

COBALT FROM SLAG- LESSONS IN TRANSITION FROM LABORATORY TO INDUSTRY

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ABSTRACT:

Anglovaal Minerals acquired the Nkana slag dump from Zambia Consolidated Copper mines (ZCCM) in 1998 and embarked on an ambitious project to pioneer the recovery of cobalt from slag using DC smelting technology. Anglovaal Research Laboratories performed initial pioneering smelting tests in a 250 kW DC furnace over a 3-year period before engaging Mintek for three piloting campaigns at the 1.5 MW level. Mintek had previously developed a reductive smelting process, patented in 1995, using a DC arc furnace for the recovery of cobalt from slags. The results of the pilot-plant test campaigns were used as the basis for engineering the commercial 40 MW operation installed at Chambishi Metals plc in Zambia in 2001.

This paper traces the development of understanding of the metallurgical fundamentals ultimately used to predict furnace performance, reviews some of the extreme technical challenges faced, how solutions were identified and highlights some of the critical issues identified during piloting and how these impacted both positively and negatively on the final commercial operation.

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INTRODUCTION:

The Nkana slag dump, which had been in existence since the early 1930s, was purchased from Zambia Consolidated Copper Mines (ZCCM) in 1998 by Anglovaal Minerals (AVMIN). The slag dump contained about 19 million tons of slag grading approximately 0.75% Co and 1.2% Cu in a fayalitic reverberatory slag. The potential for the exploitation of this dump as a source of cobalt had been realised some years before as evidenced by Whyte's [1] description of some early testwork. In addition to the dump, AVMIN also purchased the Chambishi Roaster, Acid Plant and Copper/ Cobalt Refinery.

AVMIN's intentions were to install a crushing and screening plant for the slag; a suitably sized DC furnace, an atomising facility; and to upgrade the existing cobalt refinery situated at Chambishi, about 30km from Nkana) and produce a variety of cobalt and copper products. This project, named COSAC (Cobalt from slag and copper by product) was to be fast-tracked to take advantage of the high cobalt price current at the time. Small-scale (120 -250kW level) path-finding DC smelting trials had been successfully performed during 1996-1998, at AVMIN Research Laboratories (AVRL), creating a high level of confidence in the technical feasibility of the pyrometallurgical treatment route envisaged for the slag.

The project was designed to incorporate the following aspects:

- To continue to operate the existing Chambishi Cobalt and Sulphuric Acid Plant while improving cobalt plant recoveries and production and upgrading and debottlenecking various portions of the plant.
- To install material handling- crushing & screening facilities at the Nkana dump.
- To implement cobalt production from slag by means of DC furnace smelting of 700 000 tons per annum of dried slag, followed by atomization of the alloy produced.
- Installation of a leach plant to solubilise the atomized alloy, and integrate the leach plant into the existing refinery.
- To dispose of the barren slag from the furnace on a new slag dump at Chambishi.
- To develop a variety of added-value cobalt products, including a superalloys vacuum melting plant in South Africa.

PROCESS DESIGN PHILOSOPHY AND KEY DESIGN CHALLENGES

The Crushing and Screening Plant was designed for a 12 hours per day operation at the dump, producing a minus 12mm crushed slag. The dump had been extensively sampled and "mining" of the dump followed a rigorous plan to ensure consistent feed grade to the metallurgical plant. This was necessary because of the extremely wide variation in quality of the dump slag, varying from a small tonnage of high grade (>6% Co) converter slag dating back to the early years of operation (predating cobalt recovery circuits in the beneficiation plants) to the more recently produced low grade 0.3% Co reverb slags. Crushed slag was transported by road to the Chambishi site where it was dried prior to smelting.

Full details of the complete facility have been described previously by Munnik et al, [2]. The intention in the present paper is to track the evolution of the project from an initially somewhat vague concept, through metallurgical trials into eventual commercial operation. A simplified flow diagram is included in Figure 1 for clarity.

As recovery of cobalt by pyrometallurgical treatment had not been practiced previously, the project was pioneering in many respects. A significant issue was that the plant was to be installed in Zambia at a time when the country's infrastructure was in poor repair, and many "First world" facilities and services were simply not available. Most of the engineering and fabrication was performed either in South Africa or in Europe, and shipped to site for erection. Transport and construction challenges relating to the rainy season also influenced procurement decisions. In certain cases, failure to deliver key components on time could result in a six-month delay. A specific example of this was that the furnace piling and foundations were installed well before the final furnace layout had been confirmed, and the eventual furnace design had to fit within dimensions dictated by the concrete already in place.

