Development and implementation of a novel pressure leach process for the recovery of cobalt and copper at Chambishi, Zambia

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Synopsis

The Chambishi Metals Cobalt Plant in Zambia has recently installed a novel oxidation/precipitation pressure leach process as part of the COSAC (CObalt from Slag And Copper as by-product) Project. An iron-cobalt-copper alloy, produced by the carbothermic reduction of slag, is atomized and pumped to the COSAC Leach Plant, where cobalt and copper are recovered as dissolved sulphate salts, and iron is rejected primarily as hematite. Cobalt and copper are subsequently recovered as cathode metal in the existing Chambishi operation by conventional electrowinning. This paper examines the challenges faced during the design and subsequent successful commissioning of the COSAC Leach Plant. The challenges discussed include alloy handling, autoclave design, autoclave cooler design, residue liquor recovery and the water balance design.

Keywords: Chambishi, Avmin, COSAC, Zambia, Copperbelt, cobalt, copper, alloy handling, pressure leach, autoclaves, cooling, filtration.

Introduction

The Copperbelt region of Zambia (and the DRC) is well known for copper mining since the first discoveries of copper in the early years of the twentieth century, and significant smelting and refining operations can be found in this area, including those at Nchanga, Mufulira and Nkana. The Copperbelt is also one of the major cobalt mining areas of the world, although electrolytic cobalt made an impact somewhat later than copper, with the first plant, the Shituru refinery of Gécamines, coming on-stream in 1947[1,2], closely followed by Nkana (formerly Rhokana, Rokana) in 1952[3], the Luilu plant in the early 1960s[3], and finally Chambishi in 1978[3].

The Chambishi Cobalt Plant, located near Kitwe, commenced operation with a production capacity of 2500 tpa electrolytic cobalt metal in 1978[4,5], after several years of producing a cobalt intermediate material that had either been stockpiled or treated at the nearby Nkana (Rokana) Cobalt Plant. In 1998, Anglovaal Mining (Avmin) of South Africa purchased a 90% holding of the refinery,[6] together with the nearby Nkana slag dump[7]. In order to process the estimated 20 million tonnes in the slag dump, averaging 0.76% cobalt and 1.2% copper, Avmin undertook to expand the production capability of Chambishi to over 6000 tpa of cobalt, and at the same time, embarked on the development of a process, denoted COSAC (CObalt from Slag and Copper as by-product), capable of treating this material. The COSAC process is an improved modification of that[8] developed by the then NCCM (subsequently ZCCM) in the 1970s, wherein the slag undergoes carbothermic reduction in an electric furnace to produce an iron-copper-cobalt alloy, which is then atomized and leached under oxygen pressure to generate a solution suitable for the recovery of copper and cobalt metals within the framework of the existing Chambishi plant.

The COSAC process is unique to the base metals industry, being the first of its kind to recover cobalt and copper from discarded slag. As a result, this proved to be a technically challenging project, which required a thorough understanding of the process, equipment capabilities and a practical yet conservative operating philosophy. Ample focus was placed on the operating philosophy, with much effort spent on automation of the process to ensure safety, quality, recoveries and throughput were not compromised, yielding a sophisticated and modern design.

A general flowsheet of the 1988-modified Chambishi process, together with the new and/or expanded unit operations due to the COSAC project, is given in Figure 1. As can be seen, copper is recovered by electrowinning, with the resulting copper-free cobalt liquor being purified first in the cobalt purification stage (hydroxide precipitation), carbon

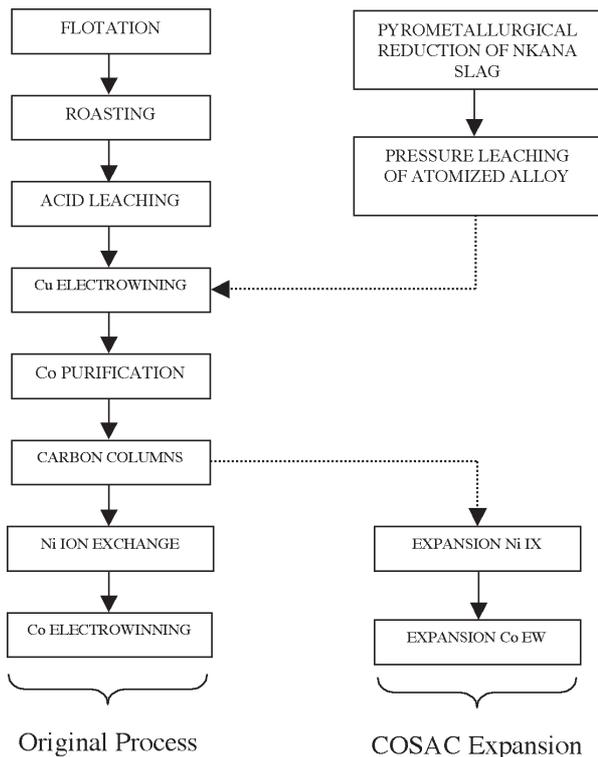# Development and implementation of a novel pressure leach process

Figure 1—Existing Chambishi flowsheet and COSAC expansion

columns and finally in the nickel ion exchange circuit before recovering the metal via cobalt electrowinning. The existing copper tankhouse has sufficient capacity to handle the additional copper from the COSAC process. However, both the nickel ion exchange and cobalt electrowinning circuits required expansion to be able to manage the additional loads of nickel and cobalt.

A previous publication[9] has described the installation of the ISEP® continuous ion exchange process for the control of nickel, whereas the present paper reports on the development and commissioning of the alloy leach process.
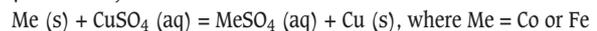
Process development

Initial development of the COSAC Leach Process focused on a two-stage leach, which aimed to selectively extract cobalt in the first stage, whilst recovering copper and the remaining cobalt values, and at the same time rejecting iron by precipitation, in a second stage. High recoveries and good product quality were achieved in the testwork programme, but the capital costs were high which prompted a re-consideration of the process.

