Application of fundamentals in optimizing platinum concentrator performance

by M.M. Valenta* and H. Mapheto*

Synopsis

A number of challenges face platinum concentrator plant operators. These challenges include the increase in operating costs, the increase in smelter cost for the processing of concentrate, the shortage and cost of power, and the tightening of specifications on concentrate quality by the toll-smelting operations.

Over the years the focus has moved from extracting the platinum group metals (PGM) from the Merensky Reef to the UG2 reef. This has a number of advantages, including the higher 'basket price' for UG2 concentrate, reduced mining cost per unit volume as a result of the higher density of UG2, and the reduction in overall concentrate tonnage to be smelted. In many cases the Merensky ore has been fully exploited and it makes sense for the focus to shift to the UG2 ore that can be accessed through the Merensky shaft infrastructure. The presence of relatively high levels of chromite in UG2 concentrate is, however, a major disadvantage due to the problems associated with smelting such a concentrate in conventional submerged arc furnaces.

In addition to increasing the specification on the minimum PGM grade of concentrates, smelters have had to impose strict specifications on the levels of chromite in the concentrate. The threat of high penalties has forced concentrators to change their modus operandi, often resulting in a significant loss in recovery.

The final concentrate grades and PGM recoveries are shown to vary significantly throughout the industry. The reasons for this include varying ore mineralogy and different operating philosophies. This would therefore imply that the opportunity exists to optimize the operations by considering fundamental aspects such as the PGM mineralogy and the application of appropriate technologies.

By returning to the fundamentals of flotation and applying the findings of detailed process reviews, it has been possible to increase the concentrate PGM grade, reduce the concentrate chromite grade, and in some cases increase the recovery of PGM to concentrate. This paper presents case studies where this approach has been used to successfully optimize concentrator performance, resulting in lower operating cost and higher PGM production.

Keywords
Process optimization, platinum concentrator, milling, flotation, grade, recovery

Introduction

The volatility in the platinum market, combined with the fluctuation in the operating cost observed since 2008, has motivated operations in the platinum industry to optimize their processes. This has not only been the case for the smaller operators in the industry who have limited resources, but also for the larger organizations that recognized that a fresh un-blinkered perspective may be needed.

The concentrators have been threatened by increases in the cost of labour and consumables as well as higher smelter costs. Direct smelter costs are linked to the grade of concentrate related to overall concentrate tonnage produced. Additional smelter costs are incurred as penalties that have resulted from more stringent specifications on the minimum PGM grade and maximum chromite grade (in the case of UG2 concentrate).

Advances in technology have offered avenues to increase recoveries and concentrate grade. However, this would require the investment of significant amounts of capital that has not been readily available in recent times.

Concentrators have been forced to re-evaluate their current operations and identify opportunities to improve recoveries and grades. This paper presents case studies where, through analysis of current operations and the application of fundamental principles, improvements have been made in concentrator performance.

The case studies have been limited to UG2 concentrators due to the added complexity in the UG2 concentrator design and more stringent concentrate specifications.

The UG2 Reef

The UG2 ore is a platinum-bearing chromitite ore that contributes a growing proportion of the platinum group metal (PGM) production from the Bushveld Igneous Complex in northern South Africa (McLaren and de

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Villiers, 1982, Phillips *et al.*, 2008). The complexity of design of a UG2 concentrator circuit is primarily due to the PGM mineralogy and the fact that the ore contains multiple gangue phases with significantly different characteristics, i.e. silicates and chromitite (Valenta, 2007).

The ore contains trace amounts of base metal sulphides that, despite their low concentration, are of importance as they frequently occur in association with many of the PGM-bearing minerals. The copper head grade varies between 0.005% and 0.02%, and the nickel grade varies between 0.025% and 0.05%.