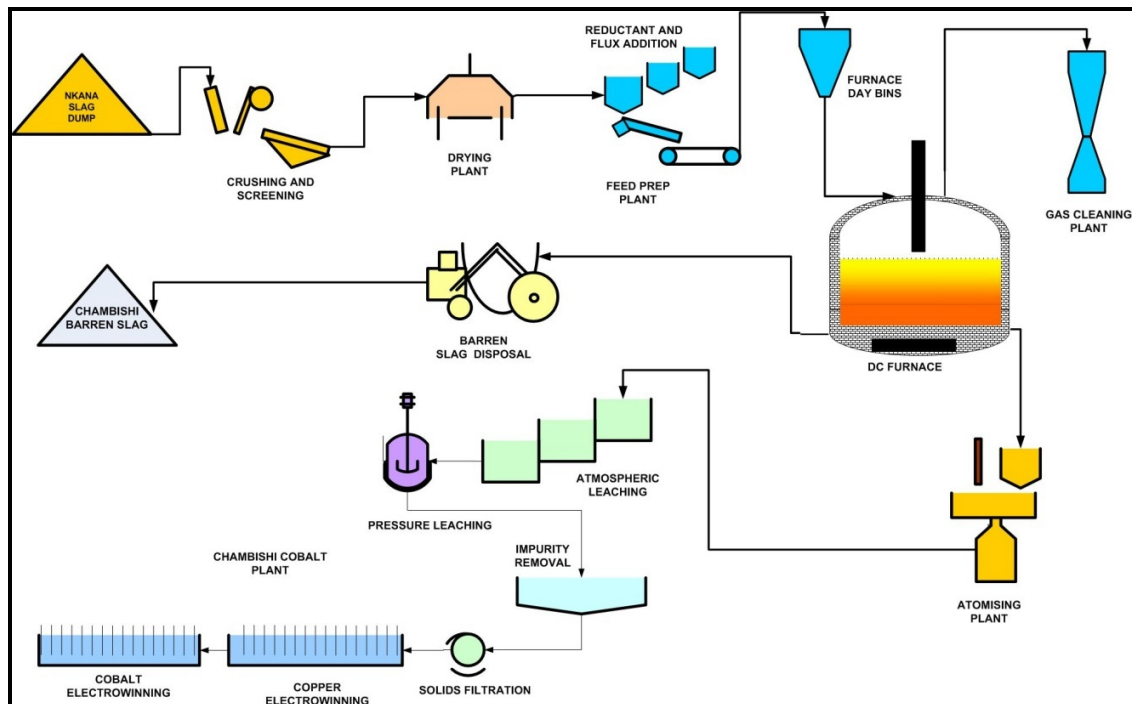


Figure 1- Simplified Flowsheet for the Cosac Project.

As already mentioned, the project was pioneering in many aspects:

- The very high degree of slag superheat ($>400^{\circ}\text{C}$) necessary to ensure tapping of the molten Fe-Co-Cu-S alloy placed very high demands on the furnace cooling system.
- Side feeding of a DC furnace (as opposed to hollow electrode feeding) was novel.
- The atomisation process selected for the molten alloy, necessitated by its toughness, which made it almost un-millable, also provided its own set of challenges.
- The leach circuit design was novel and untested prior to this project.
- The operational staff had no prior experience of operating a plant of this type.
- The infrastructure in place in Zambia at the time raised concerns about security of electrical supply.

Given the above challenges and degree of novelty, a Type 1 startup (Based on McNulty's [3] criteria) (see Figure 2 below) was unlikely, as was a Type 2 or 3 startup. In spite of this the project timeline was based on type 1 expectations. McNulty had identified four types of project start-up curves summarized as follows:

- Type 1: Mature technology, used elsewhere
- Type 2: Prototype technology, incomplete pilot testing
- Type 3: As Type 2 but with limited piloting and/or feed variability
- Type 4: As Type 2 but more complex flowsheet

Furnace Design Availability

An example of how these various aspects influenced design is shown in the very important issue of furnace availability: It was accepted that a properly designed new furnace, in isolation from peripheral equipment, should have a mechanical (equipment) availability of better than 98% and that utilisation of this equipment should be at least 98%. A furnace has, however, a finite lining campaign life. In the absence of any previous campaign history the design assumed a campaign life for the smelting furnace of 5 years, at the end of which period a complete rebuild would take place. The Zambian location suggested that until operating information became available it would be conservatively assumed that a rebuild could take 6 months, including flushing, accretion removal, hearth demolition, mechanical repairs, relining, drying, cold commissioning and ramp-up to full production. It was believed that with experience a 3-month reline duration would be achievable. 3 months every 5 years equated to a 5% decrease of availability.

It was recognised that problems that inevitably occur on peripheral equipment- feed system, granulation, atomising, taphole repairs, launder maintenance, etc. could reduce furnace utilisation by up to 5%. Electrical supply problems external to the furnace could decrease utilisation by 1-2%, and limitations imposed by maximum demand criteria could also lower utilisation. Also, it was generally accepted by furnace operators that a one-hour production stoppage equated to a 2-hour production loss, as a result of energy losses. While none of these issues related to the mechanical availability of the furnace, or minimised the requirement for the vendor to ensure 98% mechanical availability of the specific equipment, they did, however, have tremendous implications on the annual throughput, and hence financial performance of the operation, and it was considered prudent to plan for a somewhat lower net combined availability/utilisation at full power, especially since this was the first commercial use of a DC furnace in the non-ferrous industry, and there were only the Middleburg Ferrochrome DC and Namakwa Sands ilmenite smelting operations to use as partial benchmarks.

