Sampling of run-of-mine mill feed— A practical approach

by K.I. Afewu and G.O. Lewis*

Synopsis

A modified approach to the implementation of Gy's sampling theory is discussed and used to design a procedure for sampling crushed gold ore feed to metallurgical process plants. The approach involves calibration of certain ore-specific parameters. These parameters were calibrated for two different crushed gold ores *en route* to grinding mills in process plants. The model predictions were compared with the predictions from Gy's original model. Far smaller masses of samples are required by the modified model than by Gy's model at the same relative variance. The representativeness of the sampling procedure if normally distributed errors are assumed was also calculated. These procedures and evaluations lead to significant savings on cost of handling and preparation of the samples and to a quantifiable degree of confidence in the analytical results for the process plant.

Introduction

Sampling and analysis of broken ores are the first critical elements in project evaluation and continuous monitoring of any geochemical exploration, ore reserve estimation, mine planning, and flowsheet development and optimization of a process plant. The analysis of a sample is quite straightforward and generates a result that has unknown errors.

Sampling is the operation of removing a part, convenient in size for testing from a whole which is of much greater bulk, in such a way that the proportion and distribution of the quality tested (e.g. the specfic gravity, metal content , recoverabilty) are the same in both the whole and the part removed (sample). The sample must be completely representative of the whole as regards all aspects save bulk[1].

A fundamental aspect of nearly all mineral deposits is the inherent variability of the mineral assemblage and composition. The major problem with sampling broken ores lies in ensuring that the sample being analysed is representative of the bulk (or its parent population) and is particularly acute in the analysis for minerals which occur in trace amounts in ores (for example, gold). This problem is exacerbated by the size of the laboratory specimen ultimately analysed, which is much smaller than the initial field sample.

An essential condition of any sample is that it be representative of the greater bulk. It thus becomes necessary to establish a correlation between the properties of the population of the sample mass and the maximum particle size of the sample, so as to ensure that any sample taken is representative of the bulk. A number of approaches have been proposed to address these problems. The most notable one is the work of Gy[2]. Most practitioners have used his model for gold ores, though without much fulfilment in the results. The reasons for this are two-fold[3]. Firstly, because the presentation of the model in textbooks and journals is always complex, and secondly, misapplication of the model by users. These have led to impractical results, frustrations and abandonment of the use of the model. Recently, Francois-Bongarcon[4] has modified Gy's model. More realistic results are being obtained as a result of the modification for mineral exploration and geostatistical evaluation work[3,4].

The importance of sampling to the mineral process engineer cannot be overemphasised. The reasons for sampling in a mineral processing plant are[5]:

➤ to acquire information about the ore entering a treatment plant to ensure valid metallurgical accounting,

➤ to inspect its conditions at selected points during its progress through the plant for comparisons between the optimum requirements for efficient treatment and those actually existing, and

➤ to disclose recovery and losses in order to improve the former and reduce the latter.

The purpose of this paper is to highlight

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the successful application of this modified Gy's model to design a means of establishing the minimum sample size and the various stages of comminution required for broken ores (particularly gold ores) within a specified precision range, for mineral processing plant feed.

Sampling theory

In order to determine adequate sample size, the precision expected in the analysis of the desired element (in this case gold) must be quantitatively defined.

Sampling errors have been classified into four main groups: fundamental, segregation and grouping, integration and operating errors[3]. The fundamental error is due to irregular distribution of ore values in the particles of broken ore. Segregation or Grouping error arises as result of lack of thorough mixing and taking of samples. Integration error results from sampling of flowing ore. Operating error is due to faulty design or operation of the sampling equipment, or to the negligence or incompetence of personnel.

Gy derived a model comprising the precision of sampling, to calculate the probability that particles of mineral, randomly distributed in a host matrix with a specific top size, would be collected in a sample of specific mass. This model expresses the relative variance of the fundamental error, σ^2, of sampling as[2]:

$$\sigma^2 = \left(\frac{1}{M_S} - \frac{1}{M_L} \right) fgcld_n^3 \qquad [1]$$

where :

d_n is the nominal size of fragments in the sample. This is the maximum particle size in the lot to be sampled. In practice, d_n is taken as the mesh size which retains 5 per cent of the lot being sampled, and is measured in centimetres.

f is the shape factor. It is an index varying between 0 and 1. In practice, most values are between 0.2 and 0.5, the actual value depending on the shape of the minerals and the degree of comminution. For most ores, a value of 0.5 is used.