The mode of occurrence of the PGMs in the ore is complex and the PGMs occur in various minerals including metal alloys, sulphides, oxides, tellurides, etc. PGM-containing grains in UG2 ore are small and rarely exceed 30 micron in diameter. The average grain size can be as fine as 6 micron. This complex mineralogy poses a challenge to the metallurgist from a mineral liberation and extraction perspective.

The PGM grade of the feed to the UG2 concentrators typically varies from 2.0 g/t to 5.5 g/t. The feed grade is affected by the stoping height in the mine, how much of the hanging wall and footwall is included in the mining cut, and the presence of gangue partitions between the chromitite seams. The chromitite content in the feed can vary from as low as 25% to as high as 40% (Philips *et al.*, 2008).

Processing of UG2 ore

The supply chain for the production of PGMs is complex, as is illustrated in Figure 1. The first metallurgical stage in the production of platinum group metals and gold (PGM+Au) is the production of a concentrate as feedstock to the smelter. The smelter-converter stage is followed by a refining stage for the production of primarily copper and nickel, and a further refining stage for the production of the individual PGM+Au metals.

This paper focuses on the optimization of the concentrator operation that has an influence on all the downstream processes. The preferred concentrator process for the extraction of PGM to concentrate is akin to sulphide flotation and also results in the recovery of the base metal sulphides (BMS). This process utilizes the natural and induced hydrophobicity of the PGM minerals to separate the PGM minerals from the gangue phases. The success of the process is thought provoking given the low concentration of BMS and PGM sulphide minerals. However, recoveries in excess of 90% have been achieved.

The presence of multiple gangue phases and the relatively fine grain size of the PGM require a more involved circuit design than is typical for a conventional base metal sulphide circuit.

The entrainment of chromite to the UG2 flotation concentrate adds another challenge to the concentrator operator in that limits are placed on the amount of chromite allowed. Excess chromite in the concentrate results in the build-up of highly refractory chromite spinel layers in the furnaces, thus affecting furnace operation furnace availability and increasing operating costs (Philips *et al.*, 2008).

The circuit that forms the basis of most UG2 concentrators in the industry is illustrated in Figure 2. This is commonly referred to as an MF2 circuit. The particle size of the feed to the primary flotation circuit is typically 35% passing 75 μm and that of the feed to the secondary circuit is typically 75% passing 75 μm. The conventional flotation stages typically consist of rougher, cleaner, and recleaner stages.

There have been a number of advances in the design of the circuit, including the cleaner cell configuration, the open circuiting of the cleaner tailing stream and separate secondary processing of the chromite and silicate-rich streams. Ultrafine grinding of tailing steams has also been introduced with grinds as fine as a P80 of 53 microns resulting in improved recoveries.

A typical reagent suite for UG2 flotation includes an activator ($CuSO_4$), collector (SIBX), CMC depressant (carboxy methyl cellulose) and frother (DOW200) (Overbeek, Loo and Dunne, 1984). These reagents are added in various proportions in a variety of addition points throughout the circuit.

UG2 concentrator grades and recoveries

The graph in Figure 3 presents data gathered by the authors that illustrate the typical variability in the concentrate grade produced on UG2 concentrators operating on the Bushveld Igneous Complex. In addition, the PGM recovery can vary from 70% to 90%. The variability in the results can be attributed to PGM mineralogy, circuit configuration and operating philosophy.

Amendments to the penalty clauses in the toll treatment agreements have become more stringent, and concentrates with a Cr_2O_3 grades exceeding 1.5% start incurring penalties. Some toll smelters will not tolerate concentrates exceeding 3% for long periods, and instances have occurred where concentrates have not been accepted.