A single stage leach for the extraction of both cobalt and copper, followed by a much smaller second-stage high-pressure leach for copper polishing was then evaluated. High cobalt recoveries were achieved in the first-stage leach, while copper recoveries were slightly above 80%. A techno-economic analysis showed that the operating costs were higher than the estimated revenues for recovering the additional 20% copper using the second-stage high-pressure leach. Hence, the process was ultimately simplified to a single-stage leach, which became the basis of design for the COSAC Leach Process.
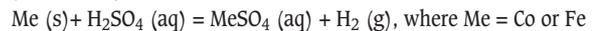
Process description

Figure 2 and Figure 3 are simplified process flow diagrams of the COSAC Leach Plant. Atomized alloy is received as an 8% slurry in the COSAC Leach Plant, where it is first dewatered to 70% solids before being discharged to a surge tank. The thickener underflow is then repulped to a controlled density and fed into the autoclaves. In the cascading series train of five autoclaves, the alloy is mixed with a sulphuric acid, copper sulphate lixiviant in the first vessel. Copper sulphate is added and undergoes cementation with cobalt and iron metal values in the alloy (Reaction 1). This helps minimize the risk of hydrogen gas formation in the autoclaves via direct acid dissolution (Reaction 2). The copper precipitate and copper metal value in the alloy require oxygen to be leached (Reaction 3), which is sparged into the bottom of each vessel. Ferrous iron is also oxidized and precipitated as a hematite/goethite residue (Reactions 4, 5 and 6). The leaching reactions in particular are highly exothermic, and heat is removed by controlled cooling to temperatures between 135°C and 150°C across all vessels. In order to facilitate the reactions, a significant oxygen overpressure is maintained by controlling the leach pressure between 8 and 10 bar(g).
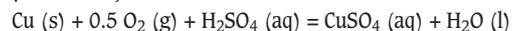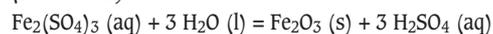
(Reaction 1)
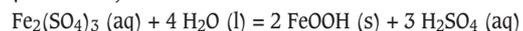$$Me\ (s) + CuSO_4\ (aq) = MeSO_4\ (aq) + Cu\ (s),\ where\ Me = Co\ or\ Fe$$

(Reaction 2)
$$Me\ (s) + H_2SO_4\ (aq) = MeSO_4\ (aq) + H_2\ (g),\ where\ Me = Co\ or\ Fe$$

(Reaction 3)
$$Cu\ (s) + 0.5\ O_2\ (g) + H_2SO_4\ (aq) = CuSO_4\ (aq) + H_2O\ (l)$$

(Reaction 4)
$$2FeSO_4\ (aq) + 0.5\ O_2\ (g) + H_2SO_4\ (aq) = Fe_2(SO_4)_3\ (aq) + H_2O\ (l)$$

(Reaction 5)
$$Fe_2(SO_4)_3\ (aq) + 3\ H_2O\ (l) = Fe_2O_3\ (s) + 3\ H_2SO_4\ (aq)$$

(Reaction 6)
$$Fe_2(SO_4)_3\ (aq) + 4\ H_2O\ (l) = 2\ FeOOH\ (s) + 3\ H_2SO_4\ (aq)$$

The copper sulphate concentrations in the plant are fairly saturated, hence, the requirement to prevent hydrogen gas formation is controlled by the lixiviant flow. In order to avoid excessive flows through the autoclaves, which would impact on their size, alloy is added in stages so that the copper sulphate precipitated in an upstream stage is re-leached and available in the downstream autoclaves. The COSAC Leach Process adds alloy in three co-current steps, cutting the lixiviant volume flow to one-third of the overall requirement, reducing the plant size significantly.

The slurry from the alloy leach vessels is letdown in the flash vessel and the liquor is collected in the discharge tank before being cooled for downstream processing. The leach vessel vent is let-down before joining the flash vessel vent in the discharge scrubber, where the gases are scrubbed free of acid mist and particulate. Scrubber effluent is cooled and reports to the belt filter for washing of the cloth and subsequently, washing of the residue.

The cooled, leached slurry is thickened, where primary separation of the cobalt and copper-rich liquor from residue occurs to produce a dense underflow. The underflow reports to the residue filter and the overflow is stored in a surge tank.

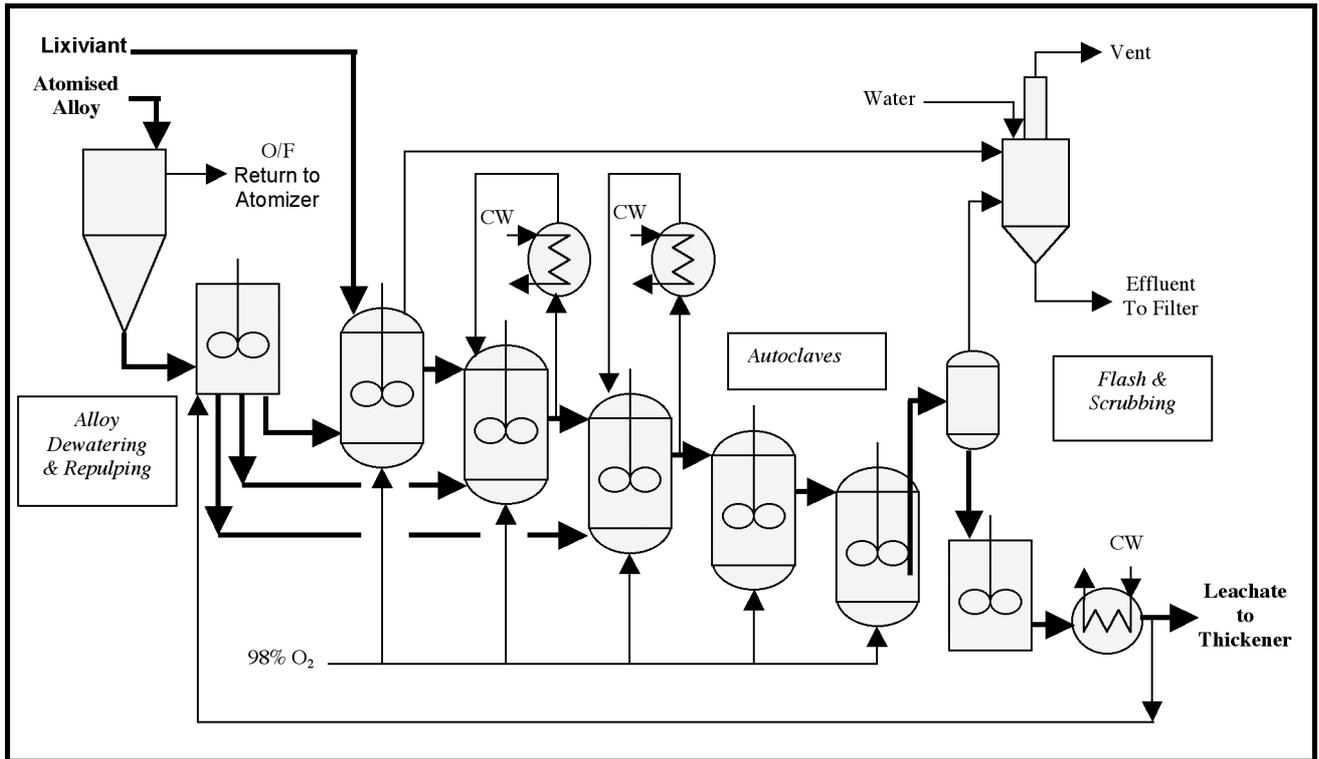# Development and implementation of a novel pressure leach process

