

UG2 metallurgical variability

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It is important to understand the metallurgical variability of an ore since this can drive the economic viability of some projects. A poor understanding of the metallurgical variability can (and has) led to shortcomings in strategic and tactical forecasting for some operations. Therefore this study was carried out to highlight the metallurgical upgrading variability of the Upper Group two chromitite layer (UG2) from the Bushveld Complex in South Africa. Forty five different UG2 samples were received from several different platinum operations and characterized using a standard laboratory flotation flowsheet to highlight the metallurgical variability. Two metallurgical parameters, i.e. maximum recovery (R_{max}) of values (for Pt, Pd and associated Au, 2PGE+Au) and a relative cleaner index (RCI, descriptive of the sample upgrading characteristic), were used to characterize the samples. Furthermore, there was no definitive relationship between the metallurgical and the head assay parameters measured, indicating that the metallurgical response has a more complex relationship and cannot be attributed to a single parameter or even simple combinations of the parameters.

Keywords: Metallurgical variability, UG2 processing, flotation, classification of ores.

Introduction

The principal function of project evaluation is to determine the potential profitability of a project. In mining, the project profitability is influenced by many factors such as mining methodology, ore treatment methodology and production rates. However, in the buildup to project evaluation a feasibility study must be carried out to assess the mineral resource and reserve. Mineral resource refers to the *in situ* mineralization estimates whereas mineral reserve refers to the mineable production estimates. To translate a resource into a reserve certain 'modifying factors' need to be considered; these include: mining, metallurgical, economic, marketing, legal, environment, social, and governmental factors.

During the exploration phase of a feasibility study, a geostatistician is tasked with planning a drilling program to calculate the resource, and project principals have little option but to conform if they want a compliant resource. On the other hand, the test program for reserve calculations is often limited to leftover geological samples from the resource calculation program. The result of this is a poorly characterized deposit where reserve calculations are often not near the level of confidence that they should be.

Historical records of test work carried out by Mintek for some platinum operations (processing UG2 ore) showed that the level of attention (largely due to capital and time constraints) given to understanding the metallurgical variability for reserve calculations across the deposit is limited. Furthermore, the test program and procedure used to understand the variability differs from one sample to another, which makes it difficult to interpret the information for a better understanding of the metallurgical variability. Therefore the premise of this study was to illustrate the macro variability of UG2 samples taken from

across the Bushveld. Mintek was supported in this endeavour by provision of samples from Anglo Platinum, Lonmin, Impala, Two Rivers Platinum, Barrick-Sedibelo, Eastplats, Xstrata, Aquarius, and Northam to illustrate the extent of UG2 variability on a regional scale.

This publication presents and discusses the results of an investigation aimed at highlighting the metallurgical variability of the UG2 ore. A laboratory cleaner rate was carried out to characterize the samples. Two metallurgical parameters, i.e. the maximum values recovery (R_{max}) and the relative cleaner index (RCI) for the values (2PGE+Au) were used to characterize the metallurgical response of the ore.

Materials and methods

Materials

Forty five UG2 samples were received from several platinum operations, operating in the Bushveld Complex in South Africa. Each operation was urged to submit at least two samples, one that they believed to have a 'bad' metallurgical response and the other which they believed to have a 'good' metallurgical response. Prior to taking each sample, each operation was notified that the samples should have the following features:

- Be taken at the correct mining depth
- Have the operation's typical mining width, i.e. the chromitite reef with a certain proportion of the hanging and footwall added
- The bad samples must not be taken from areas affected by ironrich ultramfic pegmatite (IRUP), potholes or be taken from near surface
- Be supervised and approved by a senior geologist
- Be accompanied by a geological log sheet
- Samples can be taken as drill core or rock chippings.