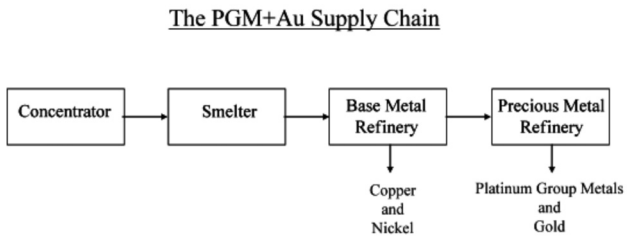

Figure 1—The platinum group metal supply chain

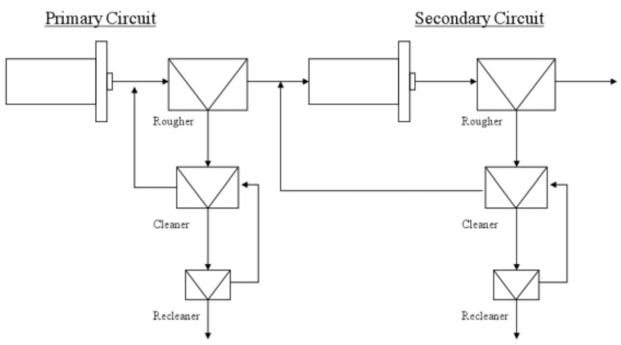

Figure 2—Typical circuit for the processing of UG2 ore

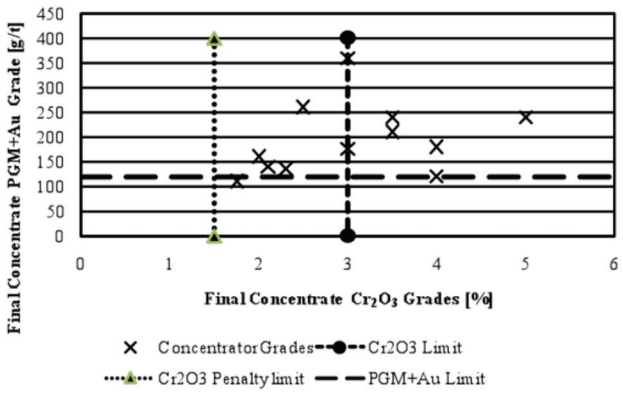

Figure 3—Variability in concentrate grade for UG2 concentrators

It is interesting to note that no UG2 concentrators within our database are able to achieve the minimum of 1.5% Cr_2O_3. Some concentrators producing concentrates with high Cr_2O_3 grades are saved by blending their concentrates with low Cr_2O_3 containing concentrates from Merensky concentrators or other UG2 concentrators producing low Cr_2O_3 grade concentrates.

Considering the sub-processes in flotation

It has been widely accepted that the process of flotation is a combination of a number of individual sub-processes. Simply put, the process whereby solids and water are transferred from the pulp phase to the froth phase can be described by considering the sub-processes of true flotation and entrainment (Bradshaw *et al.*, 2005).

True flotation can be described as the selective chemical sub-process where hydrophobic minerals are attached to the bubbles rising through the pulp. Entrainment is described as a non-selective physical sub-process where solids and water are transferred to the froth in the bubble lamella and bubble interstices.

In the recovery of PGMs and BMSs to the concentrate, the predominant process is true flotation. The predominant process responsible for the recovery of gangue phases in the UG2 flotation process is, however, entrainment, with a lesser proportion being recovered by flotation.

Both sub-processes affect the grade of the final concentrate and the metallurgist must consider the interaction between the sub-processes and the effect these processes have on the response of the various minerals in the ore.

Parameters to consider in flotation

In controlling and optimizing a flotation plant, various parameters have to be considered. These can be grouped as those parameters that are inherent in the ore, parameters that are related to the mechanical equipment, and the operational parameters relating to the environment within the solids-water mixture. (Table I)

The parameters can be controlled to a lesser and greater extent by the metallurgist to optimize the circuit and maximize the amount of saleable PGM.

Optimization methodology

The process of conducting a review of the operation of a concentrator varies for every concentrator depending on the available metallurgical resources, the state of the operation, and the level of instrumentation/control being exercised on the plant.