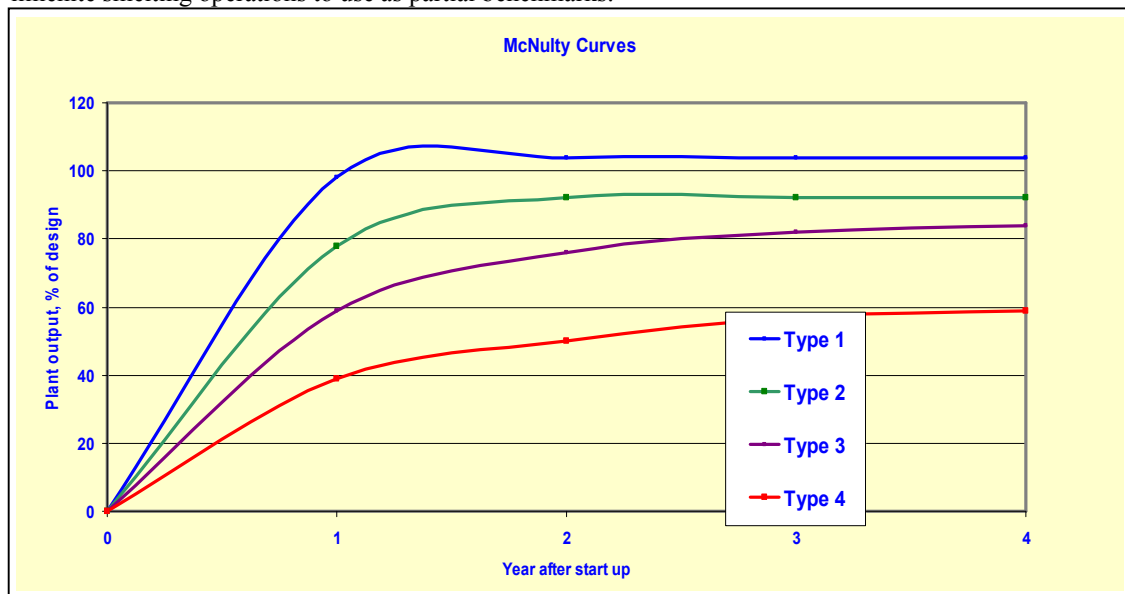


Figure 2: McNulty Startup curves

Based on the design long-term annualised throughput of 700 000 tpa slag treated for the life of the project, there was a requirement for the installation of substantially larger instantaneous maximum capacity at full power, which was calculated at 92 tons per hour of slag.

Engineering Design Issues

During the design phase of the project a number of alternatives had to be considered. These included:

- Selection of DC over AC power for smelting
- Selection of hot slag tapping into ladles and hot tipping rather than granulation.
- Selection of wet scrubbing over combustion followed by cooling and filtration.
- Atomisation of the alloy over sulphidisation, granulation and milling.
- Side feed to the furnace rather than via hollow electrode
- Installation of a 10MW pilot furnace prior to full-scale commercial operation

Many of the decisions made at the time were controversial or pioneering, and in some cases the decisions were taken to select a particular route simply to ensure progress of the project, as the technical and financial issues were unable to clearly produce a preference. In other cases the decisions taken were made following very extensive discussions with numerous advisors before finalising the eventual choice. To ensure clear communications, a “decision file” recording all decisions and their motivation was maintained for the duration of the design phase.

Method of Slag Disposal. The choice of slag disposal method (wet granulation or hot tipping from a pot) was debated at length. Ultimately the decision was made to tap hot slag into ladles and hot tip on a dump. The decision was made on a combination of economics and productivity- the removal of slag from the furnace being recognised as one of the potentially production limiting issues.

Feed Configuration.

An important decision revolved around whether hollow graphite electrodes would be used or whether feed should be through the roof. An important factor was that the hollow electrode feed configuration required disconnection of the feed system every time a new electrode section had to be installed, and this carried a high capacity penalty for apparently little metallurgical gain.

Furnace not designed to accept sulphide concentrate feed.

This decision was unanimous amongst the design team yet was ultimately disregarded by operations! Numerous reasons for the recommendation are listed below, not in any order of preference:

- The focus of the project was to efficiently recover cobalt from slag, and to attempt to design a furnace system to handle unknown quantities of unknown concentrates was considered detrimental to project timing. Sufficient quantities of concentrate were not available to perform extensive testwork. The results of small-scale tests did not show any benefits from concentrate addition.
- The installation of the atomising operation removed the need for sulphidisation of the alloy to make milling possible.
- The additional sulphur contained in concentrate would result in some SO₂ evolution as a result of labile sulphur in the mineral, and would also increase the sulphur grade of the alloy tapped, without any additional cobalt recovery. This sulphur in the alloy would form elemental sulphur during the leach and would report with the iron residue. The hydrometallurgical circuit was significantly more complex if a high sulphur matte was to be leached than a low-sulphur alloy, and leaching testwork indicated that the elemental sulphur precipitated during leaching, coated the metal, and inhibited further reaction.
- In a “slag only” operation, the CO-rich off-gas from the furnace was combustible, and could be used in the plant. If the gas contained sulphur a more expensive gas cleaning plant would be required.

Choice of Furnace

The selection of a DC furnace was initially controversial, yet the project team believed the reasons for selecting DC over AC were compelling:

- Given the known variability of slag feed composition, DC was able to handle wide variations
- The power input on DC is not constrained by slag conductivity.
- The DC configuration could handle fines without operational issues resulting
- DC was able to utilise high-volatile coal as reductant if required.
- The DC furnace had lower electrode consumption
- DC provided better control on iron reduction, and decoupled any requirement for the slag to have specific electrical properties.

The advantages of DC operation, controversial in the 1990s, are now well established, as Jones has recently pointed out [4].

THEORETICAL UNDERSTANDING AND MODELLING

When carbon is added to the slag, the various metallic elements reduce to different extents, at a given level of carbon addition, governed by their relative thermodynamic stabilities. This behaviour allows a reasonable degree of separation to take place during smelting. The intention of the process is to separate the valuable nonferrous metals from the iron and gangue constituents present in the slag. The desirable area of operation is clearly somewhere in the region where the recovery of cobalt is high, and the recovery of iron to the alloy is still reasonably low. This process needs to operate at a temperature above the liquidus temperature of the alloy containing the Co, Cu, and Fe, requiring an operating temperature somewhere between 1500 and 1600°C. This is substantially above the liquidus temperature of the principally fayalite slag, around 1150°C.

