# PIT-TO-PLANT OPTIMISATION AT MORILA GOLD MINE

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## ABSTRACT

*AngloGold, the operating partner in a joint venture with Randgold Resources and the Government of Mali, own and operate the Morila Gold Mine. The open pit gold mining operation is located in Southern Mali, approximately 280 kilometres by road south-east of Bamako, the capital of Mali, West Africa.*

*The mined ore is processed through a conventional semi-autogenous grinding and ball milling, including a recycle crusher, (SABC) comminution and carbon-in-leach (CIL) circuit together with a gravity gold recovery step with final residue reporting to a tailings storage facility by way of cyclone deposition.*

*Pit-to-Plant (p2p) is a concept incorporating the various disciplines of geology, mining, metallurgy and engineering, to optimise the entire comminution circuit. At Morila, comminution commences in the pit with the fragmentation of ore through blasting and continues at the metallurgical facility with primary crushing, SAG and ball milling including a recycle pebble crusher.*

*The primary objective of the p2p programme is to increase mill throughput, but it must be stressed that this must not be at the expense of grade control, thus it is imperative that ore body dilution be minimised.*

*This paper takes the reader through the various project phases, including the cost of the p2p project at Morila where the mill throughput was substantially increased from 365 tph to approximately 400 tph (approximately 10% increase in throughput) with the aid of digital, online size analysis.*

*It is clear that the mill feed size distribution is a major contributing factor influencing mill throughput. There is a greater percentage of finer material being fed to the mills during periods of increased tonnage. Increased focus on blasting parameters, not so much for the reduction of top size material but more for the increase in the generation of fines, termed the "fines envelope" has been identified as being paramount for the increase in mill throughput.*

*The online, digital size analysis has made it possible to monitor and control the feed size through each phase of blasting, primary crushing and stockpile apron feeder ratio control. With the installation of the cameras at the primary crushing tip, more detailed size distributions, per blast, are now available for interpretation. With the information gained using the "Spilt-Online®" system as a tool, Morila is able to make significant progress in quantifying the effect of changing the various blasting parameters for increasing the production of "finer" material thus enhancing mill throughput.*

*Keywords: Morila, Mine-to-Mill ®, Pit-to-Plant, Optimisation, Blasting, Fragmentation, Quality control, Digital online size analysis, Split-Online®, Fines envelope.*

## 1 INTRODUCTION

The primary objective of the optimisation programme is to maximise mill throughput and the greatest influence on mill throughput can be realised through improved fragmentation of the blasted ore within the pit. One needs to be able to measure "what" is being delivered to the primary crusher to better understand the size distribution of the blasted material and monitor the effects of changes in blasting and mining parameters that affect fragmentation and consequently ultimately affect the mill throughput.

Three key areas were identified as being fundamentally important when motivating the p2p optimisation programme and the areas have been listed with their associated aims in order of priority (Steele, 2002).

- *Dump truck payload monitoring:* Feedback of this information to the drill & blast team will facilitate improved control in blast fragmentation and through this focus, major improvements in throughput will be realised as a result thereof.
- *Crusher gap control*: An overall benefit in throughput to be realised through improved control of "top size" material.

- *Mill feed composition control:* Reconstituting the mill feed size through blending post stockpiling, will undoubtedly result in a more stable feed with an associated increase in mill throughput.

# 2 BACKGROUND TO MORILA

## 2.1 History

Morila Gold Mine is located in Southern Mali, approximately 280 kilometres by road south-east of the capital city, Bamako. It is an open pit gold mining operation.

In summary, as per the feasibility report by Reynolds (1999), exploration in the area commenced in the early 1950's. Randgold Resources Limited (RRL) acquired the Morila permit when they acquired BHP Minerals Mali in October 1996. The Morila pre-feasibility study was completed in 1998 with the final feasibility study completed in February 1999.