The authors' experience has been that the establishment of a formal steering committee including key role players from the operation as well as outside consultants has had a better chance of success than simply introducing outside consultants or expecting the in-house metallurgists to simply continue with their duties. The steering committee should include key role players such as the senior engineering personnel and operations personnel so as to ensure continuity and commitment.

The process of optimizing the plant operation includes:

- A review of the design test work upon which the plant was designed
- A review of the mineralogy of the ore
- Gaining an understanding of the geology of the deposit
- Analysis of the historical operating data and results
- Analysis of the plant running time and availability of unit operations
- Generating of a mass balance of the current operation to assess the unit capacities, residence times, stage efficiencies, upgrade ratios and flotation response
- Laboratory flotation tests of samples taken from the plant to determine flotation characteristics
- Conducting assay-by-size analysis of the various streams
- Developing various scenarios based on the data
- Identifying opportunities for process optimization
- Determining the cost of changes
- Optimizing operational parameters
- Implementation of recommendations.

The most successful exercises have been those where the consultant is part of a steering committee and decisions are made as a collective. Decisions are seen as group decisions and not as instructions from an outside party. In many cases this does result in debate and discussion; however, once the decision has been taken the collective can then move forward.

Applying the flotation fundamentals

In a number of cases the authors have found that in optimizing a PGM flotation circuit it has been necessary to

Table I

Parameters to consider in optimizing a flotation circuit

Inherent parameters	Mechanical parameters	Operational parameters
PGM mineralogy Feed grade Gangue dilution	Circuit design Type of equipment Classification circuit Flotation cell geometry	Mill power Pulp density Reagent type Reagent dosage Reagent dosage point Froth depth Aeration rate

consider the fundamental processes occurring within the flotation cell. For the purposes of this paper the authors have divided the case studies into four areas, namely, the effect of pulp characteristics, the reagent suite, the mineralogy, and the cell geometry.

Considering the pulp characteristics

In the first instance it is important to consider the mixing environment within the flotation cell. The duty of the impeller arrangement is twofold: firstly to suspend the slurry within the cell, and secondly to disperse the air within the flotation cell.

The efficiency of the impeller is governed by the impeller size, speed, geometry and cell geometry. The mixing efficiency is also governed by the viscosity of the slurry that is affected by the particle size distribution, the slurry density and the nature of the ore. High viscosity slurries inhibit the dispersion of the gas within the slurry. The effect of high slurry viscosity on flotation performance has been described by Nel *et al.* (2007).

Slurry density also has a significant effect on the entrainment of solids, and the effect of high slurry density on the entrainment of chromite has been described (Valenta, 2007).

Ensuring a balance in the reagent suite

The importance of maintaining a balance in the reagent suite has been discussed by Valenta (2007). Frother has been shown to have a significant effect on the entrainment of solids to the concentrate. Particularly in the case of chromite entrainment, this has been shown to be significant.

The amount and type of collector used is important in determining what floatable minerals species are recovered to the concentrate. It is has been shown that collector can affect the amount of entrained solids being recovered to the concentrate (Valenta, 2007) and that the importance of collector choice must not be underestimated.

The use of depressant in the depression of floatable gangue species is well understood in the industry (Bradshaw *et al.*, 2005). The balancing of the depressant addition with the dosage of collector and frother addition is very important in the optimization of a PGM flotation plant.

Considering the flotation response of the ore

A key consideration in plant operation is the variability in the floatability of the various mineral species in the ore. The flotation response is governed by a number of factors including what minerals species are present and their relative abundance, the degree of oxidation, the degree of liberation, and the mode of occurrence.

This can be determined by comparing the flotation response of samples in the laboratory taken from the plant and studying the resultant flotation kinetics. Furthermore, the capacity to perform detailed mineralogical analysis has increased significantly of late, allowing the metallurgist to quantify the relative abundance of the mineral species, the degree of liberation, and the mode of occurrence of the various minerals. Unfortunately there is still a time delay of up to three months to obtain detailed mineralogical analysis.