The interchange between Co and Fe can be seen by studying the liquid-liquid reaction between the oxides in the slag and the metals in the alloy:



A recovery equation was derived by Jones, [5] to express the recovery of cobalt to the alloy, R_{Co} , in terms of the recovery of iron, R_{Fe} , the equilibrium constant K , and the ratio of the activity coefficients γ . This is shown in Equation [2].

$$R_{Co} = \frac{K\gamma \cdot R_{Fe}}{1 - (1 - K\gamma)R_{Fe}} \quad [2]$$

It is certainly possible to calculate a value for $K\gamma$ from published theoretical data (if this is available for the particular system and conditions of interest), but this would only apply strictly to a perfect equilibrium system. It may be more useful to use the form of the theoretically derived equation, and to fit the model to actual plant data. Values of $K\gamma$ may be found by fitting Equation [2] to experimental data. For illustrative purposes, the curve in Figure 1 below shows a value of $K\gamma = 14$. (A comparison will be shown later in the paper between the model and experimental data.)

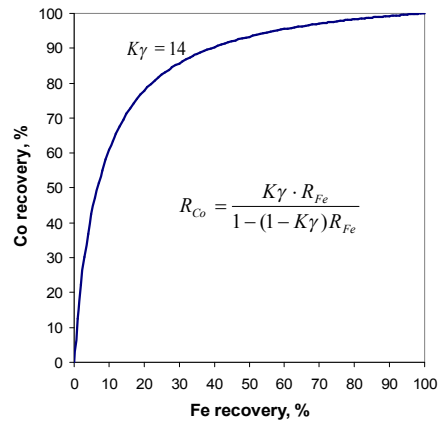


Figure 3: Illustrative relationship between Fe recovery and Co recovery to the alloy

Application to Co recovery by slag cleaning

Mineralogical studies had shown that cobalt was present largely as CoO in the Nkana dump [6]. Copper in this slag is mainly present as copper-rich sulphides. Reductive smelting of the slag, using carbon as the reductant, permits recovery of the valuable cobalt into an iron-rich alloy. The cobalt oxide, and, to a lesser extent, the copper oxide associated with the silicate/oxide phases, is reduced by Fe from the alloy to form metallic Co (and Cu), resulting in the formation of FeO in the slag. The CoO in the slag is associated primarily with fayalite (Fe_2SiO_4) and analysis by scanning electron microscopy showed some Fe_2SiO_4 particles with no detectable Co or Cu, thus demonstrating that it is, in principle, possible to remove all the Co and Cu from this phase.

The equilibrium constant, K , is strictly a function of temperature only. Over the temperature range of interest (between 1500 and 1600°C), K has a value of approximately 30. For the calculation of γ , individual activity coefficients may, in principle, be obtained from the literature. However, in practice, for this cobalt-recovery slag-cleaning process, it was difficult to obtain a good enough set of activity coefficient data to calculate $K\gamma$ on a theoretical basis, and it proved to be more useful to use pilot-plant data for the empirical fitting of a value for $K\gamma$ to be used in the recovery equation.

A set of experimental data from previously reported work on the recovery of cobalt from copper reverberatory furnace slag [6] shows a fair degree of scatter and covers mostly the high Co recovery area. This data is shown (+ symbol) in the upper portion of Figure 4. The scatter is to be expected as the data covers a wide range of experimental conditions, and is based on chemical analyses that have significant

uncertainty associated with measurements of CoO and Fe³⁺ in the slag at low concentration. For a considerable period this relationship was believed to be a straight line passing with y=55-65 at x=0 depending on the cobalt head grade. Subsequent work provided further data in the low recovery area. This data is shown as solid dots in Figure 4. The combination of these two sets of data fits a curve, with a best fit obtained with a value for $K\gamma$ of about 14.

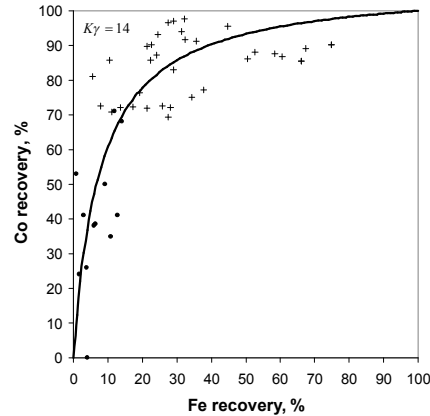


Figure 4- Recovery curve for Co ($K\gamma = 14$) and two sets of data points

The $K\gamma$ recovery equation produces curves of the correct functional form that are applicable across the whole range of different extents of reduction. $K\gamma$ for a particular metal has a characteristic value that depends on the temperature of operation and to some degree on the composition of the slag and metal system under consideration (as this affects the individual activity coefficients). Process design calculations can be carried out with confidence, knowing that there is good theoretical justification for the form of the equation, and that the single parameter $K\gamma$ is based on experimental work. The equation clarifies the dependency of the recovery of the valuable metals on the recovery of iron in the furnace, and this allows a furnace operator to control the smelting process more easily and to target the correct degree of reduction.

BATCH SMELTING TESTWORK

Initial batch testing 120kW level

Avmin commenced small-scale metallurgical testwork using a DC furnace at the 120kW level in 1997 at its own research facility in South Africa, even before the sale of the dump was concluded. This was intended to provide key data for the initial evaluation of grades and recoveries that were used for the valuation of the dump. Much of this early work provided a feel for the effect of the head grade of the slag on the cobalt recovery, the optimum level of iron reduction, the terminal grade of cobalt in the slag and the grade of alloy produced. The furnace was operated in batch mode and provided initial feedstock for atomisation tests and leaching testwork.

