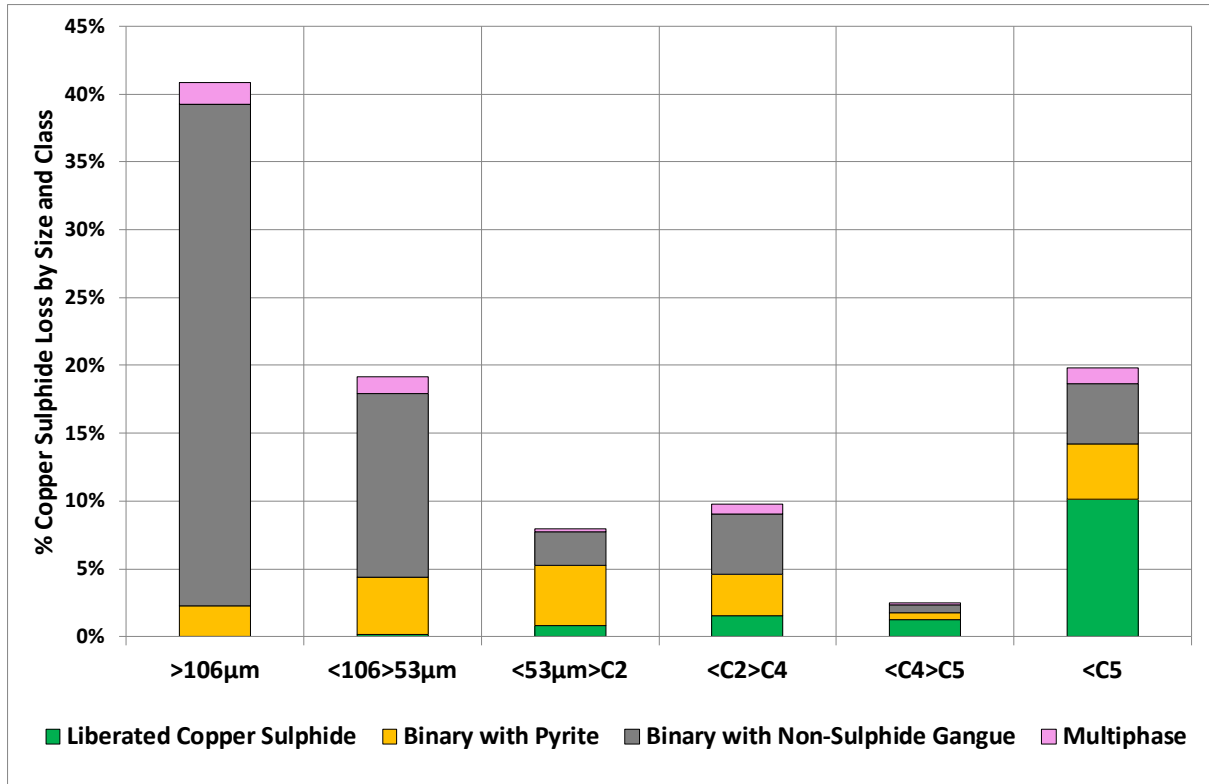


Figure 1 – Phu Kham flotation tailings copper loss by size and mineral association.

The copper sulfides generally occurred as less than 20 μ m particles locked in pyrite and non-sulfide gangue, as shown in Figure 2.

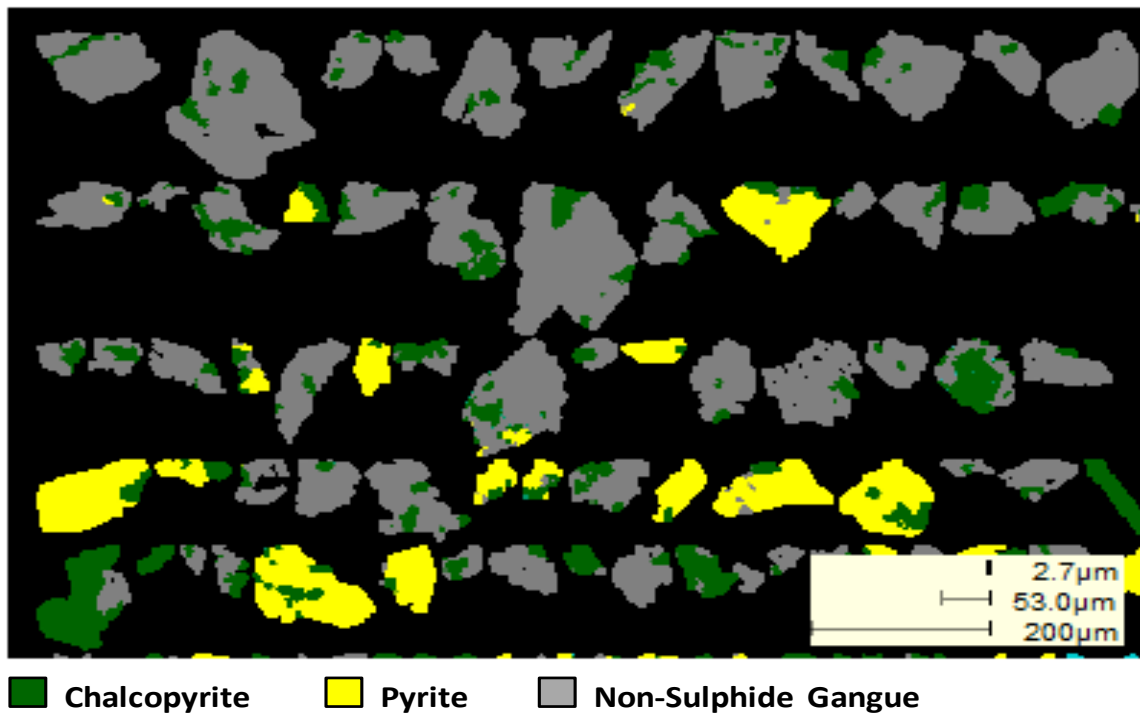


Figure 2 – Photomicrograph example of copper sulfide loss in rougher tailings.

The copper sulfide loss in the coarse (greater than 53 μm) material was targeted as the major opportunity for increasing copper recovery at Phu Kham with process development study work implemented to recover the coarse copper composite particles.

PROCESS DEVELOPMENT

During the period 2009 to 2011, study work was undertaken for development of a retreatment process for Phu Kham plant tailings to achieve a step-change in copper and gold recovery as described in Bennett et al, 2012. The process concept utilised bulk sulfide flotation with potassium amyl xanthate collector (PAX), with the bulk concentrate re-ground to 18 μm before cleaning flotation to produce a 3% copper concentrate. The concentrate would then be leached in an Albion leach process with copper cathode produced from the pregnant Albion leach solution via a solvent extraction – electrowinning process. The Albion leach residue would then be neutralised prior to carbon-in-leach processing to produce gold dore.

The process was developed to a level which indicated that up to 15% copper and 25% gold additional recovery could be achieved based on bench and bulk laboratory scale test work results. The test work program included a flotation pilot plant at Phu Kham in order to produce a concentrate from the plant tailings for pilot plant testing at the HRL/Core Resources Albion pilot facility in Brisbane. The pilot facility consisted of an M20 pilot IsaMill and a 1.5 t/h three stage flotation plant.

During the early stages of the pilot testing, the updated capital estimate for the tailings retreatment flotation and hydrometallurgical process plants was developed, with the updated estimate significantly higher than previous estimates. The change in the project cost base, coupled with the higher project risks for a tailings retreatment plant incorporating hydrometallurgical processes, led to a further review of alternative recovery improvement strategies utilising existing concentrator technology.

The first alternative reviewed was a ‘mainstream grinding’ process which classified the rougher tailings from the plant to recover the plus 53 μm fraction containing the low quality copper sulfide and gangue composites. This coarse fraction of tailings would then be reground to 80% passing 53 μm in order to further liberate the chalcopyrite prior to an additional scavenging flotation stage, with scavenger concentrate to report to regrinding with rougher concentrate. Although the process was highly predictable (as it was based on particle size rather than mineralogy) the capital and power costs for the coarse fraction grinding were very high due to the large tonnage of liberated and coarse silicate and pyrite gangue in the rougher tailings required to be ground for no value.

The second alternative was a ‘less selective’ sulfide rougher flotation process followed by increased regrind capacity in order to liberate the copper sulfide from coarse and low quality composites, and increased cleaner circuit capacity in order to treat the additional and lower grade rougher concentrate mass.

Absolute recovery improvement estimates based on the test work for the ‘mainstream grinding’ and ‘less selective flotation’ processes were similar at approximately 5-10 per cent for copper and gold. The advantage of the ‘less selective flotation’ process was that it was not a tailings retreatment process and project value could not be eroded with improvements in existing plant performance. Whilst it was acknowledged that the mainstream inert grinding process could also be performed on the plant feed, the less selective rougher flotation process had significantly lower capital and operating costs providing higher return on investment.

The disadvantage of the ‘less selective flotation’ process was that it is based on ore feed mineralogy, and therefore generated a higher risk of mass balance errors which would affect the process design and certainty of achieving expected copper and gold recovery.

Project selection

The pilot plant work program was changed to test the ‘less selective flotation’ process, which successfully demonstrated that significantly increasing the copper and gold recovery from roughing was achievable utilising PAX with over 90% copper recovery achieved at 30% mass recovery. The rougher concentrate grade was significantly reduced due to the bulk sulfide flotation regime employed during the test.