g is the granulometric factor (or the grain size distribution factor). It takes values between 0 and 1; low values of g denoting a large range of particle sizes and high value denoting a narrow range (g =1 denotes all particles are of identical size). For most ores, g = 0.25.

c is the mineralogical composition factor and is given by:

$$c = \frac{1-a}{a} \left[(1-a)\rho_m + a\rho_g \right] \qquad [2]$$

where :

a is the decimal proportion of the valuable mineral
ρ_m is the density of the valuable mineral
ρ_g is the density of the gangue mineral. For low grade ores (for example, gold ores),

$$c \approx \frac{\rho_m}{t} \qquad [3]$$

where t is the grade of the ore, and should be expressed in grams of gold per gram of ore, g/g,

M_S is the mass of sample measured in grams.
M_L is the mass of material from which sample is taken, measured in grams.

l is a liberation factor of the mineralogic constituents.

For unliberated particles, Gy assumed:

$$l = \sqrt{\frac{d_0}{d_n}} \qquad [4]$$

where d_0 is the liberation size of the mineral particles, i.e. the maximum particle diameter which ensures complete liberation of the mineral, measured in centimetres.

Since $M_S \ll M_L$, Equation [1] reduces to Equation [5]:

$$\sigma^2 = \frac{fgcld_n^3}{M_S} \qquad [5]$$

The results obtained for most gold ores from using Gy's model Equation [5] have not been meaningful in practice, as the results are sensitive to the liberation factor[3]. This has led to many practitioners abandoning Gy's model. An example to illustrate the impractical minimum sample masses for gold ores by the application of Gy's model is given later.

Francois-Bongarcon's modification treats Equation [4] in the more general form, and Equation [5] may therefore be written as:

$$\sigma^2 = \frac{Kd_n^\alpha}{M_S} \qquad [6]$$

where α is a parameter depending on the type of mineralization of the deposit, and which can be calibrated, and K is given by:

$$K = cfgd_0^{3-\alpha} \qquad [7]$$

For a particular ore, c, f, g, d_0 are constants. K depends on microscopic properties and the grade of the mineral. The values of K and α are constant throughout the sample preparation stages[4].

Taking logarithms of Equation (6), Equation (8) is obtained:

$$\ln\left(\sigma^2 M_S\right) = \alpha \ln d_n + \ln K \qquad [8]$$

Plotting $\ln(\sigma^2 M_S)$ against $ln(d_n)$, a straight line will be obtained, with slope α, and intercept on the $\ln(\sigma^2 M_S)$ axis ln K. Hence α and K can be calibrated for a particular ore and d_0 can be computed.

Nomograms

Another form of Equation [8] is:

$$\ln \sigma^2 = -\ln M_s + \alpha \ln d_n + \ln K \qquad [9]$$

For a particular ore, at a known degree of comminution, the sum of the last two terms in Equation [9] is a constant, Z. Equation [9] then becomes

$$\ln \sigma^2 = -\ln M_s + Z \qquad [10]$$

Plotting σ^2 against M_S on a log-log scale, leads to parallel lines, with slopes of -1 and gives the plot of the nomogram for a particular ore. These charts based on the calibrated parameters of deposits, constitute useful tools for the design, assessment, control and improvement of the sampling processes.

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Experimental

Sample tree

A sample tree experiment involves splitting at different top sizes, followed by comminution and assaying[5]. Figure 1 shows the sample tree flowsheet. Two ores were evaluated via this sampling technique. The first sample was a low-grade gold ore (about 5 g/t gold) mill feed sample from a South African gold mine. The particle size was 95% -12 mm. The second sample consisted of boulders of high-grade gold ore (about 60 g/t) that were crushed to yield a 95% -12 mm product. The masses of each of the two samples were between 20 and 25 kg.

Series 1

Each sample was split using a ten-cup spinning riffle sampler. The contents of each of the ten cups were further separated, yielding a hundred samples. Alternate cups were combined to produce 50 samples. Thirty-five products were chosen and the remaining fifteen rejected but not discarded. Each of the 35 samples was pulverised to -75μm and further split into 10 samples. One of the samples from each batch became an aliquot for analysis by fire assay. The other 9 samples were rejected and discarded.