This testwork quickly established a number of key issues. The first was that excellent cobalt recoveries (exceeding 80%) were possible without the need to reduce more than about 30% of the iron. It was also established that there was a “terminal” grade of cobalt (around 0.1-0.2%) remaining in the slag that proved impractical to recover even with large excesses of reductant and high degrees of iron co-reduction. The implication of this was that it was feasible to recover a high percentage of the cobalt from high-grade slags into a high-grade alloy, while conversely the low-grade slags yielded low recoveries into low-grade alloy, which were uneconomic to process. The actual cut-off grade would vary depending on the cobalt price and operating costs, but for planning purposes the cut-off grade was around 0.5% Co.

250kw Batch level testing

As the need for more and more leach feedstock increased, the decision was made to increase the size of the furnace to produce larger batch heats, typically at the 250kW power level. The results confirmed that cobalt could be successfully recovered from dump slag by carbothermic reduction in a DC furnace. The discard slags were found to contain a terminal cobalt value that was not amenable to reduction. In some

cases the targeted degree of reduction was not obtained and the discard slag contained higher amounts of cobalt and iron than desired. The terminal cobalt value of around 0.06 to 0.12% Co corresponded to an iron content in the discard slag of 10-12%.

Another 160 plus batch tests were performed, investigating a variety of issues, including the benefits of adding lime, fluoride, a small amount of rutile slag, and various reductants. These batch tests continued between the continuous Mintek campaigns. It was evident from the results that the addition of some burned lime to the mix did measurably improve iron and cobalt recoveries, particularly for the intermediate levels of reduction likely to be used in practice. The fact that the lime did not prove as successful at the higher levels of reduction was never satisfactorily explained, but probably related to the formation of significant amounts of silicon carbide and FeSi from the silica-rich slag.

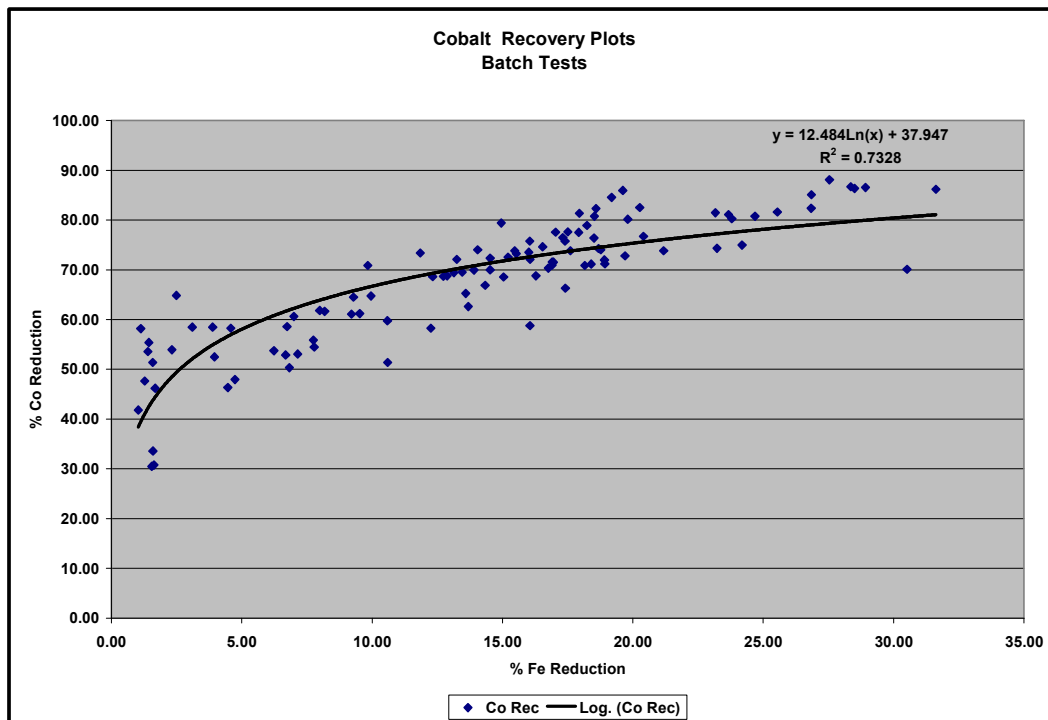


Figure 5- Batch smelting test results

As expected, the effect of the head grade of the dump slag was significant and the conclusion that higher head grades permit higher cobalt recoveries could be made, provided of course that some iron reduction had occurred.

On individual runs, the closure of the mass balances was acceptable considering the variability of the feed and the sampling errors inherent in the testwork procedure and the need to retain a metal heel in the furnace. The grades of alloy produced were very close to those predicted by the metallurgical model. When the furnace was drained at the end of each sequence the “missing” alloy returned.

Dust losses were very close to predicted levels, and averaged 0.4% of feed for the intermediate carbon additions. Dust losses increased as the quantity of reductant increased, in line with the greater volume and velocity of the off-gas, but were always below 1% of the mass of the feed slag.

In most cases, the actual degree of reduction achieved was slightly less than that predicted by the model developed from the 1996 testwork, the effect of head grade notwithstanding. Operating logs noted that on numerous occasions unburned coal was found floating on top of the slag leading to the conclusion that coal and reducible oxides were no longer in contact, and that an oxygen-depleted slag layer was

inhibiting further reduction. This observation drew attention to the physical issues of stirring, furnace geometry and associated kinetic effects, and highlighted the need for some continuous smelting testwork as opposed to batch smelting.