Methods

Sample preparation and head assay

Each sample was inspected visually prior to preparation for flotation test work. A jaw crusher was used to reduce the sample to -6 mm; this was followed by a cone crusher which was used to reduce the sample to -1.7 mm. The -1.7 mm material was homogenized and three 200 g sub-samples were taken for head assay. The remaining material was split into 1 kg batches for flotation test work using a rotary splitter. The samples for head assay were pulverized using a steel bowl and assayed for:

- Pt, Pd, Rh, Ru, Ir and associated Au (5PGE+Au) using a standard fire assay technique with nickel sulphide as a collector followed by crushing, leaching and dissolution of the nickel sulphide button in aquaregia and analysis for the six elements by ICP-OES
- Pt, Pd, Rh and associated Au (3PGE+Au) using a standard fire assay technique with no high temperature cupellation followed by pressure dissolution of the prill using sealed glass tubes and analysis of the four elements using ICP-OES
- Pt, Pd and associated Au (2PGE+Au) using a standard fire assay technique with no high temperature cupellation followed by dissolution of the prill in aquaregia and analysis of the three elements using ICP-OES
- Cr₂O₃ by fusion of the sample followed by acid dissolution in HCL or HNO₃
- Total sulphur by combustion, LECO
- Sulphide sulphur by the difference in the total sulphur and sulphate sulphur
- Acid soluble Cu, Ni, Fe and Co determined by acid digestion followed by analysis by ICP-OES
- SG determined by a pycnometer
- Furthermore, a XRF and XRD scan was carried out on each sample.

Metallurgical test work

A simplified flowsheet was adopted to determine the metallurgical response of each ore. A rod mill and a conventional Denver D12 float machine were used for the test work. The milling time required to mill each sample to a fixed grind could not be established for each sample due to the limited sample mass provided by some operations, typically < 4 kg. Therefore for consistency a fixed milling time required to get approximately 80% of the material to pass 75 µm was selected for test work. The mill time was 43 minutes and can be justified by Mintek's extensive library on UG2 processing which was supplemented by generating three rod mill curves for samples which were supplied in sufficient quantity. All samples were milled and floated with water obtained from the local municipality, Rand Water Board.

The milled sample was transferred to a 2.5 ℓ float cell with approximately 2 ℓ of water (i.e. 33% solids) for flotation test work. A typically reagent suite of sodium isobutyl xanthate (SIBX, collector), KU5 (depressant) and Dow 200 (frother) was used. All reagents were made to 1% w/w standards using distilled water. Figure 1 illustrates the laboratory flowsheet used to characterize the ores.

In the rougher stage the pulp was conditioned with 150 g/t SIBX for 2 minutes followed by 30 g/t KU5 for 3 minutes and 20 g/t Dow 200 for 1 minute. The impeller speed used for conditioning was the same as that used for floating the material, i.e. 35 Hz or 1200 rpm. After conditioning the

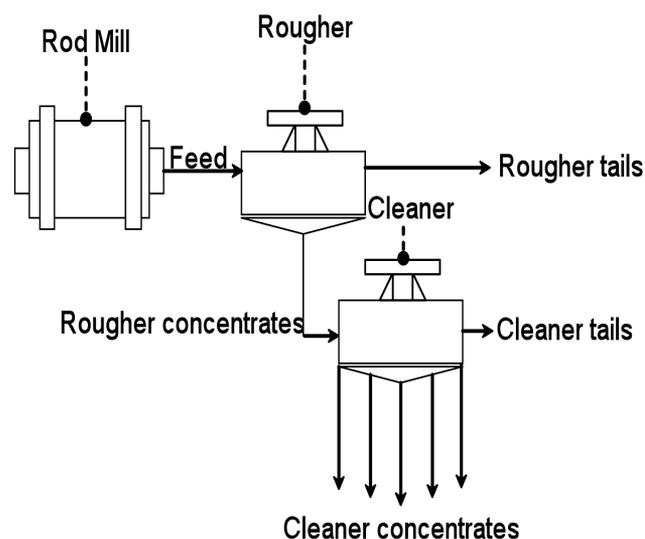


Figure 1. Laboratory flowsheet used to evaluate the metallurgical response of each ore

airflow rate was opened to 30% of the maximum rotameter airflow rate, which was 21 Nℓ/min, at a line pressure of 4.85 bar absolute. The line pressure was maintained using a pressure regulator and filtered with an in line air cleaner.