Series 2

The fifteen samples initially rejected in Series 1 constituted the sample for the Series 2. They were combined, crushed to 95% -0.85 mm, and split into ten samples. Each of the ten was further split into ten increments. Alternate increments of the resulting 100 were combined to yield 50 samples. Thirty-five samples were randomly selected from the 50. The

remaining 15 were rejected but not discarded. Each of the 35 chosen samples was pulverised to-75μm and further split into ten samples. Three of the ten samples from each batch were combined to form an aliquot for analysis by fire assay. The other seven samples were rejected and discarded.

Series 3

The 15 rejected samples from Series 2 were combined and pulverized to -75μm and split into ten samples. Each of the ten was split into a further ten. Alternate products were combined yielding 50 samples. Each of the 50 became an aliquot for analysis by fire assay.

Results and discussion

The results are summarized in Table I. As can be seen, the relative standard deviation decreased from Series 1 to 3. This is the expected trend, as a higher degree of variability of the assay values is expected from sampling carried out at larger particle sizes. The analytical variance of the method[6] is 4 per cent. The computed variances have to be corrected for, by deducting the analytical variance, as the latter is always included in the variance estimates calculated from experimental work. The single-stage variances of each of the three series were determined, starting from Series 3. From linear regression analysis, for the low-grade ore, α is 1.01 and K is 94.0 and for the high-grade ore, α is 1.13 and K is 15.7. The values of α are in the range of current research values of between 0.76 and 1.15 (reported by Assibey-Bonsu[3]) and close to ±1.5 (Francois-Bongarcon[4]) for some gold ores. The calibration plots are shown in Figures 2 and 3 for the low- and the high-grade ores respectively.
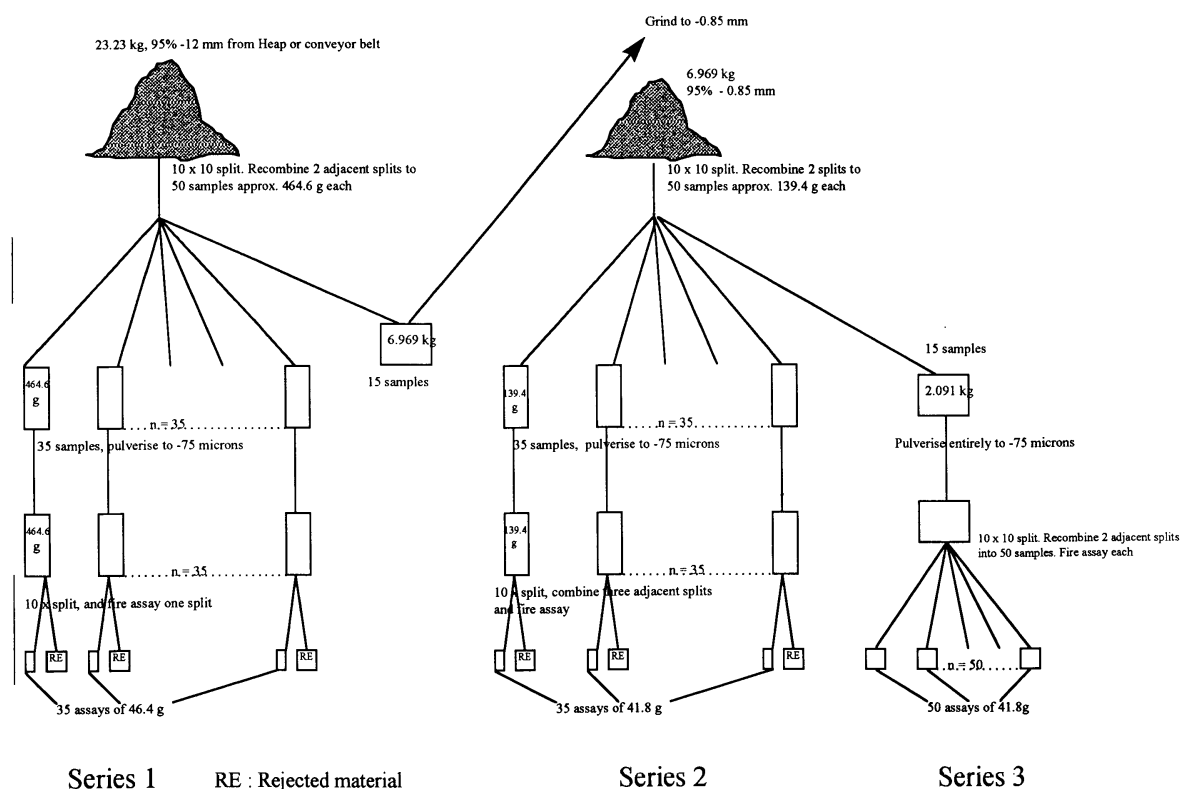