The actual power requirements for smelting were determined to be between 520 and 550 kWh/ton slag depending on the cobalt grade in particular. These values agreed very well with values calculated from thermodynamic data assuming the feed slag to be a previously fused complex oxide and the product slag to be a fused glassy complex. The implication of this value was that the specific smelting rate could be increased over what had been used in the engineering feasibility study. As far as power intensity was concerned the level of 450kW.m⁻² appeared to be the maximum for the particular feed size tested. Attempts to feed slag at power fluxes in excess of this value resulted in banks of unmelted slag forming in the furnace.

The influence of feed size on smelting rate had to be somewhat indirectly measured. The method used was to melt a 100kg charge of slag, without reductant, and hold it at 1550°C in contact with a metal heel in the DC furnace. Feed of various size fractions was introduced to the furnace at controlled rates and the effect on bath temperature and bath condition noted.

The four size fractions tested were: -5mm; 5-10 mm; 10-15 mm and +25 mm. The very fine fraction tended to float forming a thin crust, but this melted within seconds of the feed stopping. The medium 5-10 mm fraction behaviour was not tested. The 10-15 mm was slow to melt, resulting in the bath temperature rising by 50°C. The +25 mm fraction was even more sluggish, and unmelted pieces were still visible 5 minutes after cessation of feeding, and bath temperature also rose rapidly by 100°C, indicating that melting rate lagged power input significantly. This effect required more thorough investigation in a larger furnace, but the initial conclusion was that the feed rate and power intensity would have to be reduced if a large feed size were selected. (This would imply a larger furnace crucible to allow a longer residence time.)

The difficulties of obtaining a representative slag sample provided some interesting observations. Attempts to draw a sample of slag from the furnace using suction were unsuccessful, mainly because of the fragility of the fused silica tube used as the “syringe”. Metal rods inserted into the slag bath also produced variable results depending on where the sample was taken, and similar “lollipop” samples in the ladle showed evidence of phase segregation. Cup samples taken during tapping tended to produce the most reliable results providing the cooled sample was mechanically de-prilled using a magnet to remove alloy prills in the slag. No single sample method was completely satisfactory, and no single method consistently produced the lowest cobalt values. While individual sample errors were large, the trends were clearly distinguished, and the level of confidence in the overall results high. In spite of the addition of lime, terminal cobalt grades in the discard slag were not always achieved, particularly in the case of the high grade slags. Further work to examine more closely the effect of mixing and reaction kinetics was recommended.

For design and financial engineering purposes the optimum cobalt recovery was calculated for various cobalt head grades as follows:

$$\text{Co Recovery} = ([\text{Co}]_{\text{FEED}} - 0.115) \times 100 / [\text{Co}]_{\text{FEED}}$$

For optimum cobalt recovery, terminal iron values of 13% Fe represented a discard slag mass of 86% of the original mass, with a terminal cobalt grade of 0.1385 % Co. The complete results were published in a series of internal reports [7, 8].

SHORT-DURATION CONTINUOUS SMELTING TESTWORK AT MINTEK

In support of the design of the commercial installation, Mintek and Avmin conducted collaborative testwork, up to October 1999, at the 1 to 2MW scale, in which approximately 840 tons of Nkana dump slag (ranging from 0.66% Co) was successfully processed in a 3MW DC arc furnace. Good overall cobalt extraction was achieved during the tests, and approximately 100 tons of cobalt-bearing alloy was produced (containing 5 to 14% Co). Various refractories were tested in the smelting campaigns. The

testwork demonstrated that Nkana dump slag could be successfully processed in a DC arc furnace of suitable design, to produce a cobalt-bearing alloy amenable to further hydrometallurgical processing.

The large-scale testwork was set up in such a way that it would mimic the mode of operation of the industrial-scale furnace. Approximately 10% more Fe reduction was required in the large-scale testwork to achieve Co recoveries similar to those achieved in the medium-scale tests. The differences in mode of operation between the medium-scale and large-scale tests were highlighted as the probable cause for the higher recoveries of cobalt (relative to that of iron) achieved in the medium-scale testwork. The medium-scale testwork was conducted by continuously feeding mixtures of dump slag, reductant, and fluxes to an initial molten metal bath containing small quantities of slag, such that the power and feed input were balanced in order to achieve the desired operating temperature. After a preset quantity of feed, or period of time, was reached, the feed was stopped, the power reduced, and the furnace tilted in order to remove the product slag and metal. The majority of the product slag was retrieved from the furnace, and a set quantity of metal was allowed to remain in the furnace for the commencement of the next cycle. The energy requirements established in the small-scale testing were confirmed at just under 600 kWh/t feed. Details of the test campaign were comprehensively reported [9]

Examination of the furnace refractories at the end of the first 5-day campaign showed extensive erosion of the lower sidewall refractories. The erosion mechanism was extensively studied, and comprehensively reported on by Cousins, den Hoed and Nell [7, 10, 11] on, but little action was taken to implement the recommendations made in the studies.

PILOT PLANT

The rapid erosion of the hearth refractories during the first one-week duration Mintek campaign was cause for alarm, and a capital application for the installation of a 10MW pilot facility on site in Zambia was approved. This was intended to provide an opportunity for honing the operation without committing the full project to the risk of refractory or other operational problems on the furnace. Because procurement of the equipment for a 10MW facility was very much faster than for the larger project, it was anticipated that between 12 and 18 months of piloting could be completed before the 40MW furnace would be commissioned.