The pilot scale tests proved the concept that increasing mass recovery would lead to increased copper recovery, whilst highlighting the pitfalls of recovering too much liberated pyrite. Due to the non-selective nature of PAX compared to the existing Phu Kham Aero 9810 collector, the decision was made to utilise the extensive ore test work database which had used Aero 9810 as the collector for process design.

Due to the high variability in plant rougher copper and gold recovery between the low and high pyrite content ores, the rougher mass recovery likewise varied depending on feed mineralogy. In order to develop the design rougher mass recovery target, a review of the flotation recovery data was integrated into the project design process to allow for the variability in mineralogy of plant feed. Extensive rougher flotation rate test data that had been used to develop Phu Kham recovery models were statistically analysed in order to determine the mass recovery and copper recovery curves for differing ore types, from weathered and altered high pyrite (high S/Cu) material to chalcopyrite dominant and low pyrite (low S/Cu) primary material.

The statistical analysis showed that rougher copper recovery could be increased to 85% by targeting a maximum 20-25% mass recovery from rougher flotation across all ore types, as presented in Figure 3.

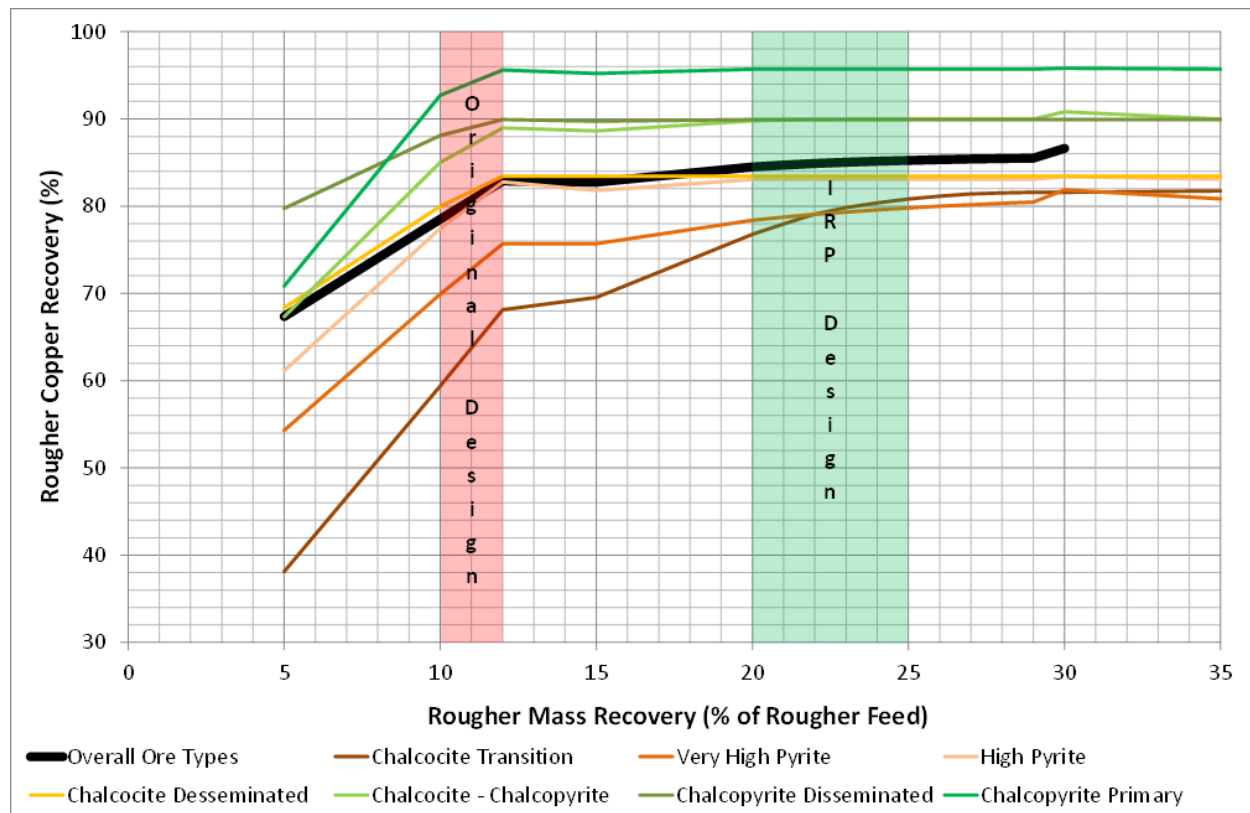


Figure 3 – Mass recovery against rougher copper recovery by ore type.

The ‘less selective flotation’ process design would increase rougher mass recovery for the slow floating high S/Cu ores while allowing flexibility for lower rougher mass recovery for the fast floating low S/Cu ores. This circuit flexibility was lacking in the original plant as the selective and capital

efficient process design had deliberately excluded bulk rougher flotation. It was recognised during the process development work that maintaining selectivity against liberated pyrite would still be required in roughing. It was considered that a true bulk sulfide flotation process would flood the Phu Kham cleaning flotation circuit when treating high pyrite ore and risk being unable to produce final concentrate within specification, hence the ‘less selective flotation’ process description.

As described previously in Bennett et al, 2012, mineral liberation analysis work was undertaken during the process development phase which determined that regrinding to approximately 20 µm would achieve the industry benchmark minimum of 80% copper sulfide mineral liberation in cleaning. The regrind specific energy requirement to achieve 20 µm size to cleaner feed was determined during the laboratory stirred mill and pilot test work.

Based on the preliminary mass and metal balance for the ‘less selective flotation’ process and the estimated increases in copper and gold recovery, equipment lists and capital and operating costs were developed which confirmed the robust economic value for the project, which was re-named as the Increased Recovery Project (IRP) and approved to progress to the next stage of development.

Although substantial laboratory and pilot scale test work had been performed to assess the benefit of the IRP, preliminary plant trials were planned during January 2012 to confirm the ‘less selective flotation’ process concept would work at full scale and confirm the copper and gold recovery values used for the investment case.

The full-scale plant trials were able to be performed as the design for the IRP flotation circuit was essentially the same as the existing Phu Kham circuit only with additional equipment added for concentrate handling downstream of rougher flotation. In order not to overload the downstream regrind and cleaning stages, the plant throughput was throttled back by 50% while the rougher flotation circuit was “pulled” very hard to produce a rougher concentrate mass recovery of over 20% of feed tonnes compared to the original plant design of 10-12%. During the trial the last two TC200 rougher cells were run with no air and low levels so that the plant trial residence time in the roughers matched the planned full-scale plant residence time to be installed during the PKU.

The preliminary plant trials were successfully completed in January 2012 over two 12 hour shifts, achieving approximately 10% additional copper recovery and up to 19% additional gold recovery results as presented in Table 1 below.

Table 1 – Phu Kham increased recovery project preliminary plant trial results.

Preliminary Trails		Trial 1 - 15 January 2012			Trial 2 - 27 January 2012		
		Baseline	IRP	Difference	Baseline	IRP	Difference
Rougher Recovery	(%)	83	92	+9	76	88	+12
Overall Copper Recovery	(%)	71	79	+8	68	79	+11
Overall Gold Recovery	(%)	48	66	+19	45	56	+11
Final Concentrate Grade	(%)	22.1	24.1	+2.0	24.2	24.8	+0.6

The trials were not used to confirm design mass recoveries or regrind size. Higher final copper concentrate grades were achieved despite the higher rougher mass recovery, although the minor difference against baseline values was not considered significant during the short trial. The ability to conduct a plant trial and successfully demonstrate a consistent metallurgical benefit at laboratory, pilot, and full plant scale prior to implementation underpinned the high degree of confidence in the IRP. The detailed mineralogy-based understanding of the causes of copper and gold loss to tailings developed over many years was fundamental in gaining this confidence.

Compression of the project schedule

During the early process design and project scoping, the project case for the IRP was determined to be compelling enough for the project development to be fast tracked with commitment for long-lead capital items approved before completion of the feasibility study. In addition to achieving the increased plant recovery earlier, reducing the project schedule had the added benefit that the PanAust construction team who were building the PKU would be able to commence work on the IRP immediately following completion of the PKU in August 2012.

The longest lead item for the project was the M10000 IsaMill with a delivery period of approximately 11 months. To place an early order for the mill the investment case for the project was developed to a preliminary level of accuracy at $\pm 30\%$ costs and with supporting data from the full-scale plant trials providing confidence, project development was approved by the PanAust Board on 23 February 2012. In early March 2012 the M10000 IsaMill was ordered as a turnkey package, including all of the peripheral equipment to the mill along with control and instrumentation required. The package also included all steelwork but specifically excluded mill foundations.

The project schedule was developed to include the completion of a front end engineering phase (FEED) level design and cost estimate by May 2012 using an external engineering consultant. The FEED phase fixed the design criteria, mass and water balances and tagged equipment lists which were then be used in the detailed engineering and procurement (EP) phase.

During the EP phase, the engineering consultant was responsible for engineering design and equipment selection up to the recommendation for purchase decision, with approval and procurement performed by PanAust staff in Brisbane and Laos. To avoid issues identified during the PKU with delays in receipt of vendor data, the engineering consultant was responsible for liaison with vendors once the purchase order was raised to ensure the engineering consultant maintained schedule control.

The basic project schedule for the IRP is presented in Figure 4.

Project Stage	2012				2013			
	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Feasibility Study/Front End Engineering								
Long Lead Procurement								
Engineering and Procurement								
Construction								
Commissioning and Ramp-up								

Figure 4 – IRP project schedule.