The rougher stage was floated for 20 minutes to produce a bulk concentrate; this was done in duplicate to generate sufficient mass for the cleaner stage. The concentrates were obtained by scrapping the froth every 15 seconds with two paddles. The paddles were used to scrape the froth from the back of the float cell and into the froth collector trays.

The duplicate bulk rougher concentrates were transferred into another 2.5 ℓ float cell for cleaning, no thickening of the concentrates were required. In the cleaner stage depressant and frother dosage were set as variable parameters. Preliminary development of the froth (with the introduction of air) gave an indication of the froth stability and if to stable, the air was switched off and more depressant was added. The test was started once the froth structure was satisfactory. The objective of adding an incremental amount of depressant and observing the effect on the froth structure was based on maximizing the selectivity between values and gangue so that differences (with respect to upgrading of values) in the various samples could be exposed.

Five sequential concentrates were collected from the cleaner stage at float times of 1, 3, 7, 20 and 30 minutes. All samples were subsequently filtered and dried prior to sample preparation for assay. The dried sample masses were recorded and the samples were pulverized in a steel bowl. In the case of the rougher tails a 200 g sub sample was taken and pulverized for assaying. The pulverized samples were assayed for major platinum group elements (PGEs); Pt, Pd with associated Au, 2PGE+Au. The rougher tails were assayed in duplicate for 2PGE+Au due to the high assay variance usually associated with the tails. The concentrates and the cleaner tails were assayed in singles for 2PGE+Au. All concentrates and tails were also assayed in singles for Cr₂O₃. Furthermore, the rougher tails was also sized at 75 µm to estimate the grind of the ore.

Application of kinetic models to data

The cumulative recovery of values was regressed to the modified Kelsall model, refer to Equation [1]. Application of the modified Kelsall model follows from the asymptotic

approach of the values to a maximum recovery after extensive cleaner flotation, indicative of recovery primarily by true flotation (particle–bubble attachment) and not entrainment, i.e. unselective recovery related to water recovery. The model has the maximum recovery (R_{max}) which is the sum of the fast floating fraction (R_{fast}) and the slow floating fraction, R_{slow} . In addition there are two rates; the fast floating rate (K_{fast}) and the slow floating rate, K_{slow} .

$$\begin{aligned} \text{Cumulative fraction floated (R)} &= \\ R_{fast} (1 - e^{-k_{fast}t}) + R_{slow} (1 - e^{-k_{slow}t}) & \\ \text{Where, maximum recovery (R}_{max}\text{)} &= \\ R_{fast} + R_{slow} & \end{aligned} \quad [1]$$

$$\begin{aligned} \text{Unfloatable fraction (R}_{unfloated}\text{)} &= \\ 100 - R_{max} & \end{aligned}$$

On the other hand, characterization of the Cr_2O_3 and mass recovery was done by regression to the two component Kelsall model, refer to Equation [2]. The use of an unfloatable fraction for these components could be logical (e.g. too coarse to be entrained or floated) but could not be justified for improvement of fit.

$$\begin{aligned} \text{Cumulative fraction floated (R)} &= \\ R_{fast} (1 - e^{-k_{fast}t}) + R_{slow} (1 - e^{-k_{slow}t}) & \end{aligned} \quad [2]$$

$$\begin{aligned} \text{Where, maximum recovery (R}_{max}\text{)} &= \\ R_{fast} + R_{slow} &= 100 \end{aligned}$$