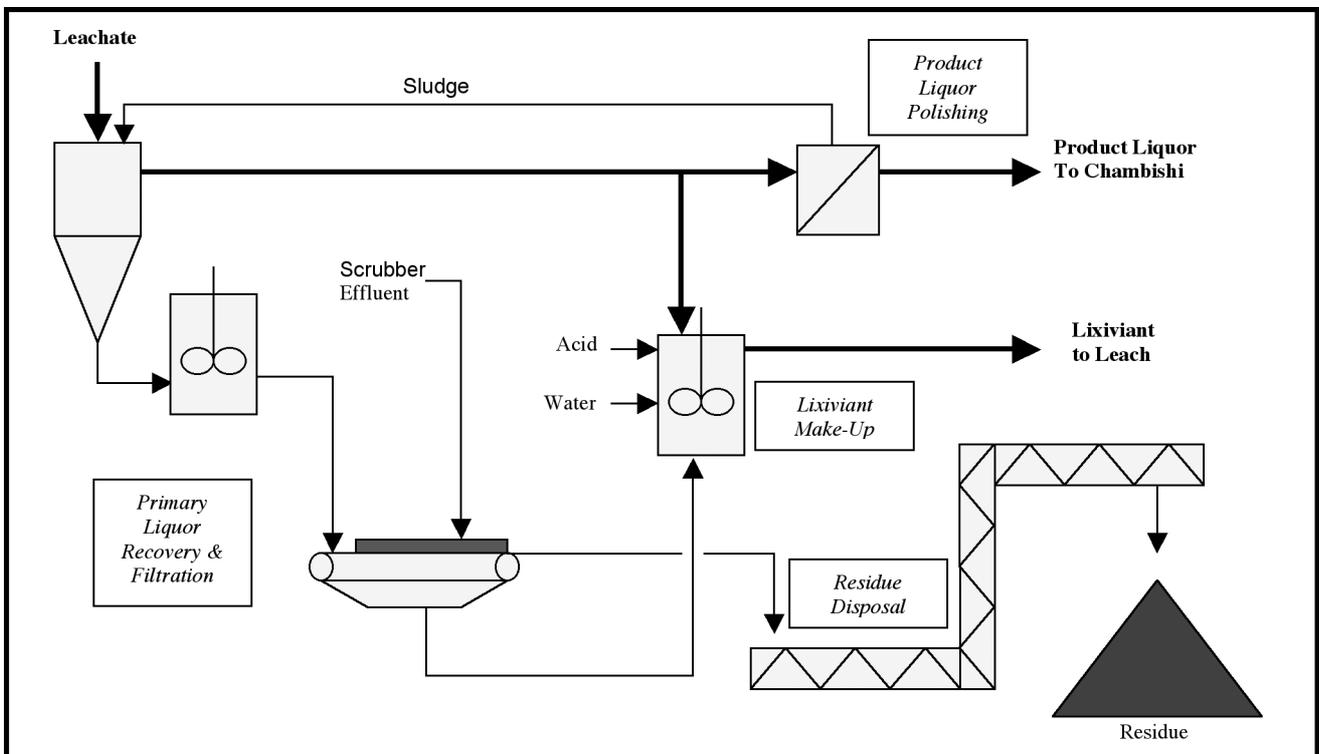**Figure 2—Alloy receiving and leaching process flow diagram**

Figure 3—Liquor recovery and lixiviant make-up process flow diagram

Once the thickened slurry is fed onto the belt filter, the cake produced is washed in three countercurrent stages. Filtrate and washate are recycled into the process. The washed cake is discharged into a series of spiral conveyors, which deliver material to a residue stockpile for transfer by front end loader onto trucks, which transport it to a disposal facility.

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A portion of the leachate thickener overflow is blended in the alloy lixiviant tank with sulphuric acid and belt filter solutions to produce a lixiviant with a controlled acidity, copper sulphate content and volumetric flowrate, prior to feeding into the first autoclave.

The remaining leachate thickener overflow is polish filtered and the product liquor pumped to the Chambishi Plant for recovery of copper and cobalt in the existing Roast-Leach-Electrowinning (RLE) Process. Nickel impurity in the alloy is fed into the RLE Plant and is removed by ion exchange. The COSAC Process doubled the nickel removal duty, which necessitated a replacement of the existing system with an ISEP® system[9]. The cobalt cathode production was tripled by the COSAC Process, which required an expansion to the existing cobalt electrowinning tankhouse.