Understanding the floatability of the minerals in the ore is not only important in determining what flotation capacity (residence time) will be required in each stage of the circuit, but will also govern the operating strategy that is to be applied to each flotation class. Fast floating minerals require very little residence time and the authors have found that they are relatively immune to high depressant dosage resulting in relatively high concentrate grades and recoveries.

Slow floating minerals are, however, prone to yield low recoveries at high grades and should therefore be treated in a different circuit.

Cell geometry and the flow of concentrate

The cell geometry affects the hydrodynamics within a flotation cell and the relatively high specific gravity of chromite often results in chromite settling at the bottom of flotation cells. This effectively reduces the residence time in the cells and increases the risk of cells sanding up. This is also governed by the relative position of the impeller arrangement, the design of the impeller arrangement, and the aspect ratio of the flotation cell.

The relative geometry between the cell volume and launder lip is an important factor when considering a cell for a specific duty within a flotation circuit. Does the operator require a short lip length for finer grade control or longer lip to maximize recovery at the expense of grade?

Case studies

This section of the paper discusses various case studies, illustrating how the findings of a process review and the subsequent implementation of changes to fundamental operating parameters have improved the efficiency of operations.

The fundamental parameters discussed are pulp density, cell geometry, flotation kinetics, circuit configuration, and the reagent suite.

The case studies and the changes made may seem to be simple and obvious within the context of this paper. However, on a complex UG2 concentrator these parameters are normally clouded by other challenges that hide the true source of the poor efficiency. The findings presented were the outcome of detailed structured process reviews.

In the case studies graphs are presented over the period prior to the change and post the change to illustrate the effect of the change in flotation parameter. Due to the inherent noise in the data characteristic of a concentrator operation, a seven day moving average is plotted to illustrate the trend.

All PGM analysis reported was done by fire assay with lead collection and the analysis is an indication of the content of the four elements (4E) i.e. platinum, palladium, rhodium and gold in the samples.

Case study 1—changing the pulp density to improve cell performance

The metallurgical team was approached to help in the optimization of the concentrator as the flotation recoveries were low and did not meet the recoveries indicated by the laboratory and pilot-plant test work.

The process review conducted by the metallurgists revealed that the flotation recovery in the primary rougher bank of the plant was very poor in comparison with the test

work findings and typical recoveries observed on other operations. In addition, the chromite content of the rougher concentrate was very high resulting in the final concentrate not meeting specification.

Mineralogical analysis of the primary rougher tailing identified a significant proportion of the PGM as being liberated sulphides PGMs that would normally be recovered. This implied that the losses were flotation process related, and not due to insufficient mineral liberation.

Attempts to increase the aeration rate to the flotation cells did not improve the recovery and the cells tended to geyser, with large bubbles rising to the surface and disturbing the froth indicating that the flotation cell mechanisms could not disperse any more air.

A number of changes were made to the reagent suite, including an increase in the addition of copper sulphate, the introduction of a dithiophosphate collector, the introduction of an alternative xanthate collector, and an increase in the overall collector dosage. The changes to the reagent suite were not successful, and the mass recovery to the concentrate remained low.

An analysis of the availability of the primary rougher flotation cells revealed that the flotation cells tended to trip regularly on electrical overload. A survey of the operation found that the flotation cells were drawing well in excess of the rated loading of the motors (3 kW/m^3 of flotation cell volume).

Running the flotation cells empty indicated that the high power draw was not due to mechanical problems but rather due to the high pulp density at which the cells were being operated. The relative pulp density in the feed to the first rougher cell was constantly above 1.5, with the last rougher cells operating at relative pulp densities in excess of 1.65.

The high pulp density/viscosity would account for the high power draw under load and the poor recovery as the pulp density would inhibit the dispersion of gas in the flotation cells. A similar situation was previously reported on cleaner flotation cells on a UG2 concentrator (Nel *et al.*, 2007).