This project had made significant progress until the contractor for the larger furnace advised the AVMIN team that all necessary design data had been obtained from the first Mintek trial and finalisation of the design of the 40MW furnace could proceed without delay, and that piloting would yield no benefit. In the light of a seemingly airtight process guarantee, AVMIN decided to dispense with piloting and proceed with the original plan to fast track the installation of the full size plant, and the demonstration plant never materialised.

Intensive investigations into the mechanisms of refractory attack commenced, along with a search for suitable refractory materials for the “inter-tidal” zone (the area contacted by both metal and slag) capable of withstanding both a sulphur-containing, carbon-hungry alloy and a superheated slag. No single suitable material was uncovered, but a combination of a thin working lining of high alumina material backed by conductive silicon carbide blocks was proposed as an interim solution. Unfortunately the furnace dimensions had been established by the concrete dimensions and the optimum lining thickness could not be accommodated.

OPERATIONAL RESULTS

Construction delays, mainly related to the logistical challenges of transporting major items of equipment thousands of kilometers inevitably led to commissioning delays but the furnace eventually powered on and accepted feed in December 2000. The engineers had planned to ramp up feed to full production within 12 months, as shown in Figures 5 and 6 below, extracted from a consultant’s report [12]. Within a few weeks, however, the furnace was stopped because of a run-out at the metal taphole and during the stoppage serious erosion of the tidal zone of the refractory lining was observed. The furnace was

restarted but stopped again two months later because of hydration and heaving of the hearth, necessitating a hearth reline.

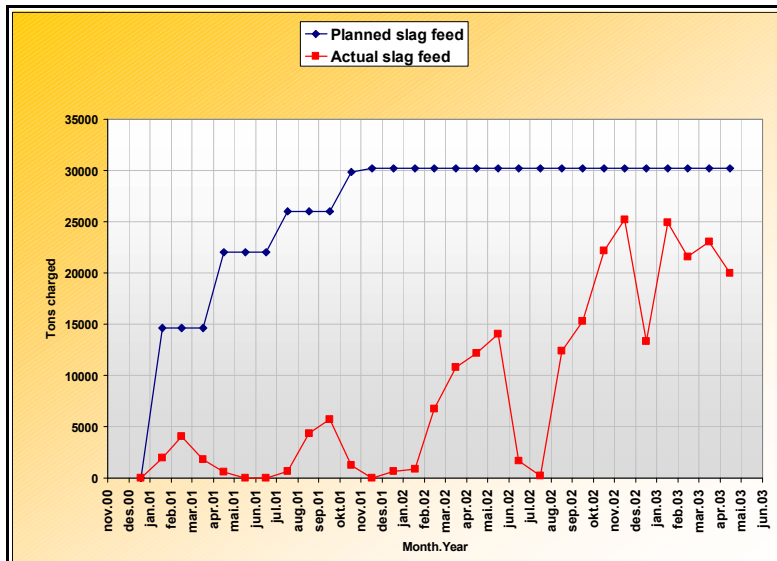


Figure 5- Furnace feed rate planned and actual. (From Jahnsen [10])

As the operations gained confidence in the furnace operation, the feed rate was increased. It was observed that periodically, scabs of frozen slag sloughed off the copper coolers, exposing them to superheated liquid slag and extremely high energy fluxes resulted. A review of the design by MacRae [13] revealed that finite element modeling performed during the design phase had utilised heat transfer coefficients many orders of magnitude lower than those applicable to the full size furnace. A substantial redesign of the copper coolers in the lower part of the furnace followed and is comprehensively described by Nelson et al [14]. The operation gradually eliminated the normally encountered teething problems around the plant and following the installation of the new coolers the furnace performed at power levels approaching the design values. Energy fluxes through the coolers were closely monitored and furnace integrity was maintained. In 2009 the owners ceased operation, citing low cobalt prices and high operating costs, and the plant was mothballed.

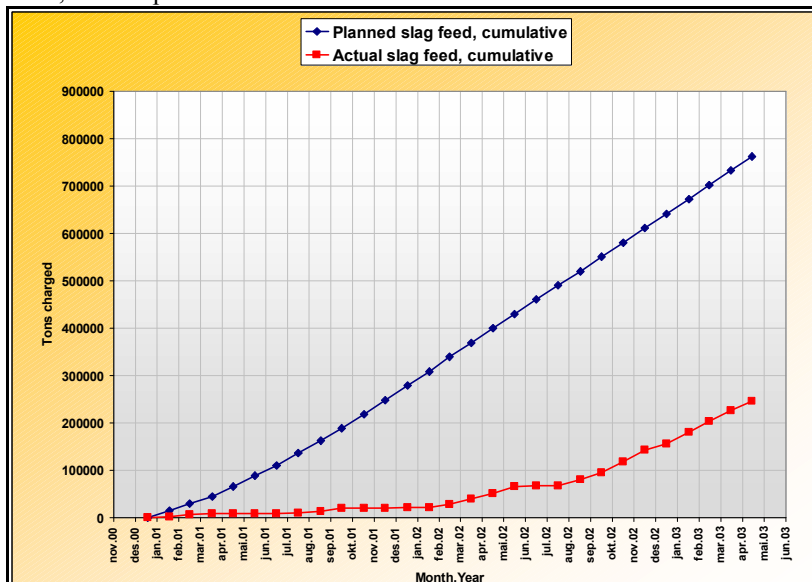


Figure 6- Cumulative furnace feed, planned and actual (From Jahnsen [10])

CONCLUSIONS AND COMMENTS- THE LESSONS LEARNED

The cobalt recovery process at Chambishi was different from anything that preceded it in that the process itself (selective carbothermic reduction of cobalt-bearing slag) was novel, as was the unit in which it was carried out. The DC arc furnace had not until then been used for this sort of application.