Design challenges

Alloy handling

Alloy dewatering

Atomized alloy is intermittently received in the COSAC Leach Plant as a dilute slurry, and hence dewatering is necessary to avoid excessive dilution in the leach process. The alloy has a very high specific gravity, in the order of 7.0–7.5 kg/*l*. Settling rates were accordingly found to be high, necessitating minimum slurry line velocities of 2.0–2.5 m/s in 25NB pipes to avoid sanding-out. However, there is an extremely fine portion to the alloy (Figure 4), which has a much lower settling rate. An ultra high-rate thickener was therefore selected for dewatering, which could claim most of the easy-to-settle material in a short residence time, and provide a means to capture fines via a well-flocculated bed.

Upon start-up of the atomizer, the alloy particle size distribution shifted significantly to a coarser range (Figure 4) over that for which the system had originally been designed. Hence, this caused a problem, since, accentuated by their specific gravity, the coarse particles settled rapidly before a bed could be established, resulting in frequent choking of the underflow system. Clearing the chokes is a high maintenance task, and hence, a more practical solution had to be found.
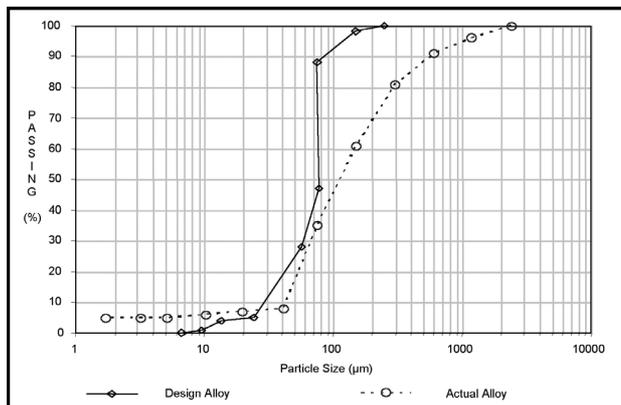

Figure 4—Atomized alloy particle size distribution

Chambishi then began testing the recovery of coarse material using magnetic separation, which has proven to be very successful, with recoveries of 95% of total material being possible at high solid concentrations (>70% w/w). Employing the magnetic separator for primary dewatering significantly relieved the thickener duty and lowered the risk of underflow blockages. The remaining 5% of atomized alloy feed, consisting of non-magnetic slag and matte particles, is recovered in the thickener using a non-ionic flocculant.

Autoclave feed system

Dewatered alloy is fed into a surge tank and repulped to a controlled density. Initially, three independent feed systems pumped alloy slurry at a controlled rate into the first three autoclaves (Figure 2). Abel dual-diaphragm pumps were selected for this application for the following reasons.

➤ Check valves, which form a part of the diaphragm pump design, also act as a safety measure to prevent autoclave depressurization during abnormal conditions.
➤ The hydraulic drive has a built-in pressure relief valve to prevent over-pressurizing the pump chambers and discharge piping should a blockage occur.
➤ Ability to pump slurries with high solids content, in order to avoid excessive dilution downstream.
➤ Ability to develop a sufficient pressure in a single stage, simplifying the system.
➤ The volumetric flow of slurry can be accurately controlled via a variable speed setting.
➤ Easy accessibility of check valve internals, which is the major wear point in the system.

However, insufficient testwork on a relatively new application, a design specification on particle size which proved to be too narrow in practice, and the coarser than expected material produced by the atomizer rendered the system extremely difficult to operate. It was found that coarse particles slowly sanded-out in the Abel pump check valves, preventing the internal ball from seating properly, which in turn prevented the pump from displacing material.

Several modifications were made to the Abel pumps, including:

➤ Reducing the internal diameter of the check valves to increase linear velocities and prevent coarse material from sanding out
➤ Reversing the flow of slurry through the pump to eliminate accumulation within the check valves
➤ Increasing the speed of the pump to increase linear velocities and prevent coarse material from sanding out.

All modifications were tested independently from one another and also combined to ensure that every possibility was tested. However, none could overcome the minimum linear velocity required to prevent alloy from sanding-out in the check valves. The reverse-flow configuration resulted in material blocking in the bottom of the discharge check valve, causing the material to accumulate and eventually prevented the ball from seating properly. Consequently, it was decided to review other types of pumps and system configurations for the application. Three options had merit; a pneumatic transfer system, a centrifugal pump ringloop system and a

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peristaltic pump feed system.

The pneumatic transfer system concept utilized a low-head centrifugal pump to fill a feed pot, which is pressurized with compressed air. The slurry is then forced into the autoclave by displacement with air, as the system depressurizes to autoclave pressure. The pot is then depressurized to atmospheric pressure and the cycle repeated, with the cycle frequency and pot slurry volume determining the slurry feed rate. However, this system was not selected because of the risk associated with intermittent slurry flow and the need for a high-pressure air system.

The centrifugal pump ringloop system utilizes two high-capacity, high-pressure centrifugal slurry pumps, arranged in series, to produce the required head for autoclave feeding. The slurry is recirculated and minimum velocities are maintained throughout the system, with a backpressure control loop modulating the pressure at slightly above autoclave conditions. The autoclaves are fed from the ringloop and flow is measured and controlled on each take-off.

However, due to the success of the peristaltic pump feed system proposed by Chambishi, which was the first alternative tested, the centrifugal pump ringloop system was never installed. The high-pressure peristaltic pump feed system required moderate slurry densities and high line velocities to work. It was decided that the autoclave discharge solution be used, as it contained clay-like goethite solids, which would provide lubrication between the ragged alloy particles, minimizing the risk of particle locking and subsequent system blockage, as well as maintaining solution metal concentrations high. Each of the first three autoclaves are fed alloy using independent peristaltic pumps with in-line automatic isolation controls installed to satisfy safety requirements and automatic water flush systems installed to provide purging during start-up and shut-down of the system to avoid blockages from settlement. The system has been operated successfully with little maintenance requirements since the installation.

Autoclave design

The autoclave design was carefully considered to ensure process requirements and system operability were addressed. A cascading train of vertical pot-like autoclaves was reviewed against a conventional multi-compartment horizontal configuration. The following issues were investigated to determine the benefits and disadvantages of each configuration.