PROJECT DESIGN

Feasibility

In order to develop a business case for the project, initial process design was developed by PanAust during the pre-feasibility stage along with factored capital and operating cost estimates. The cost estimate was considered to be at a higher level of accuracy than a normal pre-feasibility stage due to the recent completion of the engineering for the PKU and construction of the Ban Houayxai CIL plant by PanAust Ltd.

In addition to process design criteria, the equipment lists and brownfield plant layouts and integration plans were developed to a detailed level by PanAust so that the FEED stage was mainly focused on engineering detail rather than design. Plant survey work was performed during this stage to improve the mass balance which was used to confirm pump and piping specifications. The flow diagram showing the additional equipment installed as part of the IRP is shown in Figure 5.

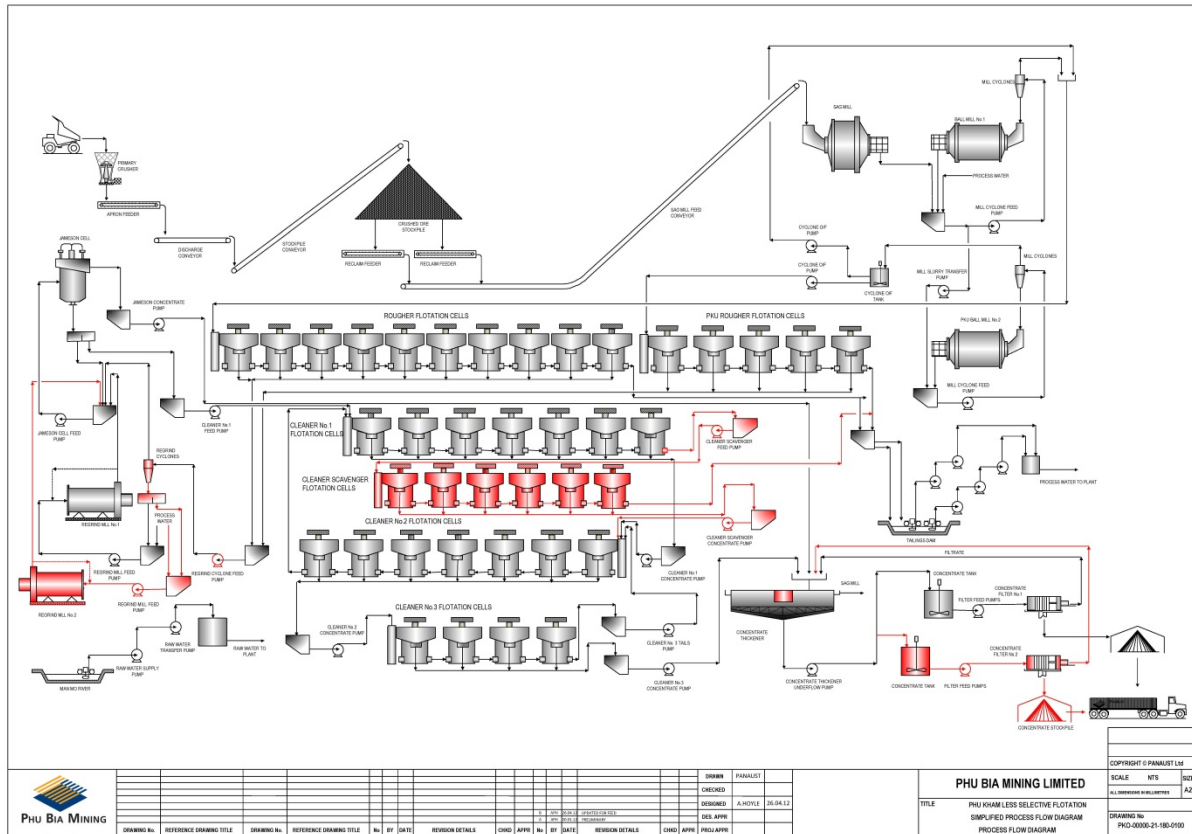


Figure 5 – Initial process flow diagram for the Increased Recovery Process.

Front-end engineering

GR Engineering Services (GRES) was selected as the FEED engineer for the IRP. During the FEED phase a number of details in the plant design were resolved which resulted in minor changes to the process flowsheet. The most significant change was the addition of a parallel cleaner bank instead of additional series cells to cleaner 1. This was changed mainly due to limited dart capacity in the cleaner 1 cells as well as the cost associated with upgrading the cleaner 1 feed pumps. The change resulted in a lower project capital cost as capital was moved from debottlenecking the existing cleaner 1 cells to installing a parallel bank of cells. The process flow sheet was also simplified due to the change. The change also reduced the associated tie-in time for construction as the new equipment was able to be added in discrete modules and did not impact on the existing plant which was a key learning from the PKU for brownfields plant expansions.

In parallel with the FEED phase, additional metallurgical test work was performed in order to assess the effect of the expected finer concentrate size on the settling rate of the existing concentrate thickener. The settling tests performed on concentrate produced from the variability trials showed that the existing 15 m concentrate thickener would be able to handle the concentrate produced from the IRP, however settling efficiency could be improved over current operation by reducing the amount of entrained air in the feed. Site investigation determined that air was being entrained through the trash screen collection box, so the decision was made to add a de-aeration box prior to the thickener in order to improve thickener performance of the thickener.

Challenges in brownfields plant upgrades were highlighted during the FEED phase with the marriage of new equipment with existing equipment which cannot be relocated. Specific design principles were included to minimise capital and operating cost based on the learnings from the original plant design and PKU. The design was developed with a focus on the use of gravity flow wherever possible and gravity lines were used in the majority of tie in locations despite the additional engineering design

required. This design philosophy increased the complexity of the design however increased operability was achieved.

The existing plant concentrate production capacity was analysed, with the existing filter able to produce concentrate below the transport moisture limit at a rate of approximately 330,000 t/y. This was determined to be marginal at the increased concentrate production rates as shown in Figure 6. This was a key risk for the project as the filtration rate was expected to decrease due to the finer concentrate size distribution. To eliminate this risk a second filter was included in the design. Although the second filter had fewer plates than the original filter, the same plate size was maintained to allow common spares with the existing filter.

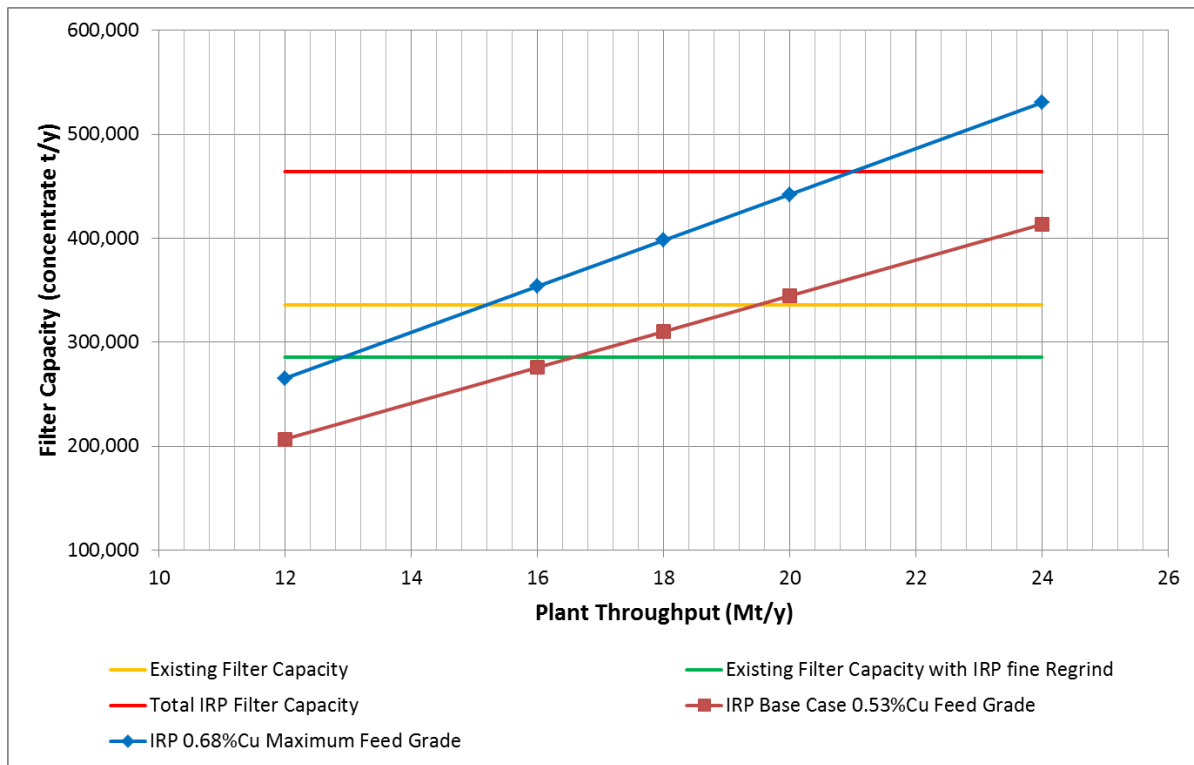


Figure 6 – Filter performance against increased recovery concentrate production targets.

A design review was conducted during the FEED to look at installing a second filter in the existing shed without disruption of the existing filter operations. The review determined that disruption would be significant, so an additional smaller storage shed was included in the design to allow for additional storage capacity during the wet season. The design of the new storage shed and filtration area was leveraged from the existing engineering designs to reduce costs for installation. Some minor modifications were made during the FEED phase to shorten the length of the shed in order to reduce storage capacity in line with the additional concentrate production.