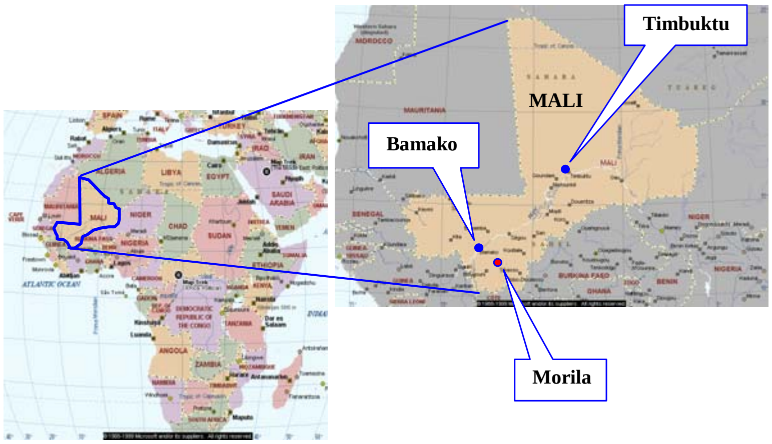


**Figure 1 – Map of Africa and Mali**

In July 2000, AngloGold acquired its third operation in Mali when it purchased a 40% interest in Morila from Randgold Resources Limited for a sum of US$ 132 million. Randgold retain a 40% interest in the joint venture with the Malian government holding the remaining 20%. AngloGold is the operating partner at Morila.

Production started on the 4th of October 2000 with first gold being poured on the 18th of the same month. In the first full year of production, 2001, Morila produced 630,000 ounces of gold at a total cash cost of US$ 103 per ounce. In the second year of production Morila produced in excess of one million ounces at a total cash cost of US$ 74 per ounce.

Mali is now the third largest gold producer in Africa after South Africa and Ghana.

## 2.2 Geology

The Morila orebody is hosted within an interpreted metamorphosed impure arkose and feldspathic arenite (formerly recorded as metagreywacke), a metamorphic rock dominated by quartz, plagioclase, biotite and alkali feldspar. X-ray diffraction and petrology studies indicate a near uniformity of gangue (silicate) mineralogy throughout the sequence, extending several hundred metres beyond the proposed final pit depth. The south-eastern portion of the Morila pit incorporates a portion of a tonalite intrusive which is also uniform in composition, consisting of plagioclase, quartz and biotite with minor chlorite and amphibole (Weedon, 2004).

According to Reynolds (1999), *"the gold mineralization is hydrothermal in origin and is contained within altered metasediments close to the contact with intrusive tonalite. Alteration is commonly silica-feldspar alteration as well as minor argillic alteration. The visible sulphide mineralization consists of arsenopyrite, pyrrhotite, pyrite and trace chalcopyrite. Coarse visible gold is a common occurrence".*

## 2.3 Mining

The mining at Morila is outsourced to Somadex, a French contract mining company, that is wholly owned by DTP, which in turn is a subsidiary of the giant construction company Bouygues. The mining contractor manages both the drill & blast and load & haul operations at Morila, with the short & long term planning, survey and mineral resource management functions being undertaken by Morila S.A.

The contractor's primary loading fleet consists of a CAT 5130 shovel, two Liebherr 994 shovels, a Liebherr 994 excavator, a CAT 990 front-end loader and two CAT-D10N dozers. The three shovels and excavator are teamed up with a fleet of 18 CAT 777D haul trucks, and one CAT-D10N dozer with a fleet of 6 CAT 631D wheel-tractor scrapers, with the CAT 990 loader dedicated to crusher-feed re-handle. The production blast-hole drill rigs consist of two IR-DM's and eight ROC-L8 machines (Christians, 2002). The annual mining capacity is estimated at 9.7 million bcm's (25.4 million tonnes).

The final pit will have a length of 1,190 metres, a width of 820 metres with the pit floor 196 metres below surface. The life of mine pit surface area equates to 66.6 hectares and on average 6.4 million cubic metres per annum (17.2 million tonnes) will be mined over the corresponding period. The life of mine stripping ratio is approximately 3.7:1.

### 2.4 Processing

Run-of-Mine (ROM) ore is transported directly from the open pit in 90 tonne payload, CAT 777D haul trucks. This material is either stockpiled on the ROM pad or directly tipped into the primary crusher. The Nordberg-54/75 primary crusher reduces the ore down to less than approximately 300 millimetres in a single stage, open circuit.

The crushed product is withdrawn from the primary crusher by an apron feeder and conveyed along conveyor 01-CVR-01 to the in-plant, crushed ore stockpile. The live capacity of the stockpile is approximately 10,000 tonnes.