Mixing efficiency

As the recovery of cobalt and copper involves the interaction of solid, liquid and oxygen phases, it is necessary to achieve a good mixing efficiency in the autoclave. The vertical autoclave configuration offers a geometry that favours mixing conditions. The cross-section diameter within the tan-to-tan height is uniform, which gives excellent flow characteristics within the autoclave. Ultimately, the recovery of metal values over a specific residence time is expected to be higher in a vertical autoclave configuration than with a horizontal configuration.

Oxygen utilization

Higher mixing efficiencies in the vertical configuration also results in a higher oxygen utilization. Mass transfer of dissolved oxygen is improved and results in a higher driving force for oxygen transfer between the vapour phase and aqueous phase.

Short-circuiting

Horizontal autoclave configurations normally experience more short-circuiting of solid particulate than in vertical configurations. This is because the cross-sectional area at the overflow point between compartments is restricted where the walls converge together as the top of the autoclave is approached. Increased rise velocities are experienced in the vicinity of the overflow point, resulting in particulate entrainment (Figure 5). This is further magnified by the fact that feed is normally introduced on the slurry surface, which is the level at which overflow is occurring. As the COSAC process relies on multi-stage alloy addition to reduce the copper sulphate requirement, it is important that short-circuiting be minimized.

The vertical configuration overcomes these deficiencies. The cross-sectional area at the overflow point is equal to the cross-sectional area at any point of the tan-to-tan height. Hence, rise velocities are fairly low and result in less particulate entrainment. Also, the overflow from an upstream autoclave can be easily directed into the bottom of a downstream vessel or, if the slurry discharge point is located at the bottom of the autoclaves, it can be directed to the top of the vessel that follows. In the case of the COSAC design, the feed enters the top and exits the bottom of the autoclaves. This ensures that all particulate that enters an autoclave must go through the agitation zone before it can be discharged, which decreases the probability of short-circuiting occurring. Hence, alloy can be added in several autoclaves without the recovery of metals being affected.

Volume utilization

The volume utilization in horizontal autoclaves is normally lower than for vertical configurations in order to avoid excessive short-circuiting (Figure 5). Should the volume utilization be as high as in a vertical configuration, additional compartments are required to ensure recoveries are not compromised due to short-circuiting. As a result, vertical autoclaves offer better volume utilization; hence, the systems can be designed with an overall smaller autoclave volume than for the horizontal vessels. A financial analysis revealed that the costs of a horizontal and vertical system configuration were very close, largely due to the improved volume utilization of the vertical configuration, despite there being five vessels versus one.
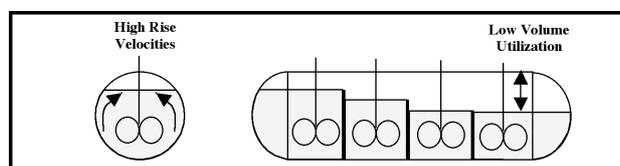

Figure 5—Horizontal autoclave configuration

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Availability

In general, maintenance of autoclave systems is high and dictates the overall plant availability, so that it is important to implement strategies to improve the availability. Should horizontal autoclaves be utilized, the availability is often improved by installing a stand-by, which is a costly investment. The much smaller vertical autoclaves, each representing a compartment in a horizontal autoclave, can be by-passed, so that maintenance can be done on a single vessel while the others remain in operation. Throughput and recovery are not significantly compromised, especially if an additional vessel is installed at the end of the train. The cost of doing this is appreciably less than the cost of installing a stand-by horizontal autoclave.

Selection and performance

Primarily due to the similar costs of operating vertical and horizontal autoclave systems, and the fact that the vertical autoclaves can be by-passed to avoid excessive stand-by system costs, the vertical autoclave configuration was selected.

In November 2001, the first autoclave in the train was taken offline for maintenance. The feed systems were re-routed to the second autoclave in the train and the system was promptly brought back into operation, without impacting on throughput or metal recoveries. This proved the system to be versatile, providing the expected advantages over the horizontal configuration.

Autoclave cooling design

The alloy leaching and iron precipitation reactions produce a net heat generation of 1630 kcal/kg of alloy feed solids. Three-stage addition of alloy is employed in the leach process to reduce the net copper sulphate recirculation rate, as explained above. In the first autoclave, the alloy addition rate is sufficient to allow the unit to operate autogenously at 135°C, or up to 150°C with supplementary steam addition, while maintaining the required stoichiometric ratio of alloy to copper sulphate in the feed streams in order to prevent the formation of hydrogen gas. In the second and third autoclaves, the net heat generation is such that heat must be removed to control the temperature to a setpoint between 135°C and 150°C. The remaining autoclaves only cater for minor copper sulphide reactions, which do not generate much heat.

The following heat removal options were considered in the design of autoclaves 2 and 3:

➤ Direct quenching using water or excess lixiviant
➤ Internal cooling coils
➤ Flash recycle cooling
➤ Reflux condensers
➤ External recirculation cooling.

Direct quenching

The established practice of direct quenching with cooling water to control temperature was not practical, as the dilution of the reactor liquor would significantly upset the water balance of the plant. Although quenching with lixiviant did not create a dilution problem, the volumetric throughputs in the autoclave train would have been significantly increased, resulting in increased reactor sizes to maintain residence times. Hence, these options were discarded.

Internal cooling coils

The concept of heat removal using cooling coils, which are suspended from the baffles within the autoclaves, was reviewed. The concept provided the most direct method of cooling, requiring minimal additional equipment. However, it was found that the large amount of tubing needed would require a great degree of anchorage, which would penetrate into the refractory lining, increasing the amount of crevices, which are susceptible to corrosion.