Application of a cleaner index

When characterizing an ore, aside from observing the maximum recovery of values, it is also important to observe the ability of the ore to concentrate into a useful product. The grade–recovery curve provides this opportunity to observe the final grade after flotation and using this value to determine which samples ‘clean better’ than others. However, one must establish a priori for instance, all samples with final grade less than 100 g/t 2PGE+Au is considered as ‘poor cleaning’ samples while all ores with final grade above 100 g/t 2PGE+Au are considered to be ‘better cleaning’ samples. An alternative to this may be the upgrade ratio which in this instance will be the ratio between the cleaner grade and the head grade of the ore. Additionally, the ratio between the fast floating value and gangue proportion can also be used to characterize the

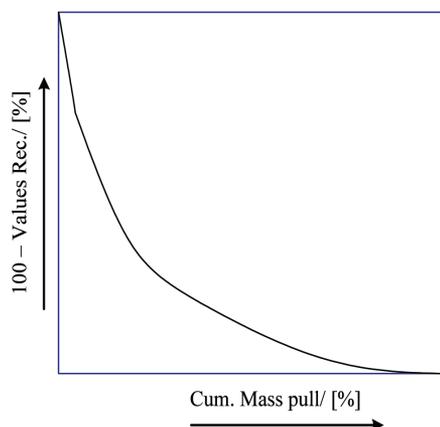


Figure 2. Illustration of the relationship between values not recovered and mass pull

upgrading of the ore as point out by Hay¹.

A more rigorous approach for understanding the ability of a sample to concentrate into a useful product may be that demonstrated by Mohanty². The procedure was developed for characterizing the cleaning of coal samples and therefore had to be tailored so that it could be applied to UG2 processing and possibly other ores. The modification involved plotting the values not recovered (y , ordinate) against cumulative mass pull (x , coordinate). Two boundary conditions were added to the experimental data; these conditions were based intuitively on the relationship between mass pull and recovery. The first end point recognizes at 100% mass pull the recovery of values is 100% and the second end point recognizes at 0% mass pull the recovery of values is 0%. Figure 2 illustrates the plot of y against x ; the two boundary conditions are also included.

In order to develop a measure of the ability of an ore to ‘clean’ the relationship between y and x must be modelled. Mohanty applied an exponential relationship to y and x . The relationship was resolved into a fraction representing fast floating material and a fraction representing slow floating material because of the heterogeneous characteristics of the feed-however, the final equation had five fitting parameters. Therefore a simplification of the model was made; in addition, the model was linked to the information obtained from the modified Kelsall model, i.e. the fast floating values fraction recovered.

The relationship between y and x was modelled has an exponential relationship. Two rate constants and floatable fractions were applied, one for fast floating material and the other for slow floating material. The fraction of fast floating material was estimated from the modified Kelsall model, Equation [1]. The slow floating fraction was taken has the difference between the total values and the fast floating fraction, refer to Equation [3].

$$\begin{aligned} \text{Cumulative fraction of values not recovered (y)} &= \\ = y_{fast} (e^{-k_{fast}x}) + y_{slow} (e^{-k_{slow}x}) & \end{aligned} \quad [3]$$

$$\text{For values, } y_{fast} = R_{fast} \text{ (from Equation 1)}$$

$$y_{slow} = 100 - y_{fast}$$

Equation [3] was used to determine the area under the curve this was done by integrating Equation [3] from a cumulative mass pull of 0% to a cumulative mass pull of

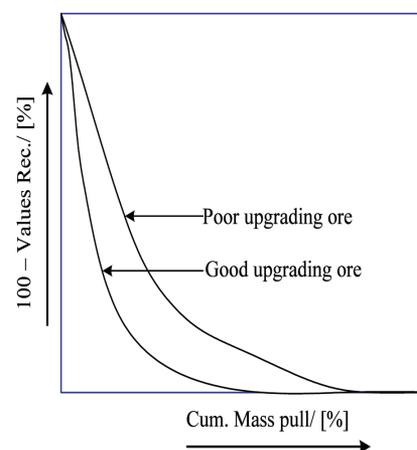


Figure 3. Hypothetical illustration of the relationship between recovery and mass pull for a poor and good upgrading ore

100%. For ores that have poor upgrading, integration and solving of Equation [3] returns a high area under the curve because these ores have high mass pulls and low recoveries. On the other hand, ores that have good upgrading have high recoveries which are obtained with a smaller mass pull. Figure 3 gives a hypothetical illustration of the difference between a poor and good upgrading ore.