Crushed material is withdrawn from the in-plant stockpile via three apron feeders situated below the stockpile. Ore is conveyed from the apron feeders to the SAG mill by conveyor 02-CVR-01. The ore is milled in a 6 MW, semi-autogenous grinding (SAG) mill in open circuit. The pulp exits the mill over a trommel screen.

Oversize from the trommel screen is crushed in a pebble crusher and is returned back to the SAG mill. Trommel undersize is pumped from the mill sump to the cyclone cluster for classification. The coarse undersize exiting the cyclone is returned to a single 6 MW, secondary ball mill operating in closed circuit. The pulp discharging the ball mill is combined in a common (to the SAG mill) discharge sump.

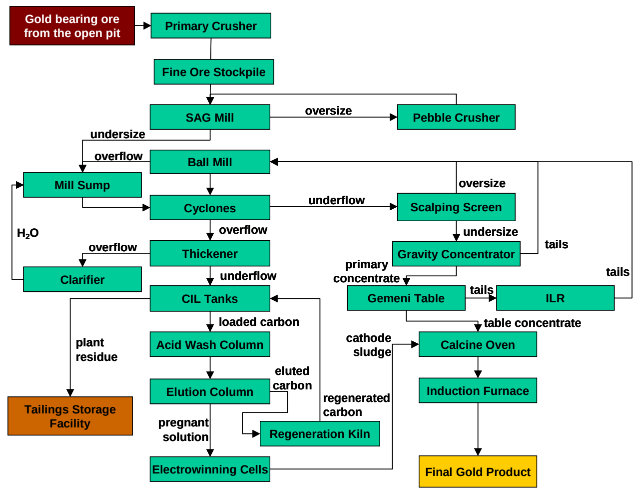


**Figure 2 – Block Flow Diagram - Morila Gold Plant**

Three centrifugal gravity concentrators operating in parallel recover coarse gold, post screening at two millimetres, from a bleed stream of the cyclone underflow. Gravity concentrate is redressed in the gold room over a Gemeni table. An Intensive Leach Reactor is integral to the gravity circuit. The cyclone overflow material passes over a linear screen for tramp removal prior to being thickened in a 35 metre diameter high-rate thickener. Clarified water is returned to the circuit for re-use in the metallurgical process.

Thickener underflow in the form of thickened pulp is pumped to the carbon-in-leach (CIL) circuit to be leached with cyanide as the lixiviant in seven 2,500 $m^3$ mechanically agitated tanks. The barren tailings material discharged from the last CIL tank is screened and pumped to the tailings storage facility (TSF).

Loaded carbon is acid treated prior to elution. The resultant pregnant solution following the twelve tonne Anglo American Research Laboratories (AARL) elution is subject to electrowinning. Gold is electrowon from pregnant electrolyte onto stainless steel cathodes. The cathode sludge as well as redressed gravity concentrate is then calcined and smelted separately into gold doré.

The plant was designed with a throughput of 250,000 tonnes per month. The overall plant recovery is 92%, with 40% of the gold being recovered by gravity techniques.

## 3 PIT-TO-PLANT

The Mine-to-Mill® philosophy has been developed by the Julius Kruttschnitt Mineral Research Centre (JKMRC) and their commercial division JKTech own the trademark. Mine-to-Mill® is a holistic approach to mining and mineral processing.

### 3.1 The Morila Concept

At Morila we have embarked on a similar programme to the JKMRC Mine-to-Mill® concept, in order to maximise mill throughput.

The programme has been termed Pit-to-Plant (p2p) and is a concept incorporating the disciplines of geology, mining, metallurgy and engineering, to optimise the entire comminution circuit. At Morila, comminution commences in the open pit with the fragmentation of ore through blasting and continues at the metallurgical facility with primary crushing, SAG and ball milling including a recycle pebble crusher.

The principal objective of the p2p programme is to increase mill throughput, primarily through increased blast fragmentation, but it must be stressed that this must not be at the expense of grade control, thus it is imperative that ore body dilution be minimised.

The program is in essence a quality control program in its approach to feed the milling circuit with the correct size and grade of material in order to attain optimum throughput.