The coils were also potentially subject to scaling and corrosion, which would require regular maintenance. As the coils could only be accessed when the plant was off-line, the overall plant availability would have been impacted, and hence, external cooling methods were considered.
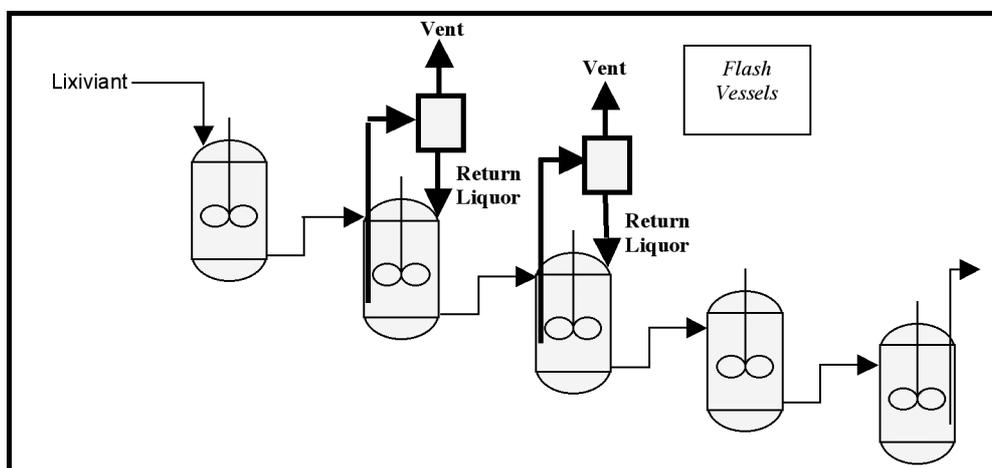

Figure 6—Flash recycle cooling flow diagram

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Flash recycle cooling

Flash recycle cooling (Figure 6) removes heat from the autoclave by continuously extracting a portion of the autoclave contents, cooling the stream by flashing and pumping back into the autoclave. The return liquor quenches the heat liberated by the reactions and by adjusting the recirculating flow being flashed, the temperature of the autoclave can be controlled.

The main advantage of this option is that operating costs are minimal, considering no cooling medium is required. However, on the downside, a heat balance revealed that a considerable amount of liquor needed to be flashed off in order to control the circuit at a temperature of 135°C, resulting in a discharge slurry that contained over 35% w/w solids, which would reduce the life cycle of ancillary equipment, such as the discharge choke valve. It was also found that the autoclave discharge solution was supersaturated with copper and cobalt sulphate, which inhibited metal leaching and reduced recoveries, resulting in rejection of this option.

Reflux condensers

The concept of reflux condensers (Figure 7) allows water from within the autoclave to evaporate, hence cooling the process. The evaporated gases pass through shell-and-tube reflux condensers, and the water vapour condenses and flows downward back into the autoclave. The vent rate determines the amount of water evaporated from the autoclaves, which is used to control the autoclave temperature. The non-condensable gas component, primarily oxygen and nitrogen, is recycled into the autoclaves for recovery of vented oxygen, which is considerable due to the high vent rates required to maintain process temperatures. A bleed is necessary to control the nitrogen content in the recirculating gases and oxygen make-up is required to maintain the autoclave circuit pressure.

The primary advantage of using vent gas cooling is that the equipment is not in direct contact with the process slurry. The less aggressive environment permits the use of more economical materials, and the risk of fouling is minimal since the film condensation in the reflux condensers tends to return carry-over material back into the reactor.

Additionally, the high vent recirculation rate will allow oxygen utilization to approach the theoretical limit of 80%, determined by the venting rate required to maintain the oxygen to nitrogen ratio in the vapour space of the reactors, in comparison to a utilization of less than 70% expected in a once-through system.

However, due to the high heat removal duty required in this specific application, the vent rate was found to be extremely high and the ratio of recirculating off-gas to vent gas was calculated to be 146 to 1. The extremely high agitator power required to disperse the high flow of recirculating gas is impractical to install. Also, combustibles, such as hydrogen (Reaction 2), could concentrate to above the lower explosive limit, even if the process generated very small amounts. For these reasons, this option was rejected.

External recirculation cooling

In this approach, slurry is drawn out of the autoclaves by a high capacity, centrifugal pump and passed through a shell-and-tube heat exchanger (Figure 8). The recirculating liquor temperature is dropped marginally to minimize thermal shock of the autoclave refractories, before being returned to the autoclave. The cooling water flowrate to the heat exchanger is varied to control the process temperature. This indirect method of cooling does not impact on the solution concentrations (dilution or saturation), autoclave retention time or autoclave gassing rate.

Since the heat transfer surfaces are in direct contact with the liquor, the potential for fouling exists. However, the entire cooling system is isolatable and tube cleaning is
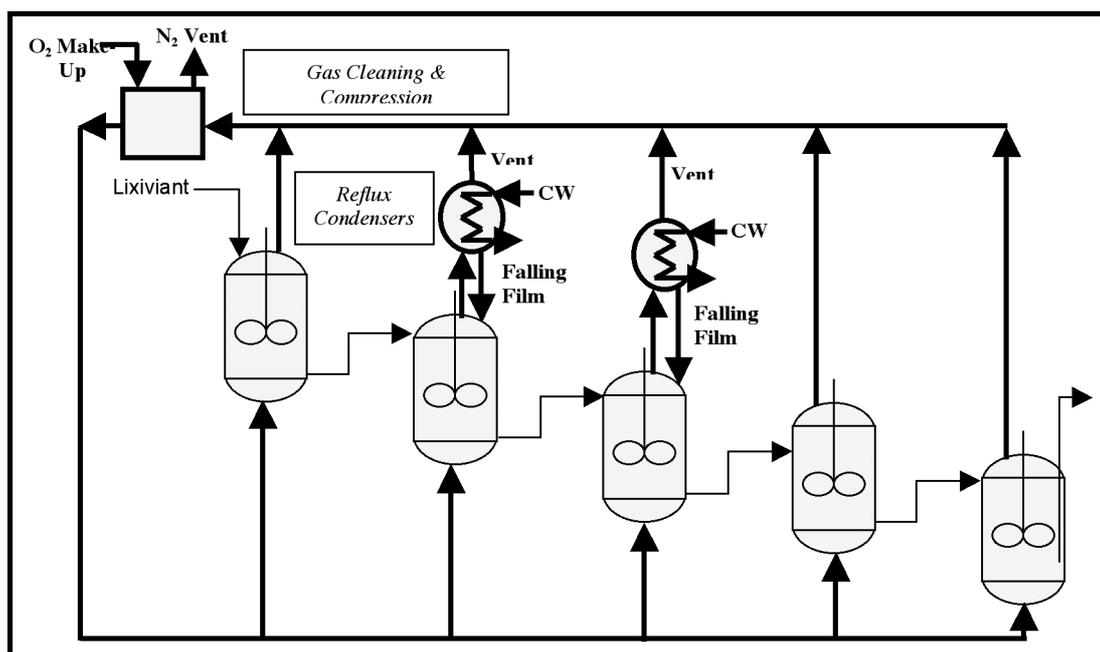