An additional item of risk which was identified during the FEED was the second cleaner flotation cell tailings darts capacity not having sufficient margin above maximum flows predicted for the IRP. Due to the lack of mitigation measures available, contingency was allowed for in the estimate to install inter-cell bypasses should the flow rate significantly exceed darts capacity. It was anticipated that cleaner 2 would become the circuit bottleneck in the IRP circuit, and if liberation was not achieved in the regrind mills, a recirculating load of middlings particles would build up between the cleaner 2 and cleaner 3 banks. This would result in recovery losses or below specification concentrate grade.

The FEED phase was completed on schedule with the capital cost reduced by \$10M due to additional savings identified during the design and a reduction in the project contingency due to better project definition. The reduction in capital was achieved despite an increase in the scope of the project in

order to minimise risk. As the project was brownfields integration, additional capital was allowed in the estimate for resolution of legacy pumping and spillage issues which had affected the Phu Kham concentrator since the original design and PKU.

Following completion of the FEED phase the final equipment list was confirmed along with the changes to the process flow diagram (Figure 7) in order to optimise both cost and design:

- Twelve 400 mm diameter regrind cyclone pack
- M10000 IsaMill
- Seven Outotec TC70 Tank Cells
- Thickener de-aeration box
- Concentrate filter feed tank
- 40 plate Ishigaki filter
- Concentrate storage shed

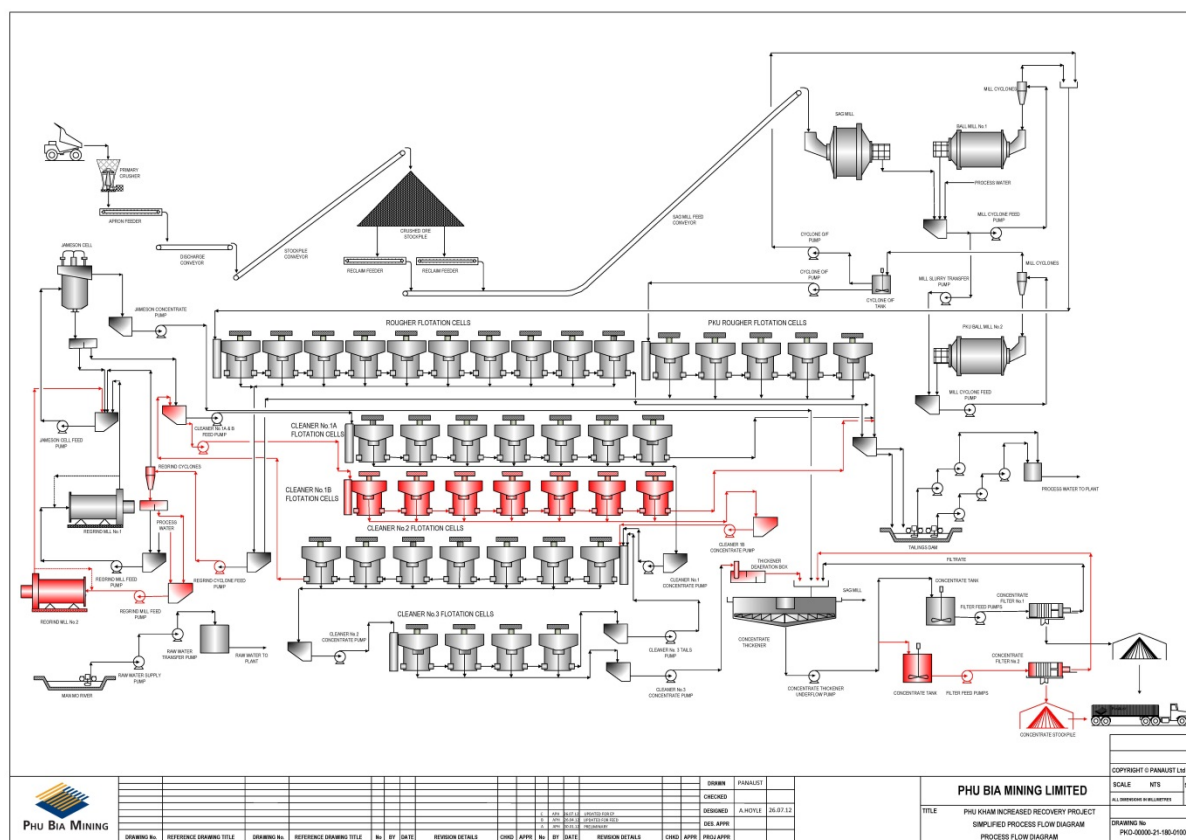


Figure 7 – Final process flow diagram for the Increased Recovery Project.

IRP COMMISSIONING AND RAMP-UP

Plant tie-ins were performed progressively as equipment was delivered to site and in line with the project schedules which were integrated with the planned shutdown schedules. The majority of construction and tie-in activities were completed by March 2013.

IsaMill commissioning

Commissioning of the new IsaMill concluded with a 72 hour acceptance test which commenced on the 26th of March 2013 with both the new and old mills run in parallel for the first time. Operational conditions during the period were identical for each of the mills. During the period, the new IsaMill

was run over a range of power draws, with the ultimate design power of 2500kW being achieved. During the test no external signs of any problems were detected, however upon completion of the test period, the mill was inspected and damage to the shell liner (Figure 8), feed end liner and three discs was identified.

The root cause of the damage to the mill liners was overheating of the mill slurry due to the high recycle of IsaMill discharge to the feed. The high recycle was caused by insufficient flow from the regrind cyclones which were exacerbated by a biased flow from the underflow splitter box to the existing IsaMill. The new regrind cyclones had also been modified by operations due to the need to run them prior to IRP commissioning and did not have the design spigots installed prior to start-up. The high temperature in the IsaMill would have normally been detected by the process control system however the resistance temperature device (RTDs), which were installed to protect the mill in such instances of low feed, were found to be providing an inaccurate output. The RTDs were replaced along with the shell liner, and the discs and feed end liner were reused.

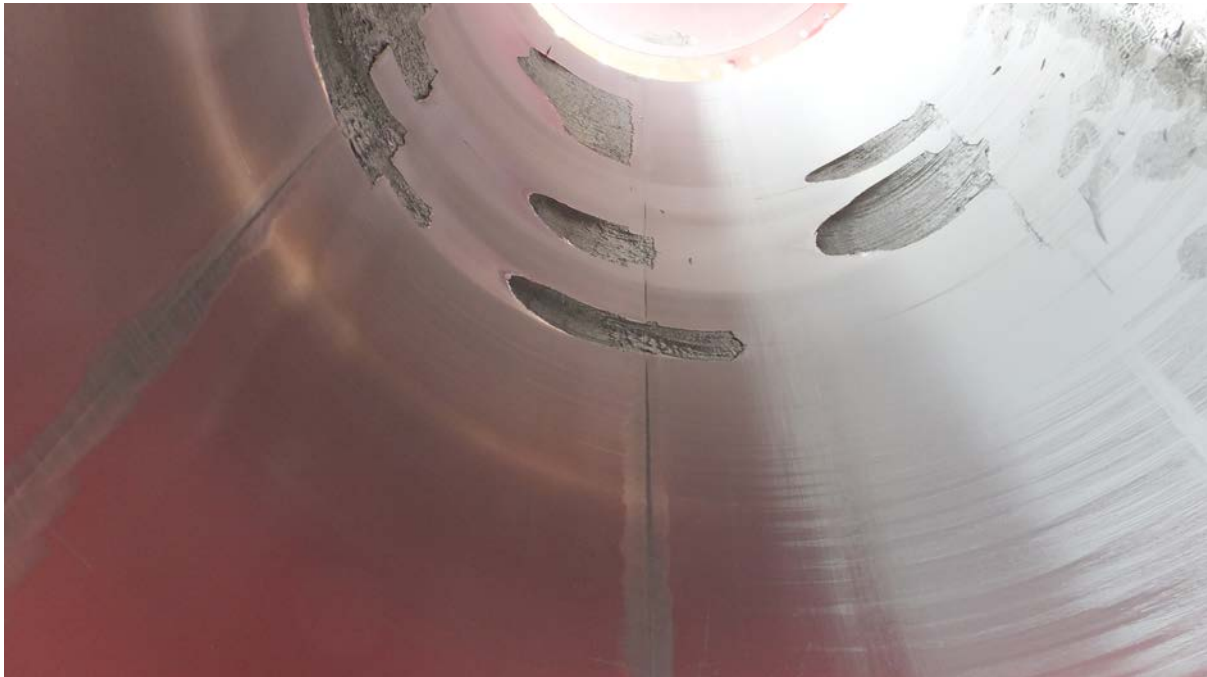


Figure 8 – Damage to the shell liner during the first 72h acceptance test.

The mill acceptance test was rerun with additional wear identified in the regrind mill due to media compression. The media compression had been caused by a high slurry feed density which was attributed to an incorrect and high solids specific gravity input into the nuclear density gauge.

In addition, the output value of the density gauge had been mistakenly assumed to be a calculation of percent solids. This meant that the density in the mill was approximately 8% higher than the actual density target, with the subsequent high viscosity in the mill causing the media compression. In order to ensure that this didn't recur, the target pulp density was decreased to 45% solids and the viscosity was monitored at hourly intervals with a Marsh Funnel.

Once the control measures were put into place the decision was made on 18th of April to run the mill for a 24 hour test. At the completion of this test, the mill was inspected and no further wear detected on the liners. The final acceptance test was then performed for 48 hours at power draws of up to 2600 kW. On the 21st of April the mill was inspected for the final time. No wear had occurred over the final trial period and the mill was handed over by Xstrata Technology personnel to PanAust operations.