### 3.2 Methodology

In order for Morila to achieve the objective of increased mill throughput, a mechanism that would be able to measure the size distributions and validate any modifications, be it a process parameter tweak or a blasting design modification, was needed. The "system" would need to be reliable and would need to measure and produce repeatable, accurate results (size distributions). In addition the system would need to be safe to operate, user friendly, cost effective, allow for continuous monitoring and be non-disruptive to the process as well as being relatively maintenance free.

The various project phases included a scoping study where objectives were defined. With regards the Split-Online® system set-up, project phases included design, fabrication, installation and commissioning. Following the scoping study the analysis stage included taking measurements, conducting trial runs, compiling photographs, collating data and interpreting the results. The optimisation phase included the optimisation of techniques and procedures. During the final controlling stage, outcomes were measured and depending on the results generated the project phase cycle was repeated.

### 3.3 The System

Two companies were identified as being able to "deliver" a product/system required for the task at hand. Split Engineering LLC based in Tucson, Arizona was the supplier of choice based on the criteria as listed in 3.2 above and justification therefore is beyond the scope of this technical paper.

The development of the Split software started at the University of Arizona in Tucson with principal developers being Dr. John Kemeny (Prof. Rock Mechanics at the University), Kirstin Girdner, and several Master's and PhD students. Since its commercialization from the university in 1997, it has been continually developed and upgraded primarily by Jeff Handy, Director of Development at Split Engineering (BoBo, 2004).

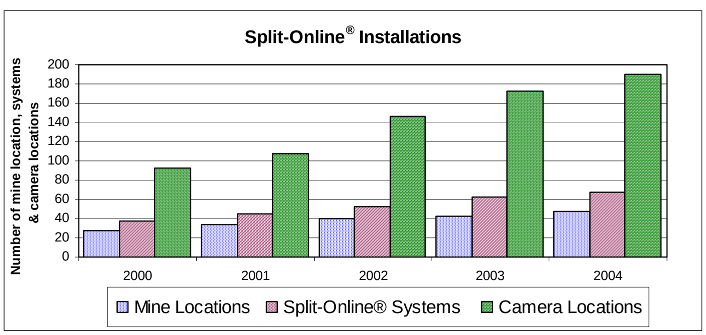


**Figure 3 – Split-Online® Installations**

As of April 2004, 68 Split-Online® systems have been installed with 190 digital cameras operating on 48 mine sites in 20 countries on 5 continents treating ore sources from copper to gold to iron ore to diamonds, as well as including a granite quarry producing aggregate (BoBo, 2004).

Trending from the data available in Figure 3, various operations are using more multi-camera installations in an attempt to better understand "their" total comminution circuit and the effect of size distribution on their production throughput.

### 3.4 Camera locations at Morila

In total 8 cameras were installed at Morila to monitor the ore size distributions at various stages of the comminution process. The list below details the locations of the camera installations.

*Camera 1 and 2*

Ore is tipped directly into the crusher from two tipping positions, 180º apart. Two cameras were installed to monitor the size distribution of the material being fed into the primary crusher, one at each tip location.

*Camera 3*

A third camera was installed at the discharge of the primary apron feeder (01-APF-01) delivering crushed product from the primary crusher via the apron feeder and stockpile conveyor (01-CVR-01) onto the crushed ore stockpile. It is the intention that the information received from the Split-Online® system installed at this location is communicated directly to the crusher's PLC, to automatically adjust the crusher gap depending on the size distribution of the crusher product. Currently the information generated is used to alert the crusher operator, who then manually interfaces with the crusher PLC via a touch screen, to make the necessary adjustments, when required.

*Camera 4*

A fourth camera was installed on the mill feed conveyor (02-CVR-01) to monitor the feed to the SAG mill.

*Camera 5, 6 and 7*

As crushed ore is deposited onto a conical stockpile (single point discharge) natural segregation of the material occurs. Each of the three apron feeders (02-APF-01, 02, 03) withdrawing material from the base of the stockpile varies in their product size distributions. The online information received from each of these cameras on the apron feeders is used to monitor the size distributions of the recovered material thereby allowing the mill throughput to be optimised by reconstituting the feed through blending. The apron feeder drives are variable speed and withdrawal rates from each unit can be individually controlled.