Results and discussion

Discussion on test program

Figure 1 illustrates the laboratory flowsheet used to characterize the different UG-2 samples. The premise behind the use of this flowsheet is:

- It is inexpensive and can be used to evaluate the variability of the ore characteristic being determined
- It is straightforward to set up and expedites data output
- Can incorporate small sample masses, < 4 kg
- Allows for depressant variation in the cleaner stage aimed at improving selectivity and thus understanding the ability of the ore to upgrade
- Use of a rod mill over a ball mill can be justified by the production of a milled sample with a narrow size distribution and less generation of fines.

It may be argued that the laboratory flowsheet influences the metallurgical response of the ores. In other words, the fixed milling time, reagent suite and dosages, rougher and cleaner operability may be biased to some ores and not others. Nevertheless the premise of a variability study is to highlight how variable samples are to a fixed laboratory procedure. For example, the fixed milling time (based on achieving a P_{80} of 75 μm) showed samples that were more competent (i.e. lower percentage of the material passing 75 μm) and ores that were softer, i.e. higher percentage of the material passing 75 μm .

For many variability studies the emphasis is usually on the maximum recovery of values. Furthermore, very little information is obtained on the upgrading characteristic of the sample. The upgrading characteristic depends on the amount of gangue that reports to the concentrate with the values. Therefore in this study it was decided that depressant dosage in the cleaner stage will be the only operational parameter that will be varied. The depressant dosage was varied to maximize selectively between values and gangue and provides further information on the upgrading characteristic of the ore.

The data obtained from each characterization test was used to obtain the metallurgical parameters from Equation [1], [2] and [3]. The cumulative recovery of values, chromite and mass was regressed to the Kelsall models to obtain the fractions R_{fast} , R_{slow} and R_{max} (for values only); and K_{fast} and K_{slow} . In addition the cumulative values not recovered and cumulative mass pull data were used to obtain the parameters for Equation [3].

A cleaner index was obtained by integrating Equation [3] to get the area under the curve. Mohanty² normalized this area to the area obtained by ideal separation. Ideal separation is the area covered by the triangle linking the boundary points and the origin. Though for complex low sulphide ores like UG2 the overall mass pull especially from the cleaner stage is small, typically less than 4% i.e. for a 2 kg feed only 80 g reports to the concentrate. Since the cumulative mass pull is small the area covered by the curve is generally smaller than that for a high mass pull ore like base metal ores. The affect of relating the curve area to the area covered for ideal separation results in the data

being skewed, i.e. almost all the cleaner index values were between 90 and 100%. To circumvent this, the worst upgrading ore characterized by a low values recovery and high mass pull (i.e. large area) was used as the benchmark and all ores were compared relative to this ore. Equation [4] shows the relative cleaner index (RCI) used to characterize the upgrading characteristic of the samples. The benchmark ore therefore has a relative cleaner index of 0%. The closer RCI is to 100% the better is the upgrading characteristic of the sample and the closer RCI is to 0% the poorer is the upgrading characteristic of the ore.

$$RCI = \frac{\text{Area of worst ore} - \text{Area of ore being evaluated}}{\text{Area of worst ore}} \quad [4]$$

Metallurgical variability

A linear relationship exists between the rougher and cleaner 2PGE+Au recovery, Figure 4. Furthermore, the relationship implies that there is little drop out of values in the cleaner stage, i.e. high cleaner efficiencies. The application of a cleaner stage in this test work provides the benefit of maximizing values upgrading (through depressant control) without incurring significant loss of values obtained from the rougher stage.