Plant ramp-up

Even with the modular design of the IRP which aided constructability, the integration with the existing circuit was extensive. Existing equipment, particularly slurry pumps, were utilised within the new circuit.

During the ramp-up phase a number of issues with slurry pumping were encountered. These were mainly confined to original pumps which had been upgraded or had duty changed as part of the IRP or earlier improvement projects. The IRP construction and pre-commissioning phases had not generally considered the condition or status of existing equipment and as it had been previously operating satisfactorily, no program of inspection was initiated prior to IRP commissioning. This resulted in delays in reaching design capacity.

As an example, the rougher concentrate pump system had the common discharge pipeline upgraded to match the higher IRP design flow-rates, with the existing duty and standby pumps and motors assessed as suitable for the higher design duty. During the ramp-up phase the pumps were not able to reach design flow-rate which resulted in both the duty and standby pumps being run to meet design flow-rate. Investigation of the pump variable speed drive systems discovered that the original overload and frequency limit settings had been retained for the original motors, although the motors had been upgraded prior to the IRP. These settings were limiting the pump speed and power draw to a level well below maximum operating capability. Once the limit was removed the pumps were found to be easily able to meet design flow-rate.

Ramp-up was also hindered by power supply continuity to site, as there was often insufficient power able to be supplied from the hydroelectric schemes which provide all power to the Phu Bia Mining operations. This was due to low water levels in the reservoir at the end of the dry season coinciding with the IRP ramp-up and extended planned maintenance periods on generating units. Operation response to power limitation was to load-shed power across the plant with power draw being lowered in the ball mills and IsaMills in order to maintain throughput. This increased primary grind size and regrind size lowering copper mineral liberation and reducing plant copper recovery. The problem continued until June with the onset of the wet season replenishing reservoir levels.

IRP PERFORMANCE

Rougher recovery

Despite the described issues during ramp-up, rougher mass recovery was quickly increased to 20% of feed mass following completion of commissioning.

At an early stage of ramp-up the operating strategy was to effectively “fill up” the cleaner circuit by recovering as much mass as possible from the rougher banks. Cleaner circuit performance and overall copper recovery were generally compromised by this strategy because excessive gangue was recovered in roughing requiring rejection in cleaning, and the regrind product size to achieve liberation targets was not being met. In addition the feed ore quality was generally challenging during ramp-up due to the requirement to mine ore from the upper areas of the pit which had very high pyrite and clay content, and a high proportion of easily oxidised secondary copper sulfide species.

During this period, operational strategies were redefined and training undertaken with site personnel to ensure that copper recovery was maximised from the rougher banks and the cleaning circuit was not overloaded with liberated gangue. Back-to-basics improvements were undertaken on operating parameters; in particular work was completed with Outotec on optimising the rougher cell level and air addition profiles which were then set to avoid excessive water and gangue recovery and improve low quality coarse composite recovery.

Process control improvements were also implemented during the ramp-up period, in particular commissioning of automated rougher mass recovery control through use of froth velocity measurements and level and air controls.

With the improvements in control, equipment performance and operating knowledge for the new circuit, the ramp-up of the IRP was mainly completed by June 2013. Copper rougher recovery increased from an average of 78% to 83% within the first month of operation, in line with expectations. The increase in gold rougher recovery was very significant from an average of 55% to 78% as shown in Figure 9. Gold recovery at Phu Kham had always been highly variable, which led to a conservative estimate of gold recovery increase used for the IRP investment case, however the observed increase in gold recovery exceeded the results achieved during the plant trials and even the most optimistic estimates. The reasons for the improvement were likely the improved rougher recovery of lower quality copper sulfide and pyrite composites containing gold and improved fine gold recovery.

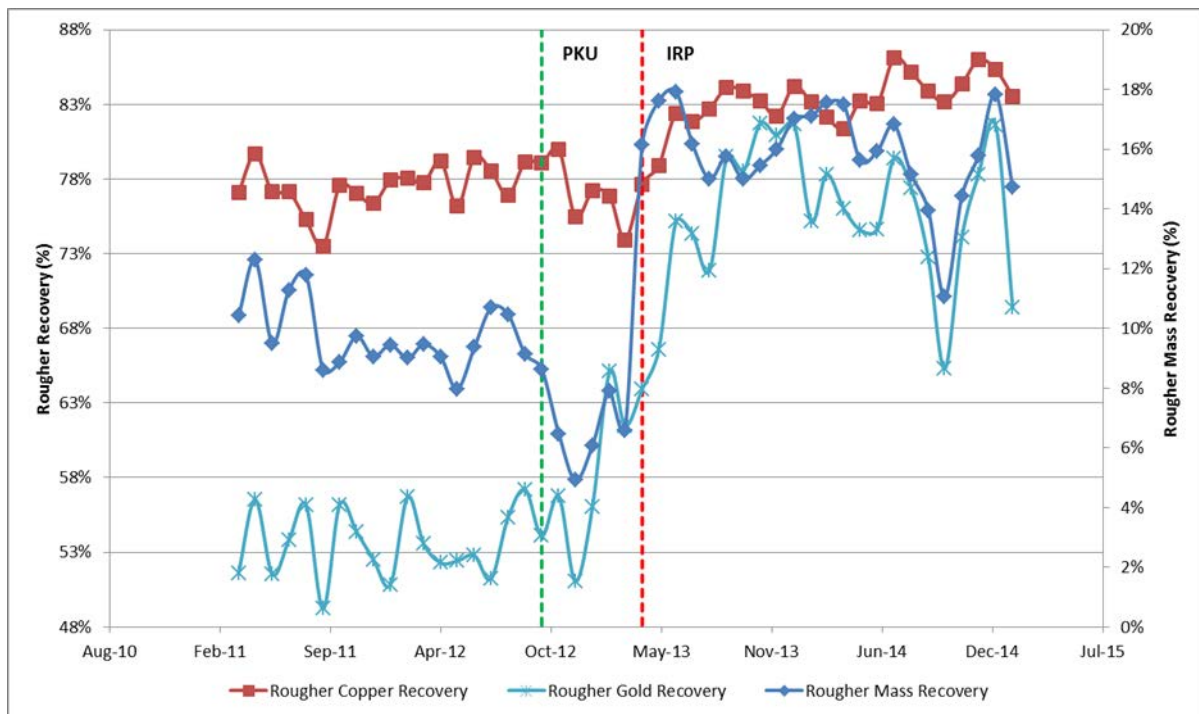


Figure 9 – Rougher mass, gold and copper recovery following the Phu Kham upgrade and Increased Recovery Project.

Once operational control was improved and selectivity against liberated pyrite and non-sulfide gangue was improved in the rougher circuit, the increased copper recovery was confirmed through a consistent reduction in the quality of the copper sulfide – gangue binary particles in the rougher tailings. Following the implementation of the IRP, the copper sulfide content of these particles decreased from approximately 15% to 7% copper sulfide indicating that the IRP target low quality copper sulfide – gangue composite particles were being recovered (Figure 10).