*Camera 8*

The milling circuit is a standard SABC circuit operated in closed circuit with a Nordberg 400HP pebble crusher. The final camera was installed on the pebble crusher feed conveyor, 02-CVR-03.

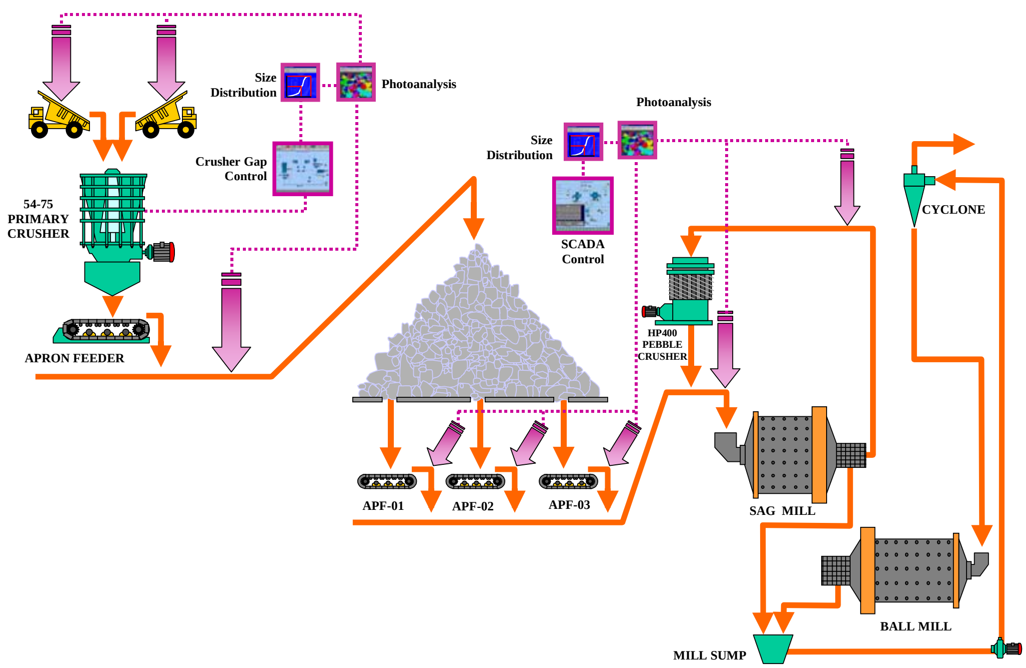


**Figure 4 – Schematic Showing the Camera Locations**

## 3.5 Cost of Installation

The final project capex cost totalled approximately US$ 250,000 and included all the equipment, the software, the initial scoping site visit, the commissioning on site and a post commissioning visit including time and travel expenses. The cost of procurement of the Split-Online® equipment (computer, cameras, lighting and various control boxes), together with the transportation costs, noting that Mali is a land locked county situated in West Africa are included. No cost was associated with the design of the various support structures and camera housings as this was conducted in-house at Morila. Material and fabrication costs are included in the figure above. Labour and the cost of a crane have been excluded from the total cost, as this was available utilising on site personnel and equipment. The cost includes all fibre optics and cabling.

### 3.6 Photographs and AutoCAD® drawings

| | |
|---|---|
| Top Left: | A photograph of the structure on the mill feed conveyor. This was the first installation and included a stairway to facilitate easier access for maintenance. |
| Top Right: | The structure on conveyor 02-CVR-03, the pebble conveyor. Note the more cost effective cat ladder. |
| Middle Left & Right: | AutoCAD® drawings (side view and front view) of the structure on the mill feed conveyor. Initially sheeting over the entire structure was included. |
| Bottom Left: | Detail showing the lighting and camera box. |
| Bottom Right: | An AutoCAD® drawing showing the box including the camera and lighting layout at one of the apron feeder chutes. |