Figure 1—Sample tree experiment flowsheet

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Parameters determined from analytical results

	Low-grade ore			High-grade ore		
	Series 1	Series 2	Series 3	Series 1	Series 2	Series 3
Mean	4.44	6.03	5.10	62.58	62.59	63.53
Rel. std. deviation	0.6838	0.2164	0.1803	0.1957	0.1128	0.0534
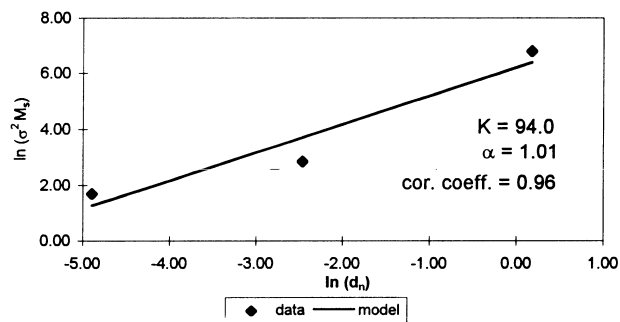
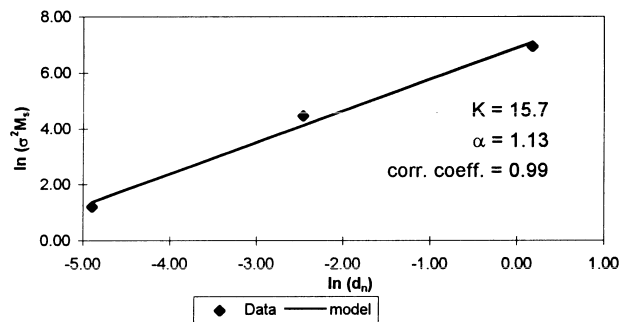**Figure 2—Fitting a model to the fundamental sampling error (low-grade ore)**

K = 94.0
α = 1.01
cor. coeff. = 0.96

Figure 3—Fitting a model to the fundamental sampling error (high-grade ore)

K = 15.7
α = 1.13
corr. coeff. = 0.99

The calibration functions for the low-grade and the high-grade ores are given by Equations 11 and 12 respectively:

$$\sigma^2 = \frac{94d_n^{1.01}}{M_S} \qquad [11]$$

$$\sigma^2 = \frac{15.7d_n^{1.13}}{M_S} \qquad [12]$$

Equations [11] and [12] were used to compute the parameters in Tables II and III, and then to plot the nomograms (Figures 4 and 5) to determine the sampling protocol that would yield the optimum final sample variance. The various sampling steps are represented by moving from the right (marked A) to the left (marked E). The crushing and pulverizing stages would not contribute to the variance and they are represented by the vertical segments (A to B)

Table II

Recommended sampling technique for the low-grade ore

Size (cm)	Mass (g)	Rel. variance σ^2	% Rel. std deviation σ
1.2	12000	0.00940	9.70
0.1	12000	0.00077	2.77
0.1	1000	0.00917	9.58
0.0075	1000	0.00067	2.59
0.0075	70	0.00959	9.80

Table III

Recommended sampling technique for the high-grade ore

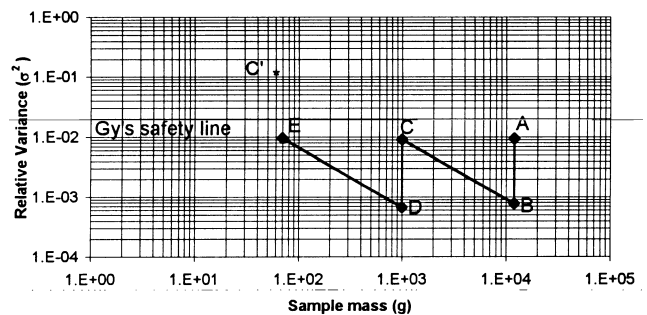| Size (cm)	Mass (g)	Rel. variance σ^2	% Rel. std deviation σ
1.2	2000	0.00960	9.80
0.1	2000	0.00058	2.40
0.1	200	0.00580	7.60
0.0075	200	0.00031	1.77
0.0075	20	0.00312	5.58