This process saw the first industrial implementation of solid (non-hollow) electrodes for a DC arc furnace. Previously, both chromite smelting and ilmenite smelting had hollow graphite cathodes through which the feed materials were introduced into the furnace. It also pioneered the successful use of side feeding of the DC furnace, and clearly demonstrated that radiation from the arc did not result in instantaneous destruction of the upper sidewalls, a surprisingly common misconception prior to this installation.

A very important lesson that came out of the Chambishi experience was the need for de-coupling of process steps. The atomizer required the molten alloy feed to have a certain degree of superheat. Since the tapping temperature of the alloy could not easily be controlled within the furnace, a holding furnace was required to ensure sufficient superheat prior to atomization. Unfortunately the plasma arc torch utilized was not sufficiently powerful. This led to significant freezing of the atomizing nozzle, forcing either furnace stoppages or dumping of alloy for subsequent remelting, a costly and inefficient process.

Considering the pioneering nature of the operation, and its overall complexity, and the interdependence of sequential operations, the project schedule can be regarded as ambitious, but “hindsight is an exact science”. The combination of process novelty, the plant location, and the aggressive schedule combined to turn what could have been a benchmark for metallurgical progress into a major disappointment particularly from a financial perspective.

Ten years after the plant was commissioned, and the operation stabilised, the plant does indeed demonstrate the technical feasibility of the use of DC furnaces in this type of application, and confirmed the validity of a control philosophy in which furnace power input could be monitored and adjusted to the energy flux through the coolers, where “open bath” conditions could be sustained, where fundamental thermochemical principles could be used to model and predict the metallurgical performance of the unit, and successfully integrated into the control system, as recently highlighted by Jones [4]

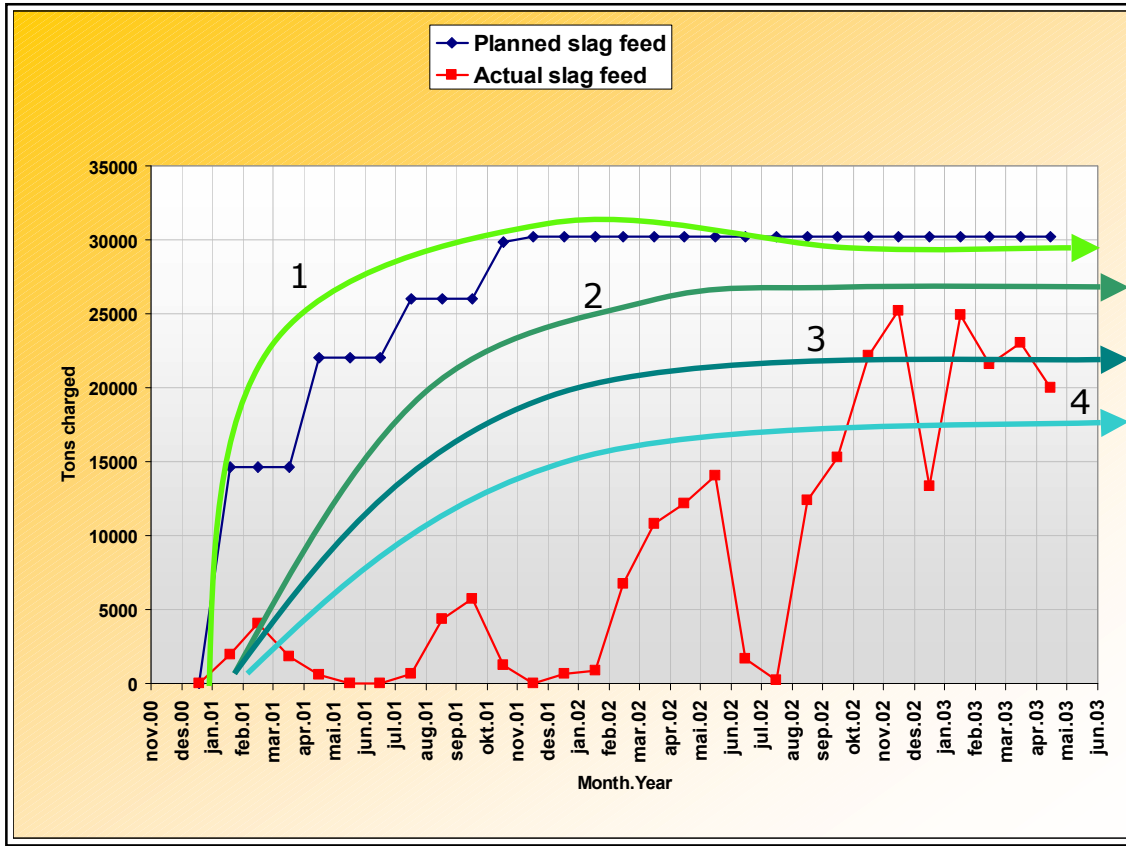


Figure 7 :- Actual performance with McNulty curves superimposed

With regard to the theme of this conference, the role of piloting with respect to implementation of new technology however, the abiding lesson to be drawn from the Chambishi experience, especially in view of the decision to dispense with piloting would have to be: “If you don’t pilot, the operation becomes the pilot plant- a very expensive one!” as pointed out by Barnes [15] and confirming McNulty’s observations as clearly illustrated by superimposing the McNulty curves on the actual plant feedrates as shown in figure 7.

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