Figure 5 illustrates the variability with respect to the maximum recovery of values obtained from regression of Equation [1] to the cleaner rate data. Eighteen of the samples had a maximum recovery greater than 80% 2PGE+Au and twenty-seven had a maximum recovery less than 80% 2PGE+Au.

Figure 6 illustrates the variability with respect to the relative cleaner index for values and was obtained from regression of Equation [3] to the cumulative mass recovery and values not recovered with time. The sample at a relative cleaner index of zero is the benchmark sample. Only five of the ores tested had a RCI greater than 90%.

To address if the maximum recovery is related to grind the samples were grouped into four classes based on the maximum recovery of values, i.e. < 66%, > 66% < 80%, > 80% < 87% and > 87% < 100%. Figure 7 illustrates the effect of grind on the maximum recovery for each class. It

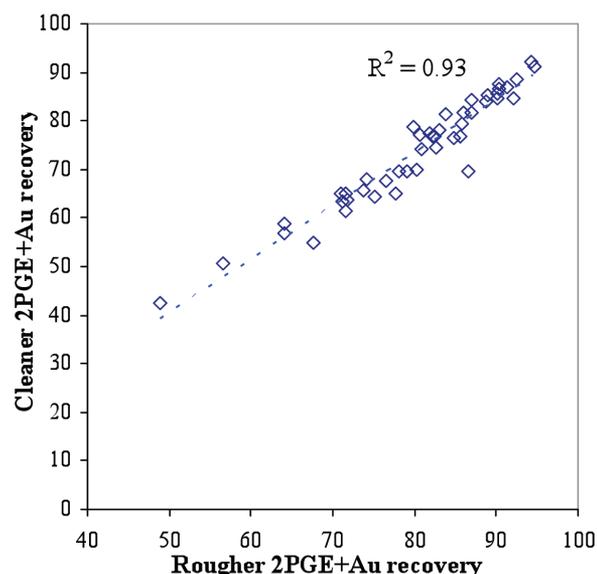


Figure 4. Influence of rougher 2PGE+Au recovery on the cleaner 2PGE+Au recovery

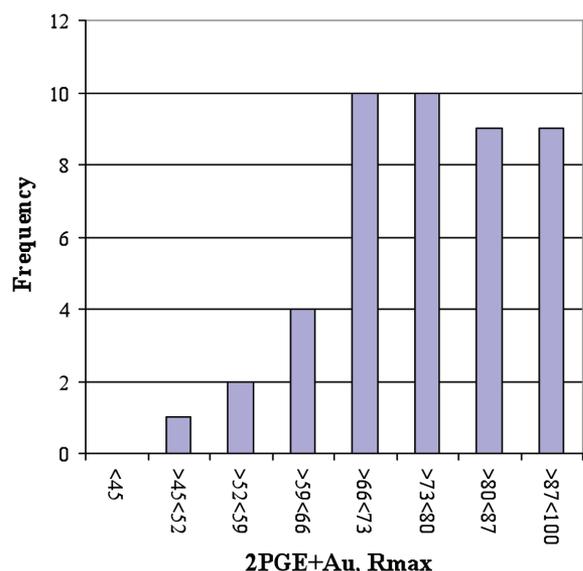


Figure 5. Variability of the maximum values recovery for the samples

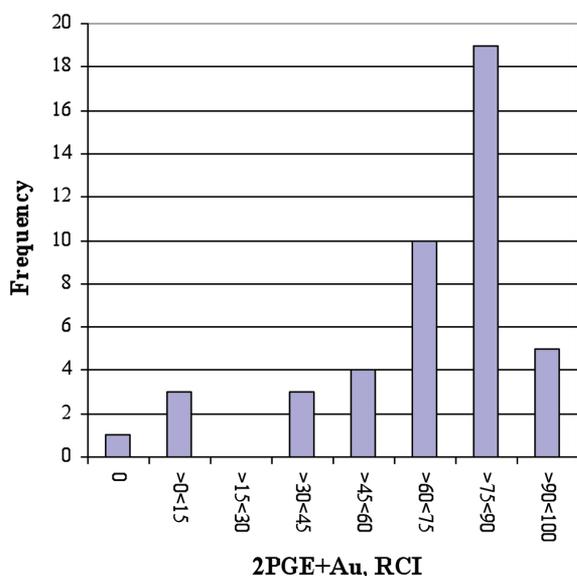


Figure 6. Variability of relative cleaner index for the samples

is evident that for some samples, having a grind greater than the target grind (80% -75 μm); the maximum values recovery is lower than 80%. On the other hand, some samples had a maximum recovery greater than 80% even though the grind was coarser than the target grind, i.e. less than 80% -75 μm . Therefore grind can not be the only factor that influences recovery.