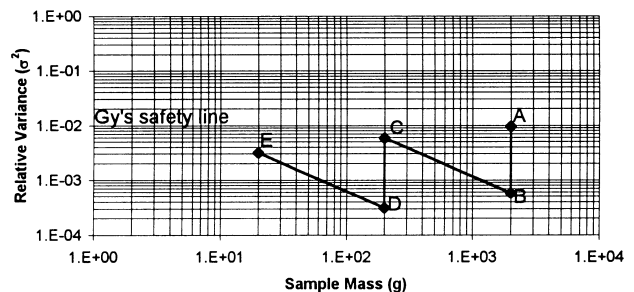**Figure 4—Sampling protocol chart (low-grade ore)**

Figure 5—Sampling protocol chart (high-grade ore)

and (C to D) respectively. The proposed nomogram (Figure 4) may be interpreted as:

Position A: A sample mass of 12 kg at 95% -12 mm from the mill feed should be taken

Position B: The 12 kg sample should be crushed in a

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laboratory jaw crusher to 95% -1 mm

Position C: A sample of 1 kg should be taken from the 12 kg sample at particle size of 95% -1 mm

Position D: The 1 kg sample should be pulverized to 95% -75 microns, and

Position E: From the 1 kg sample at -75 microns, an aliquot of 70g should be taken for analysis by fire assay.

From the nomogram (Figure 4), the incremental variances can be numerically calculated as:

$$\sigma^2 = \sigma^2\,[A] + (\sigma^2\,[C] - \sigma^2[B] + (\sigma^2[E] - \sigma^2\,[D])$$
$$= 0.0094 + (0.00917 - 0.000917) + (0.00959 - 0.00067) = 0.026573$$

corresponding to a relative standard deviation of 16.3 per cent. Note, that if the assay laboratory gets involved with the work only after the initial 12 kg sample has been taken, then it is responsible for a contribution to the variance corresponding to a sampling relative standard deviation of only 13.1 per cent. (Furthermore, assuming the fundamental errors follow a normal distribution, the final sampling representativeness is ± 32.0 per cent (for the entire process) and ± 25.7 per cent (assay laboratory's sampling) at the 95 per cent confidence level (equivalent to 1.96 x standard deviation.)

Following a similar procedure for the low-grade ore as stated above, the relative variance of the recommended sampling technique for the high-grade ore is 0.0176, corresponding to a relative standard deviation of 13.3 per cent. The assay laboratory is responsible for a sampling relative standard deviation of 9.0 per cent. Assuming a normal distribution for the final sampling representativeness is ± 26.0 per cent (for the entire process) and ± 17.6 per cent (assay laboratory's sampling) at 95 per cent confidence level.

The variances of the sampling operations comprising mass reduction at constant ore particle sizes were limited to a relative standard deviation of 10 per cent at each particle size considered. This relates to Gy's safety line at a relative variance value of 10^{-2} (or relative standard deviation of 10 per cent) as shown in Figure 3. According to Gy, for relative variance values above this value (10^{-2}), the sampling operation may involve unacceptable errors[4]. This implies the precision may get out of control, and possibly be out of the validity domain of the model. If the assay laboratory decides to eliminate the intermediate grinding stage CD in Figure 4, (thus taking the 75 g aliquot from the 12 kg sample and then pulverizing it for assay), the overall incremental variances will then be given by:

$$\sigma^2 = \sigma^2\,[A] + (\sigma^2\,[C'] - \sigma^2[B])$$
$$= 0.0094 + (0.13 - 0.000917) = 0.13848.$$

This leads to an overall high relative standard deviation of 37.2 per cent. As can be seen from Figure 3, the point C is above the Gy's safety line of relative standard deviation of 10 per cent (the prohibited region for valid results).

However, estimating the minimum mass of sample for the low-grade gold ore by Gy's original model, with $d_n = 12$ cm and gold grade of 5.19 g/t as used above, the mineralogical factor is given by:

c = density/grade = $19/(5.19 \times 10^{-6})$ = 3.66×10^6 g/cm3.

For a gold grain top size of 75μm, Gy's liberation factor, l is given by:

$$l = \sqrt{\frac{d_0}{d_n}} = \sqrt{\frac{7.5\mathrm{x}10^{-3}}{12}} = 2.5\mathrm{x}10^{-2}.$$

For a relative precision of 10 per cent, the variance is then 0.01, using the Gy's equation with g = 0.25 f = 0.5, and M_s = 1976.4 t.

For the high-grade ore used in this experiment, $d_n = 12$ cm, the gold grade is 62.9 g/t, and the mineralogical factor is given by:

c = density/grade = $19/(62.9 \times 10^{-6})$ = 0.302×10^6 g/cm3.