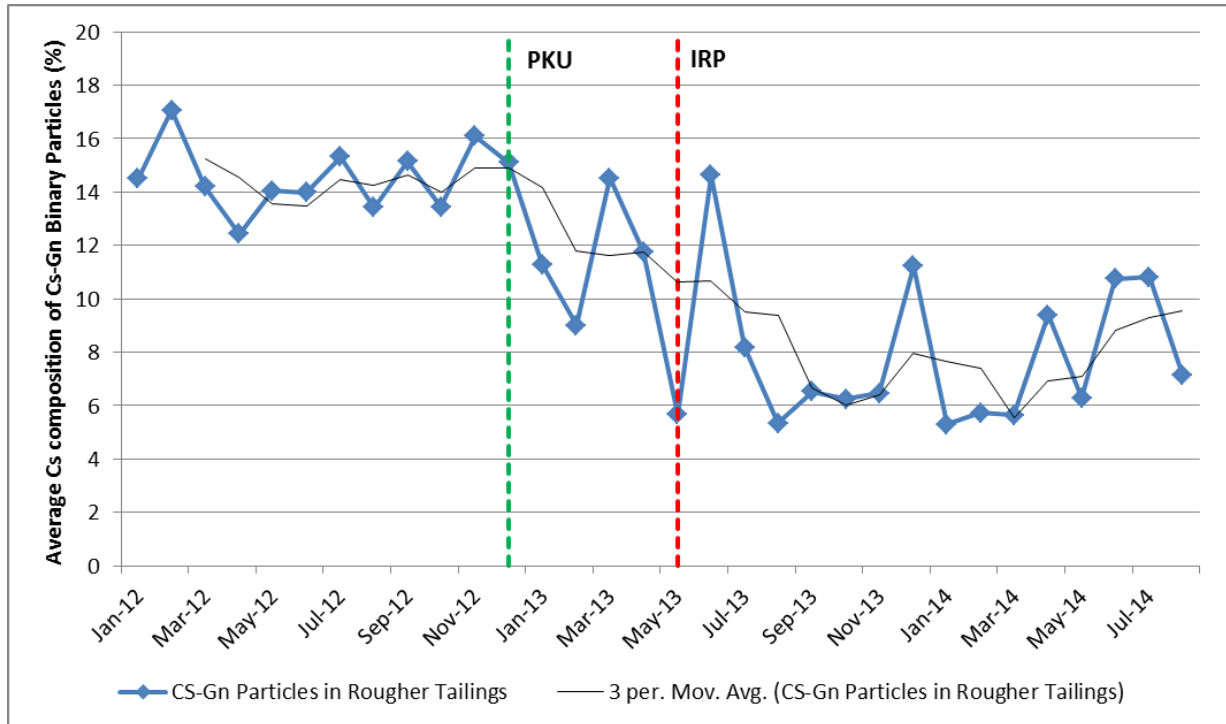
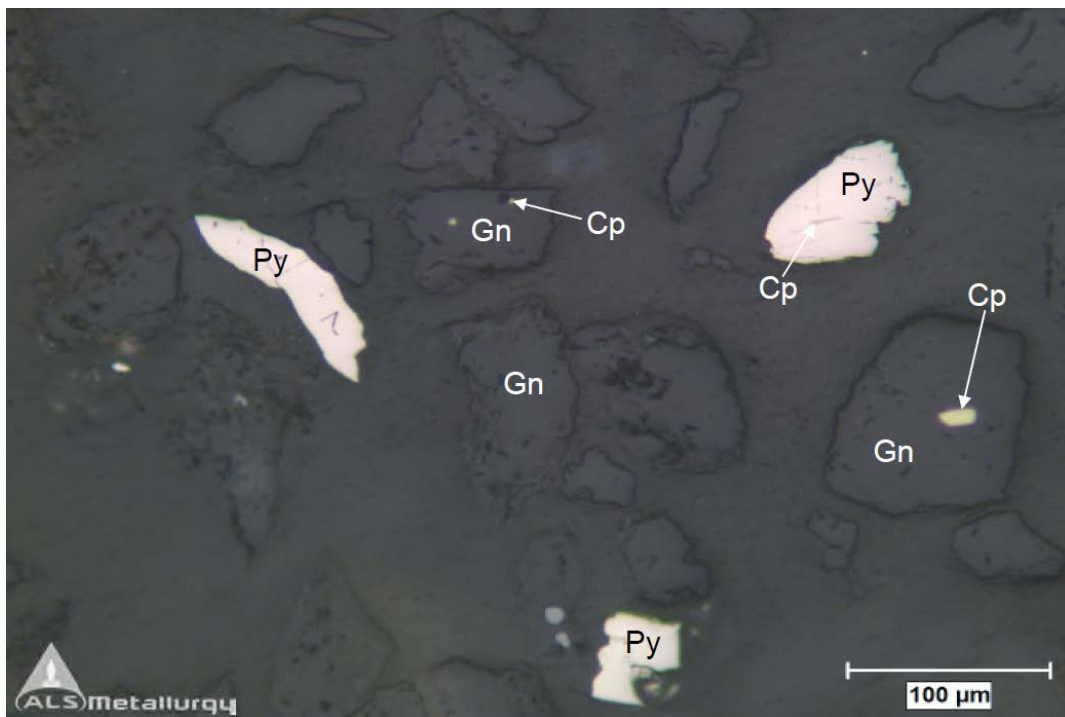


Figure 10 – Quality of the copper sulfide-gangue binary composites in the rougher tailings following the PKU and IRP.

A photomicrograph of typical rougher tailings is presented in Figure 11, indicating the lean quality of copper sulfide – gangue composites following the IRP.



Cp = Chalcopyrite, Py = Pyrite, Gn = Non-sulfide Gangue

Figure 11 – Photomicrograph of the typical rougher tailings following PKU and IRP.

Overall recovery

The ramp-up in overall recovery was achieved in June 2013 with 76.4% average weekly recovery being achieved. This was 3.8% above the baseline PKU recovery of 72.6%. In addition during this time the variability of the overall recovery decreased with the standard deviation of weekly recovery reducing from 6% to 4.6% of the mean. As presented in Figure 12, the number of weeks with recovery below 70% has decreased over time as the operational strategies implemented during the IRP have been further reinforced.

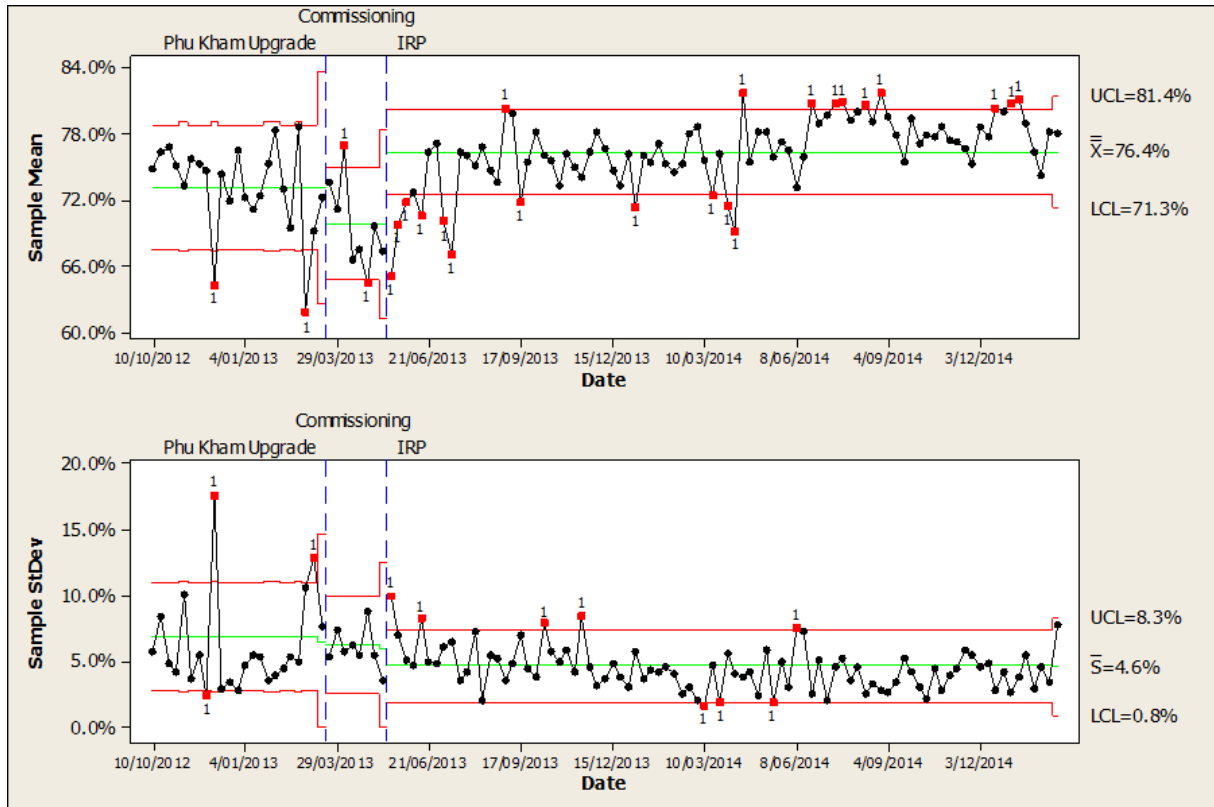


Figure 12 – Overall copper recovery following implementation of the IRP.

As for copper, initial increases in the rougher gold recovery did not correspond to a significant increase in overall gold recovery. Gold losses in the cleaner tailings were investigated during this period in order to determine the mechanisms of gold loss from the cleaner circuit (Roulston & Johnston, 2014).

Analysis concluded that although 25% of the identified gold containing particles lost from the cleaner tailings were liberated gold, the majority of these particles were less than 10 microns in size. Almost 50% of the particles were liberated or high quality composites with pyrite or copper sulfide minerals. The other 50% of gold particles lost were in extremely low quality binary composites with pyrite (Figure 13). These composites were required to be rejected in order to maintain acceptable final copper concentrate grade. As gold particles even in binary composites were fine (Figure 14), gold recovery to the final concentrate relies on liberation from pyrite due to the high lime pH environment for pyrite depression, which also reduces fine gold flotation kinetics.

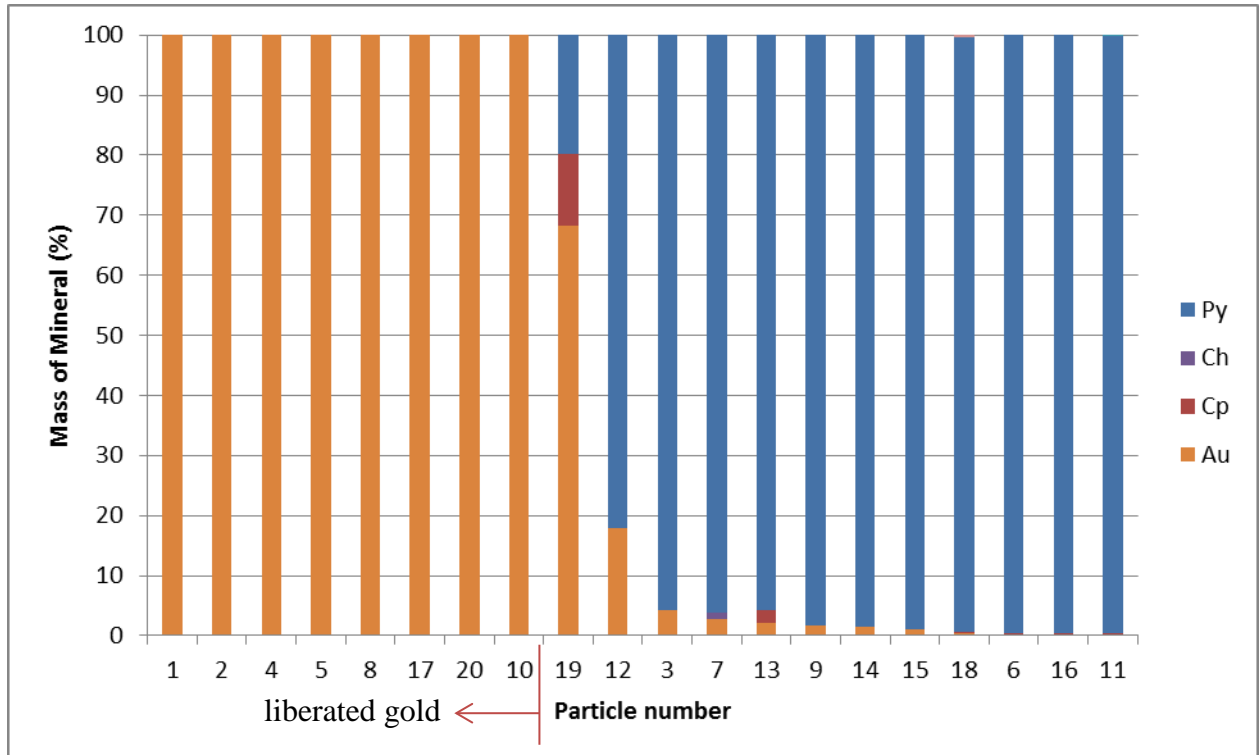


Figure 13 – Quality of the gold particles lost from the cleaner tailings.