Figure 7—Reflux condenser flow diagram

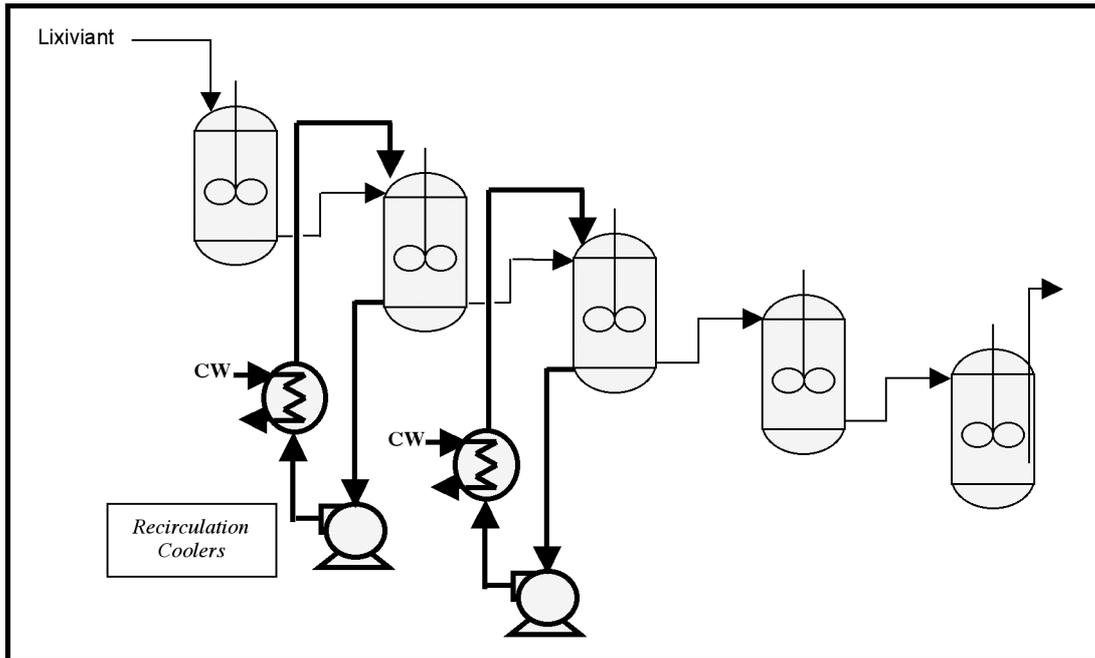# Development and implementation of a novel pressure leach process

Figure 8—External recirculation cooling flow diagram

readily accomplished without interruption of the process. Each reactor is equipped with two pumps and two slurry coolers, one operating while the other is being serviced, to permit continuous operation of the leach circuit.

Selection and performance

The external recirculation cooling option was selected. The coolers were successfully hot commissioned in early 2001, and have been operated when alloy throughput required the use of autoclaves two and three. The system was easily and safely isolatable when a pump seal required replacement in mid-2001.

Residue liquor recovery

Autoclave discharge slurry contains high concentrations of cobalt and copper. Hence, care must be taken when removing solid residue from the process, in order to ensure the best separation of solid from liquor is achieved. During the design, several liquor recovery options were investigated. These included:

➤ Plate and frame filter press
➤ LAROX filter
➤ Counter-Current Decantation (CCD) circuit
➤ Vacuum belt filter.

Plate and frame filter press

Plate and frame filters are normally advantageous over other filtration methods because of their relatively high solids loading rates. Cake moisture contents are also normally very low, particularly with a membrane squeeze step in the filtration sequence. However, normally 2 to 3 times the wash water used in vacuum filters is required to achieve acceptable wash efficiencies; nonetheless efficiencies as high as those achieved in vacuum filters are unlikely.

Major disadvantages of the filter press are the high

maintenance usually associated with these types of filters, especially when used on chemical precipitates, and the associated high labour requirements. The number of moving parts and intermittent filtration steps requires careful observation to maintain the equipment integrity.

Larox filter

Larox filters typically have high solids loading rates and low cake moisture contents, particularly when the cake squeezing steps are incorporated into the filtration sequence. Wash efficiencies are comparable, or slightly better, to those achieved in filter presses. Although the Larox filter is a fully automated system, a weekly preventative maintenance programme is normally required to minimize breakdowns, and maintenance costs are generally very high when used with chemical precipitates. Also, these units are often very expensive in comparison to other filtration options, creating a higher expectation on efficiency and operability.

CCD circuits

CCD circuits normally are used when filtration methods prove to be unfeasible, such as when solids loading rates are found to be unacceptable. The CCD thickener underflows have very high residue moisture contents, increasing the demand on wash liquor. Wash efficiencies are none the less poor when compared to filtration methods, as washing is actively achieved by diluting overflow with wash liquor.

CCD circuits are normally made up of several large diameter thickeners and consequently result in large footprint requirements. The costs of CCD circuits are generally high due to the size of thickening equipment.

Vacuum belt filter

Vacuum belt filters typically have moderate solids loading rates, when compared to other filtration methods, and cake

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Table I

Liquor recovery option results

Option	Cake moisture (% w/w)	Wash ratio (m³/t dry sol.)	Liquor recovery (%)
Filter press	30	>2.00	95.0*
Belt filter	30-45	1.00	98.0*
LAROX	27	1.50	95.0*
CCD	58	1.63	84.0**

*Based on 70% w/w solid feed cake.