For a gold grain top size of 75 μm, the minimum mass of the sample predicted by Gy's model at a relative variance of 10 per cent is 163.1 t.

These values are practically unacceptable for a mine producing between 1000 to 10 000 t per shift.

The prediction of such high masses has been the cause of most researchers abandoning Gy's original model[3]. Bartlett and Hawkins[7] documented similar problems on applying this model to gold deposits in South Africa. Though they[7] did not provide any reasons for the unacceptably high mass, they pointed out that far smaller sample masses are in fact required.

Conclusions

Francois-Bongarcon's modification of Gy's model was successfully used to calibrate sampling parameters for both low-grade and high-grade gold ores (mill feed) from South Africa. Far smaller sample masses are required than predicted by Gy's original model at the same relative variance. For the low-grade ore used in this experiment, the modified model predicted 12 kg as compared to 1976.4 t obtained by Gy's model. In the case of the high-grade ore, the modified model predicted 2 kg while Gy's model required 163.1 t. As will be expected, a smaller quantity of sample of the high grade ore is required than that required of the lower grade ore to meet the requirements of minimum mass of sample. Sampling protocol charts based on the calibrated values recommending various stages of comminution and sampling have been drawn for the two ores. The recommended procedure is well within the Gy's safety limit of 10 per cent relative variance at each stage of sampling. Using the recommended procedure, the representativeness of the samples at 95 per cent confidence level was calculated, assuming normally distributed sampling errors. These procedures and evaluations lead to significant savings on cost of handling and preparation of the samples and to a quantifiable degree of confidence in the analytical results for the process plant.

Though a lot of attention is focused on the minimum mass of sample, over-sampling (i.e. taking samples of excessive masses) should be avoided as this will lead to problems in handling and preparation, for example, high costs of transportation and drying.

It should be pointed out that the calibration parameters α and K may differ from time to time for the plant feed. Sampling and testing should, therefore, be carried out frequently on the feed mix to a process plant so that any variations in the ore mix can be detected and the necessary

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adjustments made. Sources of variation in the feed mix may arise from ore-body inconsistencies, zonal variations, feed variation from mining plan or methods and feed variation from different reefs in response to economic conditions of the product(s)[8].

Adherence to sound sampling procedure cannot be overemphasized, and the direct supervision by the engineer who is responsible for the testing programme is strongly recommended[8].

Acknowledgements

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Winning posters at Minerals Processing '98*
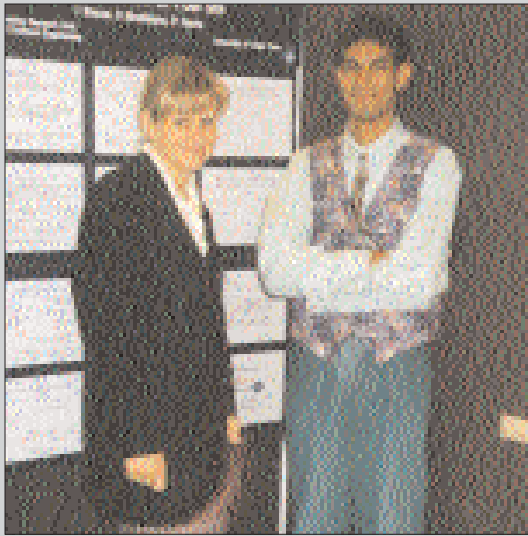
Minerals Processing '98, the 17th Conference in the series organized by the Western Cape branch of the South African Institute of Mining & Metallurgy was held at the Arthur's Seat Hotel in Cape Town from 6–7 August.

UCT Chemical Engineering masters student, Craig Beautement, celebrates his winning poster. Craig's poster won the 'best design' category at the recent SAIMM conference held in Cape Town. Craig's project covers Investigating possible treatment methods for secondary lead slag. His project is part of a broader focus on recovery of valuable products from effluents and wastes run by the Environmental Process Engineering research group. His supervisor is Dr Alison Lewis.

The category of 'best content' was won by UCT Chemical Engineering masters student, Imraan Bacus. His masters programme is supported by BOC Bulk Gases and his work investigates an aspect of the role of gases in complex sulphide mineral flotation. This project is within the scope of the research programme investigating chemical interactions in flotation that is being conducted by the Mineral Processing Research Unit. This project is supervised by Dee Bradshaw and Peter Harris. ◆

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