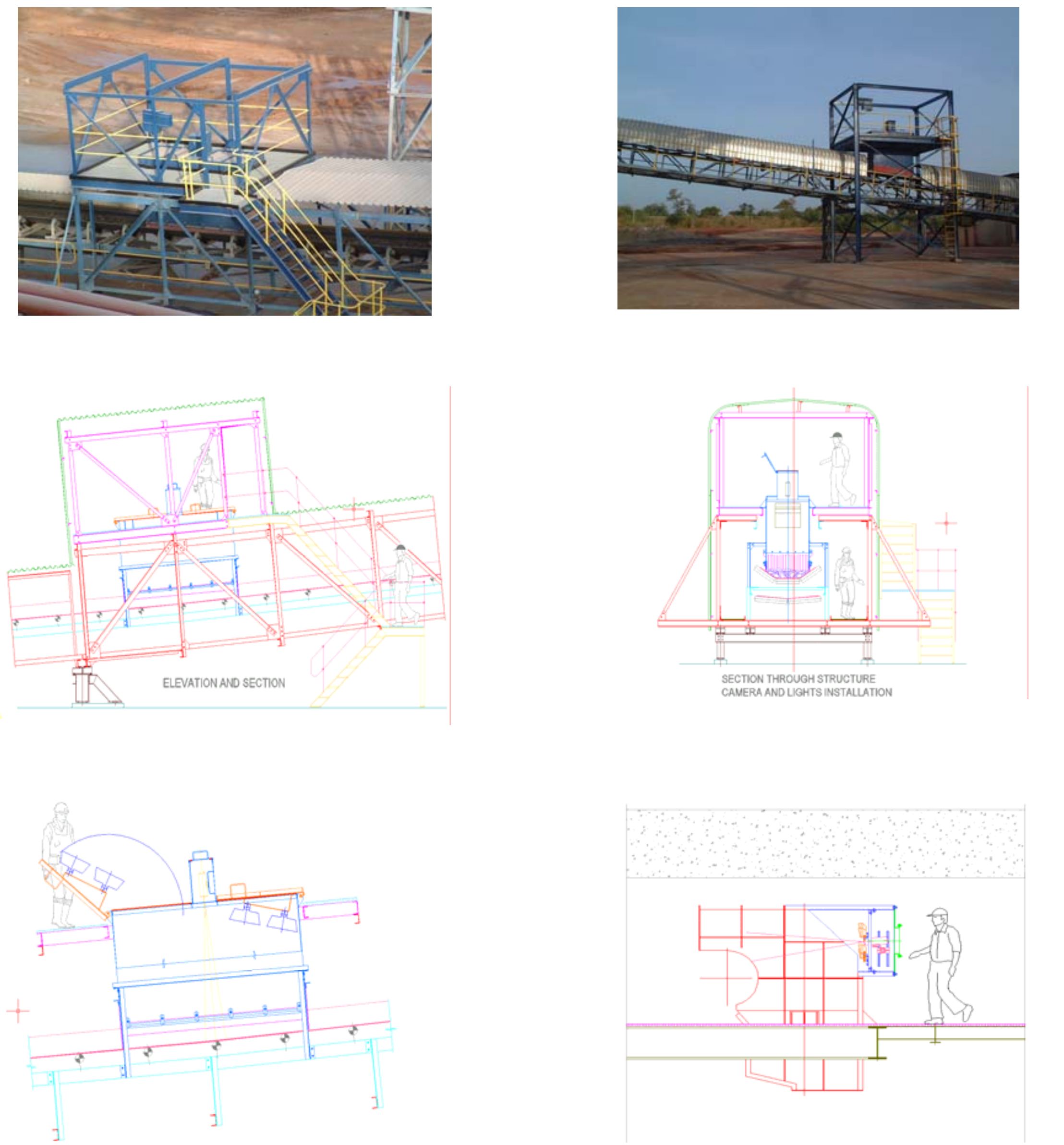


**Figure 5 – Photographs and AutoCAD® drawings**

# 4 RESULTS AND DISCUSSIONS

## 4.1 The Period

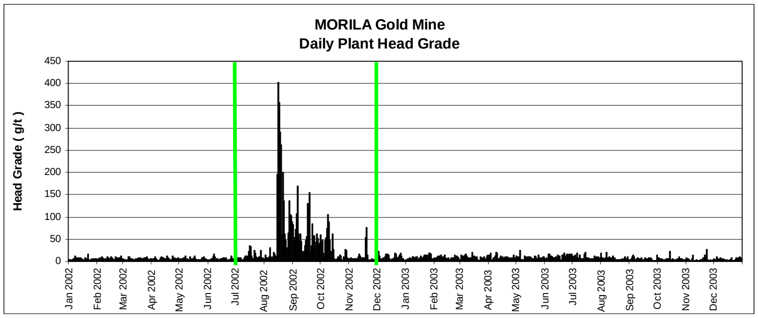


**Figure 6 – Daily Plant Head Grade ( g/t )**

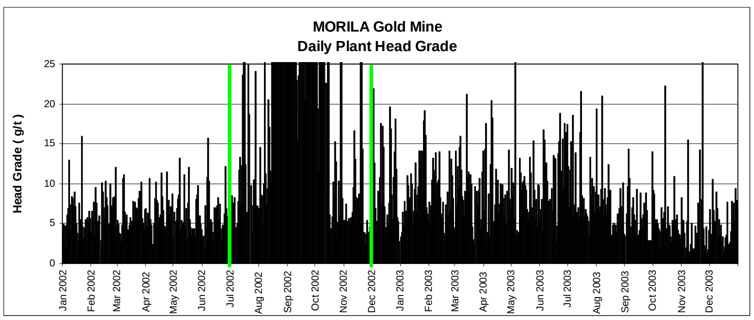


**Figure 7 – Daily Plant Head Grade ( g/t ) - Enlarged**

The Pit-to-Plant (p2p) programme was initiated in May 2002. Two months into the programme unusually high-grade ore was fed through the metallurgical plant during the period starting July 2002 to November 2002.

Refer to Figure 6 – Daily Plant Head Grade ( g/t ) and Figure 7 – Daily Plant Head Grade ( g/t ) - Enlarged, to view the enormity of the head grade assays during this period. The maximum head grade, averaged over an hour, during this high-grade period was 943 g/t in August 2002.