Figure 8 illustrates the effect of the feed 2PGE+Au assay on the maximum recovery. Samples having a high assay (for instance, > 4 g/t) do not necessarily have better recovery of values. Furthermore, some samples having a lower feed assay gave better values recovery than samples having a high feed grade.

Further attempts were made to relate the metallurgical parameters (R_{max} and RCI) to other assay parameters such (Cr, Si, Al, Fe, etc) and ratios of these assays (Cr/Si, Cr/Al, etc.) but no conclusive relationship could be found. The

recovery (R_{max}) and upgrading (RCI) characteristics for UG2 samples may be a complex relationship among a number of variables.

When processing UG2 ores aside from targeting a high recovery and grade for the values, consideration must be given to understanding the recovery of chromite into the final product. Chromite can be recovered by flotation (bubble-particle attachment) or entrainment, although the latter may be the primary mode of recovery. High chromite recovery can result in a chromite grade above 2% in the final product which can incur smelter penalties. Therefore Figure 9 was used to highlight those ores which may have an issue about chromite in the final concentrate. The fast floating fraction for chromite (assumed to be recovered by true flotation) was plotted against the maximum values recovery for each sample. It was assumed that all ores with a fast floating chromite fraction greater than 0.28% may have an issue about chromite recovery to the final product.