It was recommended that the pulp densities be reduced, and the improvement in primary rougher recovery is illustrated in Figure 4. The rougher concentrate sample is not normally taken on shift, and special samples had to be taken as part of the optimization programme. The primary rougher recovery was calculated using the two-product formula (using the shift sample assays for the feed and tailing streams) and the corresponding average pulp density was extracted from the data historian.

The change resulted in the operators being able to increase the air to the flotation cells without a repetition of the geyser effect. The ability to add more air also allowed for changes to be made to the reagent suite, specifically an increase in depressant to change to froth stability.

The control of pulp density and effect of pulp density on the operation of the flotation cells are often overlooked. The opportunity for the design metallurgists to increase the pulp density in flotation circuits in order to reduce the overall capital cost of the project is very attractive. However, consideration must be given to the effect of operating at high pulp densities on the operability of the flotation cells.

The high chromite content in the concentrate can also be explained by the high pulp density. The effect of pulp density on the entrainment of gangue mineral has been discussed (Valenta, 2006) and this fact must be taken into account when designing the circuit.

Case study 2—reagent optimization in reducing chromite in concentrate

In this case the concentrator was not able to meet the stringent Cr_2O_3 grade specified in the smelter contract. A plant audit was conducted and it was revealed that the overall frother dosage was very high and that frother was being added to the cleaner flotation stages.

The study of the effect of frother type and concentration on the entrainment of particles has been the topic of much work (Laskowski, 1993, 2004). The effect of frother dosage on the entrainment of chromite to the flotation concentrate has been presented in other work (Valenta and Harris, 2007).

Based on the success in controlling the concentrate chromite content on other plants, as reported in previous publications (Valenta, 2007), the chromite in concentrate on this concentrator was significantly reduced by reducing the frother to the primary and secondary rougher stages, as illustrated in Figure 5. The operators were given the latitude to change the frother dosage to the two rougher banks. However, the combined frother dosage was not allowed to be

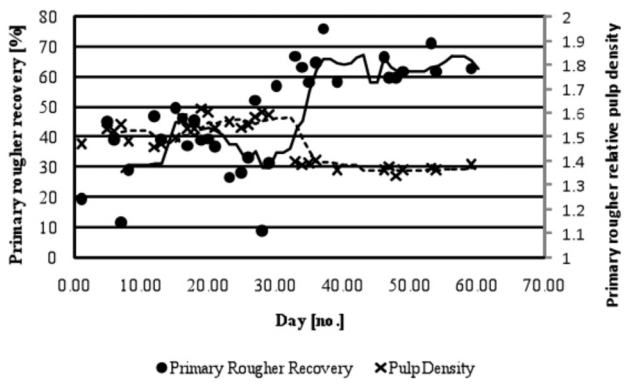

Figure 4—Effect of reducing pulp density on primary rougher efficiency

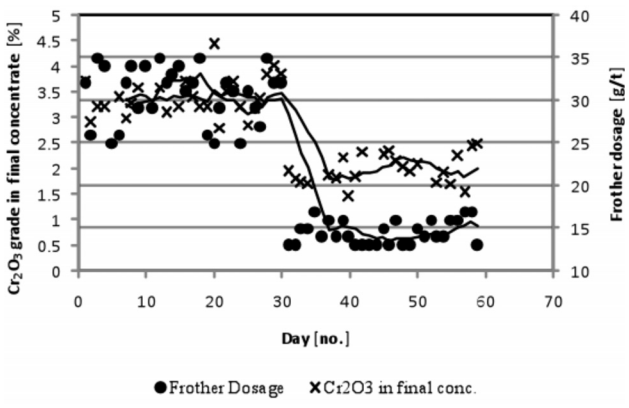

Figure 5—Effect of frother dosage on the chromite content in concentrate

more than 17 g/t of feed. Although the Cr_2O_3 grade is still above the 1.5% mark, the risk of the concentrator having the concentrate returned is significantly reduced. The smelter penalties were also significantly reduced.