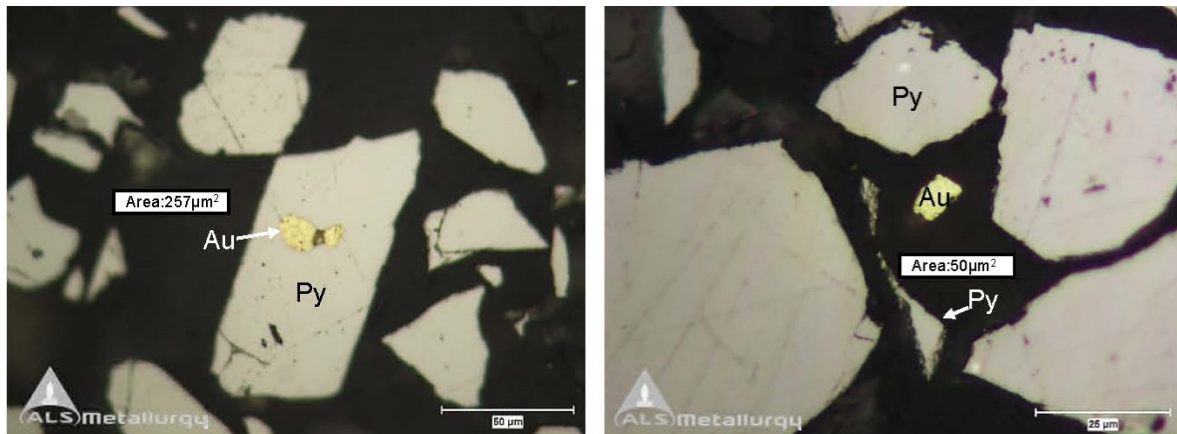


Figure 14 – Examples of poor quality composites and fine liberated gold losses in the cleaner tailings.

With improvement in plant performance, overall gold recovery increased, with weekly gold recovery averaging over 50% by September 2013 (Figure 15). No decrease in the variability of the gold recovery was achieved because the gold recovery is mainly affected by the variable mineralogy, liberation, and gold feed grades to the plant.

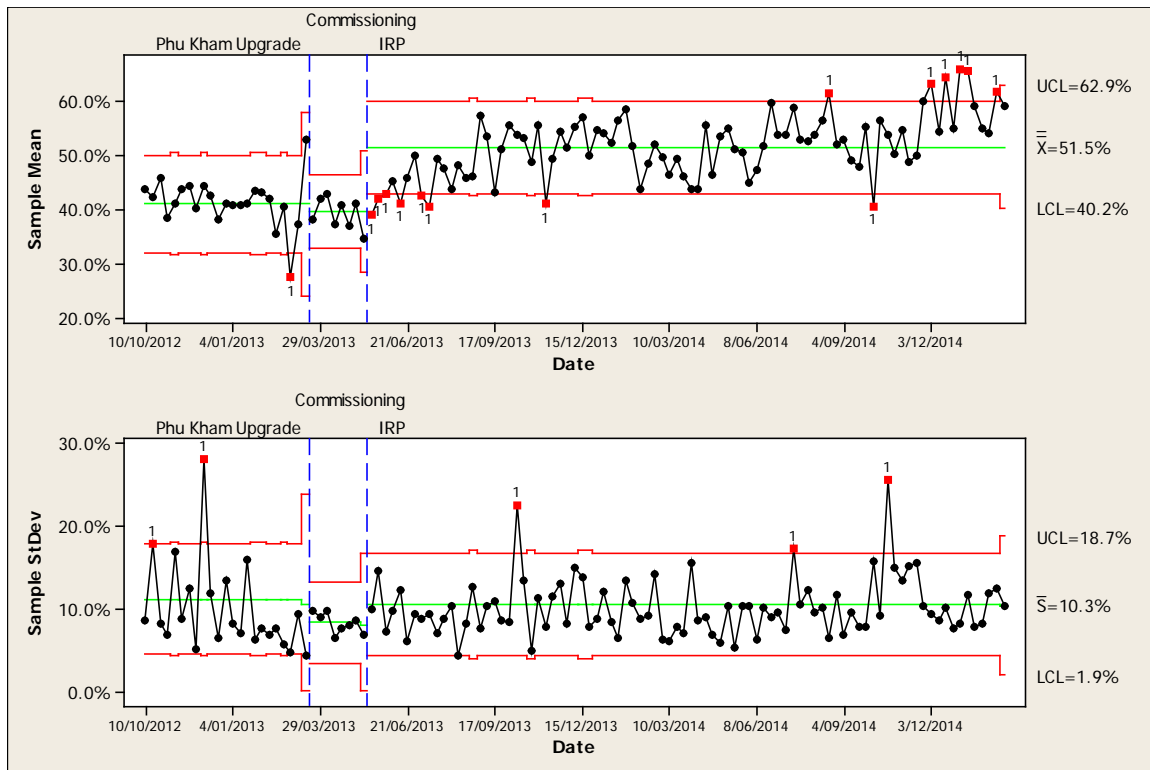


Figure 15 – Overall gold recovery following commissioning of the IRP.

High gold recovery was maintained throughout the year following the IRP even though the plant experienced a significant change in the gold head grade from 0.3 g/t to 0.2 g/t from September 2013 to September 2014 (Figure 16). Once head grades returned to normal levels in late 2013, overall gold recoveries of 60% were being achieved which was almost a 20% increase over the pre-IRP recoveries.

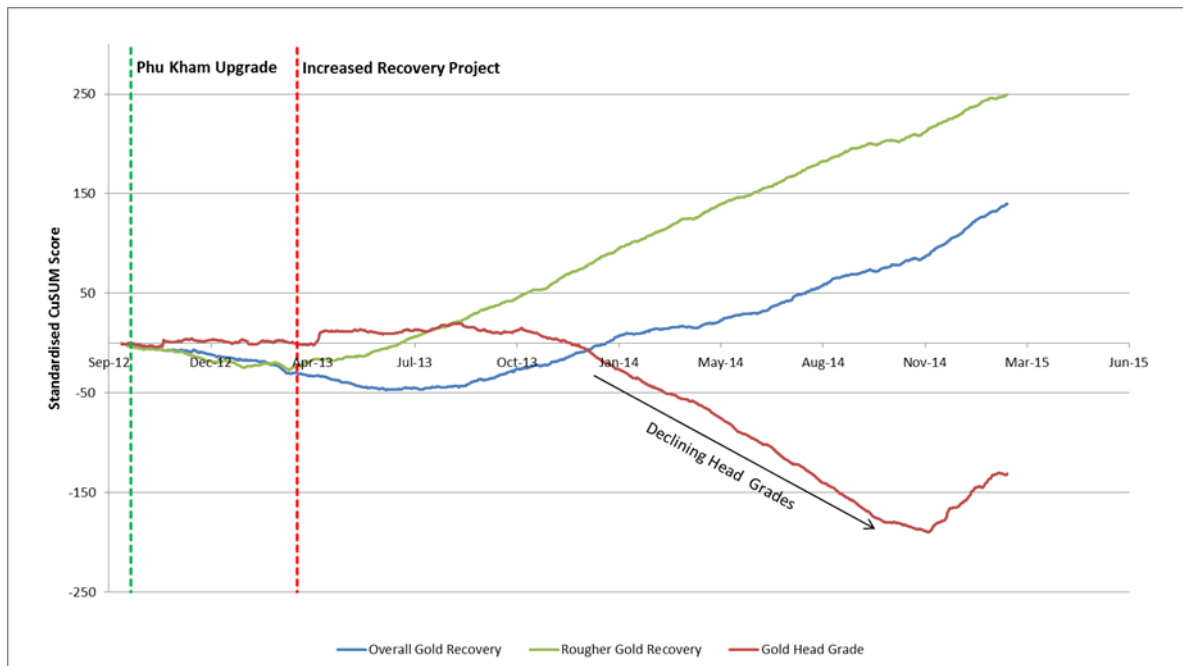


Figure 16 – CUSUM chart for gold showing increase gold recovery despite declining head grades.