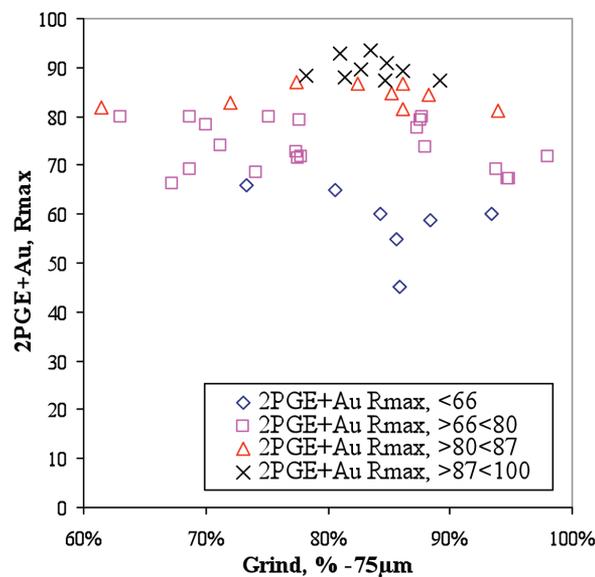


Figure 7. Effect of grind on the maximum values recovery

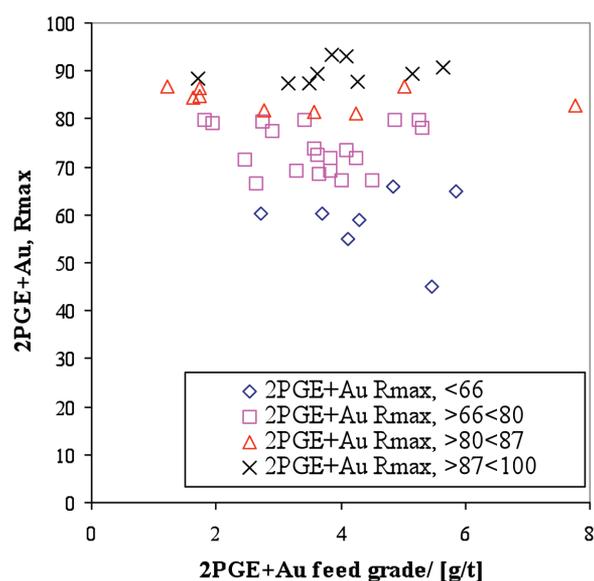


Figure 8. Effect of feed grade on maximum values recovery

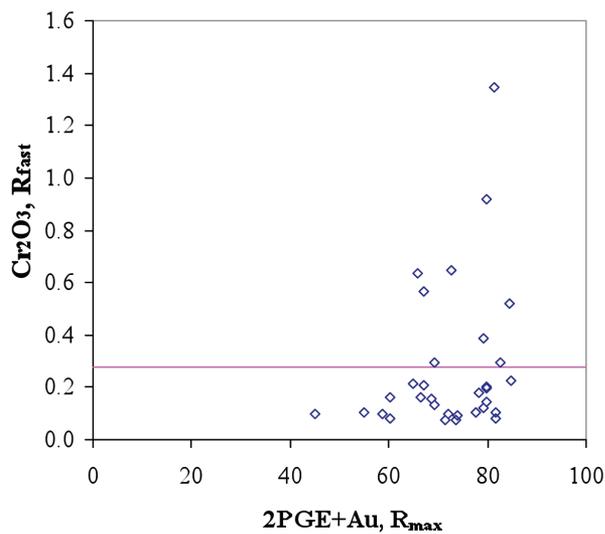


Figure 9. Relationship between fast floating chromite and values maximum recovery for samples characterised

The cut-off recovery for fast floating chromite was determined by using the average mass pull from the cleaner rates (3.5%) together with the average chromite head grade (25.1%) for all ores. Based on the feed mass for each characterization test (2 kg) the fast floating chromite recovery that results in the concentrate grading at 2% was calculated; this was 0.28%. From Figure 9 it follows that although most of the UG2 ores have little floatable chromite there does exist a group of samples that has a high floatable chromite component. It is postulated that this high floatable fraction may be due to talc rimming of the chromite, association with base metal sulphides or other floatable material.

From this study it follows that the metallurgical variability of UG2 ores on a macro scale is significant. The metallurgical variability must be carefully understood so that the flotation circuit proposed is robust enough to handle the variability within the deposit. The problem often encountered is that not enough samples are made available during the feasibility study and this affects the scope of the work necessary to obtain the batch rate coefficients for plant design. Furthermore, the laboratory batch rate coefficients used for simulating various flotation configurations is not fully understood with respect to chemical and operational factors and this results in poor circuit design.

Conclusions and recommendations

- A standard laboratory flowsheet was used to characterize forty five UG2 samples received from various platinum operations across the Bushveld
- The maximum recovery and relative cleaner index varied considerably for the samples evaluated
- There was no conclusive relationship that grind affects the maximum recovery, some ores had a coarser grind (i.e. <80% -75 μm) but gave high values recovery, i.e. > 80%. Other ores had a finer grind (>80% -75 μm) but gave lower values recovery, i.e. <80%
- No relationship could be found on the influence of the 2PGE+Au feed assay and the maximum values recovery. Furthermore, no significant relationship could be found between the other head assay parameters and the metallurgical parameters, R_{max} and RCI
- This study highlights that there is significant metallurgical variability for the UG2 ores sourced from across the Bushveld. This metallurgical variability must be understood during the feasibility phase so that the flotation circuit proposed is a compromise for the deposit and not biased to a few samples that cover a limit area of the deposit

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