It is interesting to note from the Figure 5 that, once the effect of frother had been demonstrated to the operation personnel, the variation in frother dosage has reduced significantly.

It must be noted that the change in frother dosage was not done in isolation. As discussed in the previous paper (Valenta, 2007) the control of concentrate grade requires a balancing of the reagent suite. The reduction in the frother dosage also required a concomitant increase in the collector dosage to the cleaner bank and a decrease in the depressant dosage to the rougher bank.

Case study 3—meeting grade specifications by optimizing the circuit

In certain cases the design of the flotation circuit does not suit the ore mineralogy and the flotation kinetics exhibited by the PGMs.

Matching the flotation kinetics of the various concentrates to allow for the production of a high grade primary concentrate and a low grade secondary concentrate has found favour in the UG2 industry.

In this specific case the feed to a UG2 concentrator changed and it was found that the concentrator could not achieve the concentrate specification stipulated by the smelter agreement. Addition of depressant was not successful as high depressant dosage resulted in the cleaner cell froth collapsing. A survey was conducted and the opportunity was identified to increase the grade of the concentrate without compromising the overall flotation recovery.

Flotation tests were done on samples taken of the various rougher flotation concentrates and it was found that the concentrates could be grouped according to the flotation response in the laboratory cell. It was also observed that the concentrates appeared to respond differently to depressant dosage, with some froths collapsing while other froth remained stable and yielded high concentrate grades.

Based on the findings of the laboratory test work, primary and secondary rougher concentrates exhibiting similar flotation behaviour were combined in different cleaner circuits, as illustrated in Figure 6.

The results of the change in the flotation circuit are illustrated in Figure 7.

It must be stressed that the change in flotation circuit alone did not result in an increase in the concentrate grade but it allowed for the optimization of the reagent suite. Where previously high depressant dosage to the primary cleaner circuit had resulted in a collapse in the flotation froth, this change in the circuit allowed for the addition of high depressant dosage with no loss in flotation recovery for the whole circuit.

The final concentrate grade of the overall circuit increased and the risk of not meeting the minimum grade specification was addressed.

Case study 4—increasing the PGM grade by changing the cell geometry

Flotation cells are supplied in different shapes and sizes depending on the hydrodynamics within the flotation cell. Furthermore, various configurations of concentrate launders with different lip lengths have been designed to cater for the flow of concentrate from the cell. These changes to the concentrate launder lip length are driven by a number of factors, including the floatability of the mineral and the desired grade of concentrate required from the cell in whichever application.

The metallurgical team was faced with the dilemma that the desired concentrate grade could not be achieved even though both the depressant dosage had been increased and the aeration rate had been reduced to a minimum. In other words, the flow of concentrate could not be reduced any further without resulting in a collapse of the froth and no concentrate being recovered.

It is known that flotation concentrate flow rate can be controlled by the adjustment of the cell concentrate weir (Woollacott and Eric, 1994). In banks of trough cells consisting of up to four cells per bank, cell slats have been used to control froth depth in one cell relative to the level in other cells. In so doing the introduction of cell weirs result in a reduction in the effective lip length in the bank of flotation cells.