Recovery against investment case

The initial analysis of the plant data shows that the average increase in copper recovery from initial analysis of the data was less than the 6% which was originally predicted for the project and the increase in gold recovery was above the original 6% target. This was despite other design targets (e.g. mass pull rate and rougher recovery) within the plant being achieved. The source of the copper recovery difference was due to a number of other operational and ore quality factors which were impacting on production during the period as discussed below.

It can be seen in Figure 17 that during the period of the ramp-up, the quantity of pyrite in the ore as measured by the sulphur to copper content fraction (known as the S/Cu ratio) increased from over 16 during the PKU ramp-up period to 21 during the IRP ramp-up. In addition head grades declined over the period which allowed the throughput to be increased in order to maintain the copper production rates. Pivotal in this increase in mill throughput was the implementation of new plant control strategies (Baas, Bennett, & Walker, 2014) along with the additional rougher concentrate, regrind and cleaner capacities provided by the IRP.

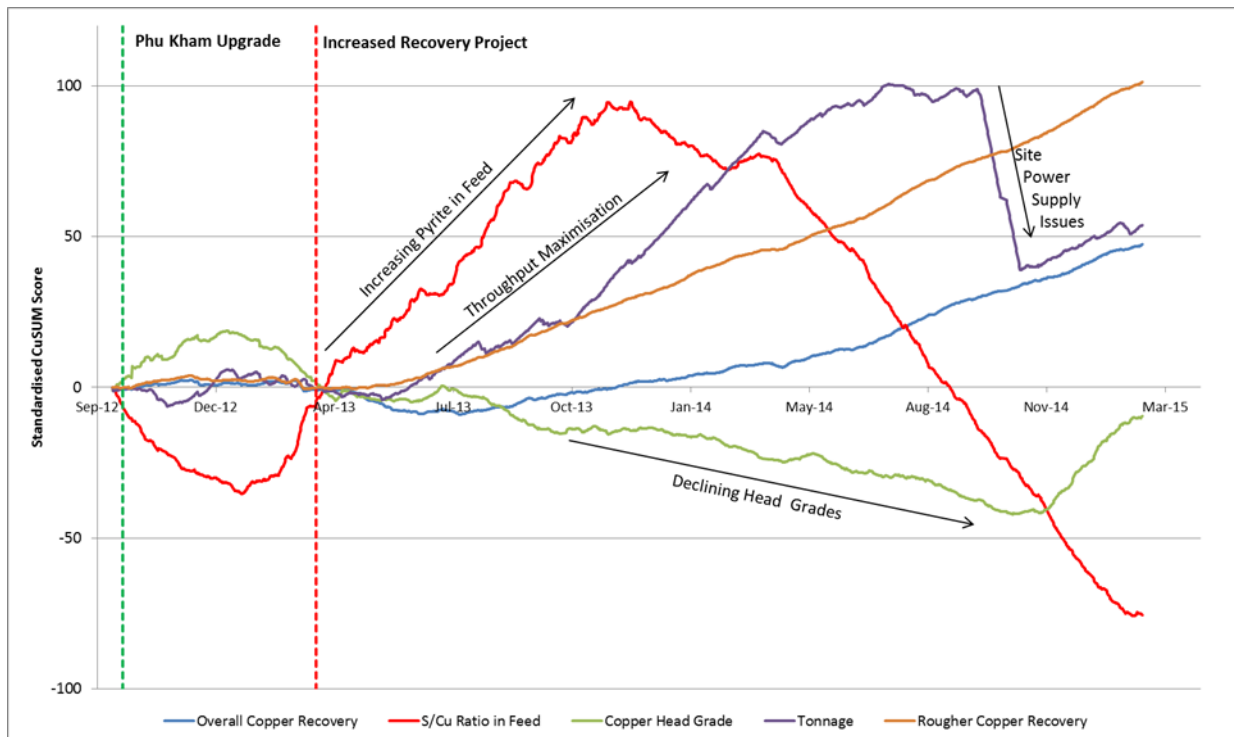


Figure 17 – CUSUM chart for copper recovery showing changes to operation and ore quality indicators.

The S/Cu ratio has been used at Phu Kham in recovery modelling as a proxy for the amount of pyrite in the ore, which is the dominant driver of metallurgical performance. Increased S/Cu ratios result in decreased copper recovery, due to the requirement to be highly selective against pyrite. On a S/Cu basis, the increased copper recovery achieved by the IRP varies from 7% at a S/Cu ratio of over 30 to 4% at a S/Cu ratio of less than 20 (Figure 18). The increase achieved by the project was determined to have reached the 6% goal when the ore quality schedule was compared on a like for like basis.

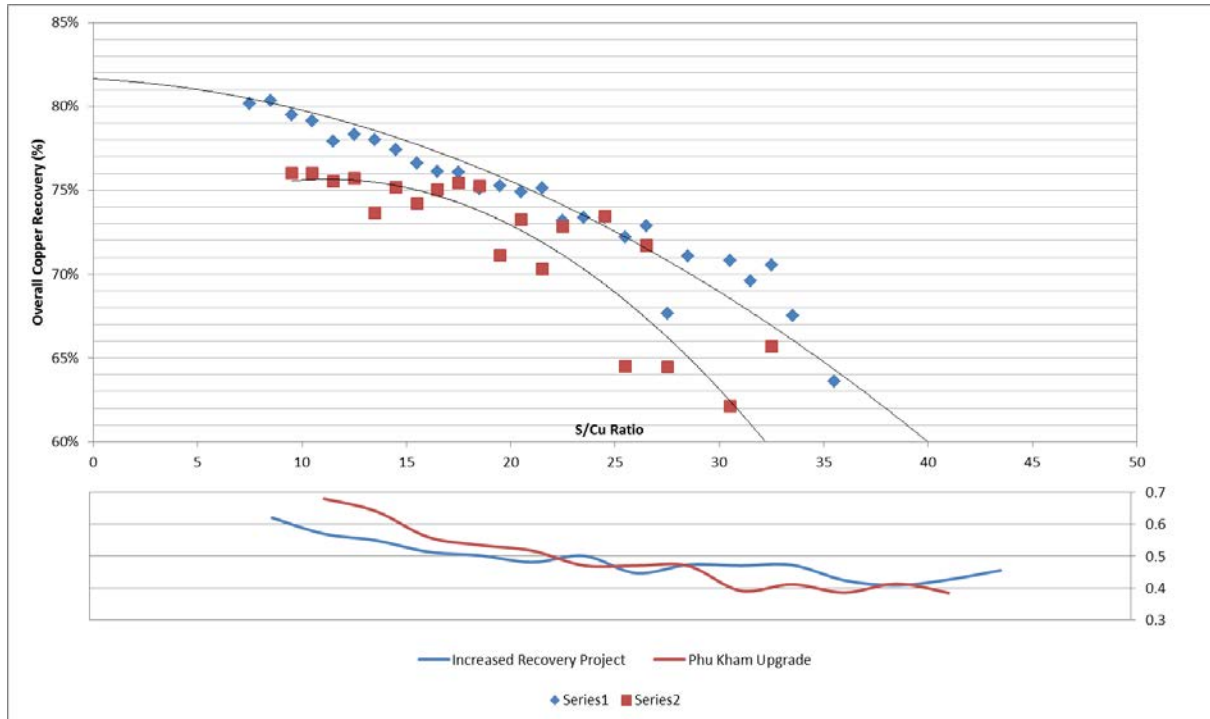


Figure 18 – Copper recovery against S/Cu ratio in plant feed.

FUTURE OPPORTUNITIES

Since the IRP has been implemented, a number of further opportunities for recovery improvement have been identified. Major additional improvement in reducing the variation in overall recovery has since been achieved through implementation of a mass recovery control system which utilises froth cameras on the rougher bank. This system has reduced surging in rougher concentrate make which has allowed the cleaner circuit to be operated in a more stable manner.

Other opportunities for improvement at the plant are focused on improved collectors for the rougher stage. Due to successful use during the plant scale trials, the IRP design utilised the existing Cytec 9810 collector (which is selective against pyrite) but at higher dose rates. Although consumption was higher than normal, the investment case was not affected by collector choice and the incumbent collector offered the lowest risk during project commissioning. The IRP design always envisaged that there might be a less selective collector suited to the process which would allow cost to be reduced whilst meeting plant performance goals. This is currently being investigated by site metallurgists with an ethyl and amyl xanthate mixture showing promise at low addition rates.

Finally, work is currently in progress to improve cleaner circuit pulp chemistry following regrind. This body of work was identified by site metallurgists during Jameson Cell optimisation test work where it was found that the copper sulfides in regrind cyclone overflow exhibit significantly faster flotation kinetics than the IsaMill regrind discharge. This has been attributed to oxygen depletion via reaction of pyrite in the closed milling system. Routine analysis of the pulp chemistry has been implemented and a trial using peroxide (dosed to the regrind discharge to modify the pulp redox potential in the cleaner feed) is currently being advanced.

CONCLUSIONS

The Phu Kham Increased Recovery Project was completed under budget and schedule, resulting in increased copper recovery of 5% and increased gold recovery of at least 10% over the first year of operation. The increased recovery was achieved despite decreasing ore quality over the period. In addition the extra cleaner capacity allowed throughput to be increased in order to increase copper feed

tonnes to the plant despite the lower head grades. The project was a major success in utilising monthly mineralogical association and liberation data to identify major copper loss classes from the circuit and implementing a sound strategy to minimise the losses.

The resultant step change in process recovery has not diminished the focus on plant improvement with future work planned to optimise the cleaner circuit through better operational strategies and analysis of the regrind chemistry.

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