**Based on overall circuit liquor recovery.

Table II

Water requirements at full plant capacity

Consumer	Reason	Required (m³/hr)
Autoclave off-gas scrubber spray water	Removal of solid particulate and acid entrainment in gas	13
Belt filter cloth wash water	Removal of fine particulate in cloth to prevent blinding	20
Belt filter cake wash water	Recovery of entrained liquor contents in cake	8
Lixiviant tank dilution	Prevent metal solutions from saturating in process	9

moisture contents are normally higher than pressure filtration methods. However, wash efficiency and wash water consumption are very good, since countercurrent washing on several sections of the filter is possible. Collection and re-use of water in a staged, countercurrent fashion to cake movement makes for excellent liquor recoveries at very low wash water rates. This is important for the process, which is very sensitive to water dilution as the liquors produced are at concentrated levels of dissolved metals.

Belt filters are normally the most forgiving of all the units, being able to accommodate quite significant variations in feed quality, which can be expected from the COSAC process. Belt filters usually have a reasonable turn down and turn up ratio, which gives further flexibility.

Testwork results

Testwork and/or simulations were completed for each of the options, to primarily determine the liquor recovery. The results are shown in Table I.

Selection and performance

The criteria for selecting an option to recover liquor from residue were as follows:

➤ Highest liquor recovery possible
➤ Maximum wash ratio of 1.50 m³/t dry solids to avoid over-dilution of the process
➤ Flexible operating system due to variable feed properties
➤ Cost.

It was found that the belt filter could produce the highest liquor recovery at the lowest wash ratio. To put the importance of liquor recovery into perspective, a 1% reduction in liquor recovery equates to more than US $150,000 per year loss in revenue.

As mentioned above, the belt filter is relatively flexible to changes in feed conditions. Finally, the belt filter was found to be one of the least expensive options, making it an obvious choice for liquor recovery in the COSAC Leach Plant.
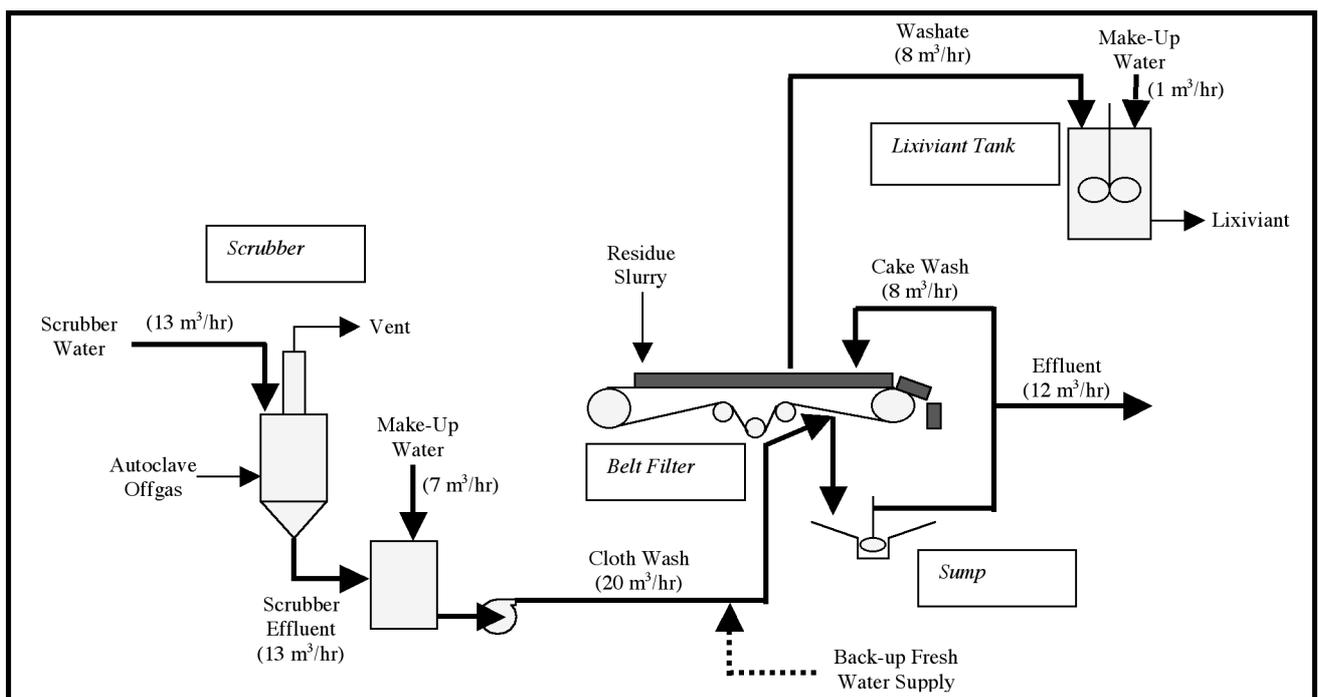

Figure 9—Circuit water balance at full plant capacity

Development and implementation of a novel pressure leach process

The belt filter was commissioned with minor problems, which were easily optimized on site. However, following several weeks of operation, a prolonged dip in autoclave temperature produced iron-oxyhydroxides, which quickly blinded the filter cloth. Chambishi installed a 20 bar(g) wash water pump to improve the performance of the cloth wash stage, which proved to be a robust solution.

Water balance design

In order to maximize solution metal concentrations in the process and minimize effluent volumes from the plant, the water balance was assessed. There are four major water requirements in the plant, which are shown in Table II, the most significant water consumer being the belt filter.

The feasibility of re-using the effluents or washates produced was investigated. Following a determination of the water quality following each step, it was found possible to arrange the water users as shown in Figure 9.

Effectively, the water balance begins at the scrubber with the fresh water input being used four times over. Make-up or bleed points are installed along the balance to satisfy individual water requirements, which is done automatically via control systems. An automatic back-up water supply is installed on the belt filter cloth wash system, in case the scrubber effluent flow is insufficient or quality drops below an acceptable level due to a temporary slurry carry-over from the autoclave system.

Conclusions

An innovative process has been developed and installed at the Chambishi Cobalt Plant in Zambia. The COSAC leach process has successfully overcome a number of challenging process and design problems arising from a very heavy metallic alloy with a high heat release and high potential for hydrogen evolution. Successful application of process know-how in deriving the operating philosophy of the process has resulted in an efficient, streamlined, sophisticated plant. The leach plant was commissioned early in 2001, and has since overcome a number of issues associated with a much coarser than planned alloy feed product.

Acknowledgements

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