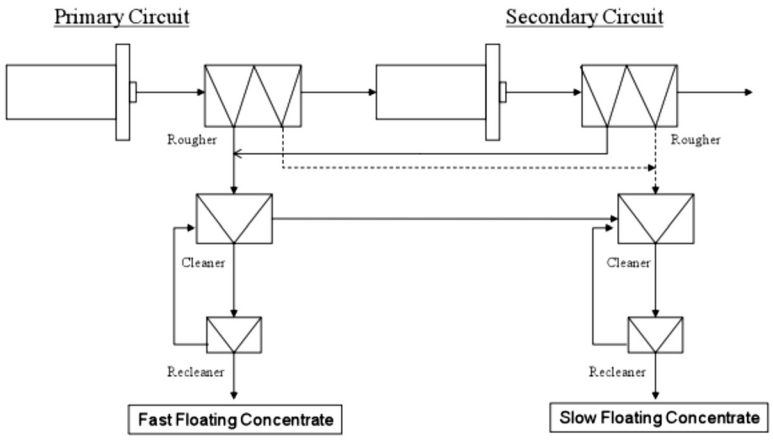

Figure 6—Changes made to the flotation circuit based on flotation kinetics

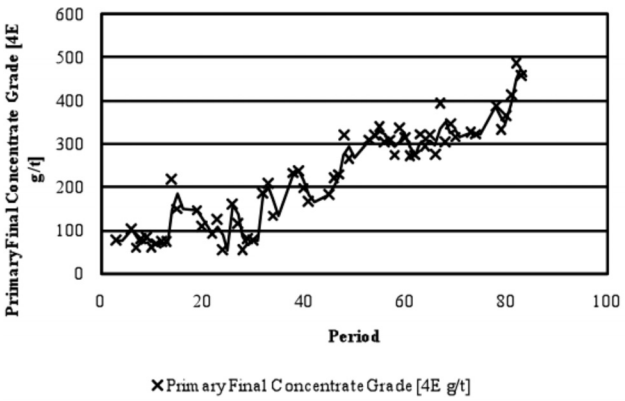

Figure 7—Increased primary concentrate grade with change in flotation circuit

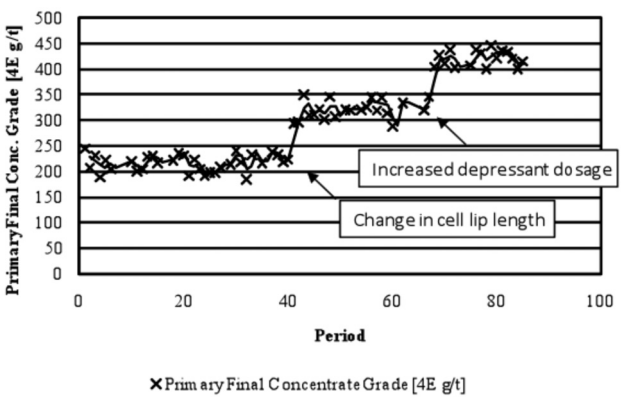

Figure 8—Increase in grade with a change in lip length

In this case the flotation cells used as primary final cleaner cells were tank cells that were introduced as part of a circuit configuration exercise similar to the one described in Figure 7. Tank cell pulp levels are normally controlled per cell and the need for slats on tank cells has not been a requirement.

Based on the successful use of slats on trough cells to control concentrate flow rate on other operations, it was recommended that slats be installed at regular intervals on the circular cell lip. The improvement in concentrate grade with a reduction in the lip length is illustrated in Figure 8. The change in the lip length allowed for a further increase in the depressant dosage with a concomitant increase in the final concentrate grade.

Conclusions

The operator of a UG2 concentrator is faced with a number of challenges from a concentrate grade and recovery perspective. In some cases operations have to compromise the recovery in order to meet strict concentrate grade specifications imposed by the smelter. This is made difficult by the presence of two major gangue phases in the ore and the complex mineralogy of the PGM minerals.

The case studies discussed illustrate that the opportunity exists to optimize the concentrators through a process of detailed analysis, identification of problem areas and the optimization of the fundamental flotation parameters. The changes do not have to involve complex costly circuit changes and incremental improvements can be made to the concentrator performance through the implementation of changes to the fundamental parameters.

The optimization methodology has been successfully used on UG2 and Merensky operations and will be applicable to most flotation plants.

The establishment of a team including the consultants, maintenance and operational staff is paramount to the success of any such optimization endeavour. Alienation of the metallurgist is very common in cases where the metallurgist is introduced as a 'fixer'. That is not the objective, and a common goal for the identification of opportunities and optimization of the operation must be established from the outset.

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