It is important to note that in order to combat the unusually high residues encountered as a consequence of the additional precious metal reporting to the circuit during this “high-grade” period a reduction in mill throughput was required. The effect of deliberately reducing the mill throughput, reduced the flow through the Carbon-in-Leach circuit thus increasing leaching residence times in order to maintain the recovery of the high-grade ore.

## 4.2 Mill Throughput

The tonnage throughput (weighted tonnes per hour) through the milling circuit from January 2002 to April 2002 was 365 tonnes per hour (tph). The tonnage throughput (weighted tonnes per hour) from May 2002 through to December 2003 was 399 tonnes per hour, excluding the period from July 2002 to November 2002 at 287 tonnes per hour.

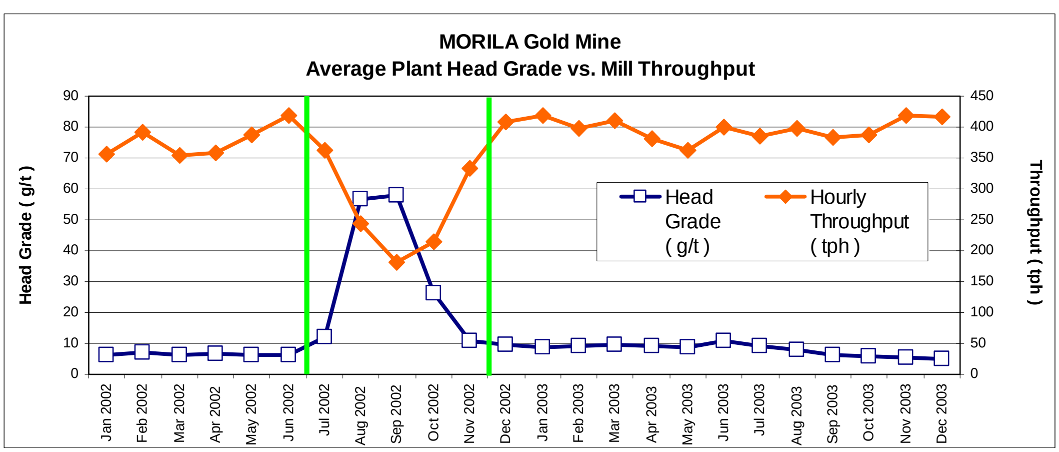


**Figure 8 – Average Plant Head Grade ( g/t ) vs. Mill Throughput ( tph )**

## 4.3 Size Distributions

### *4.3.1 Fines Envelope and Mill Throughput*

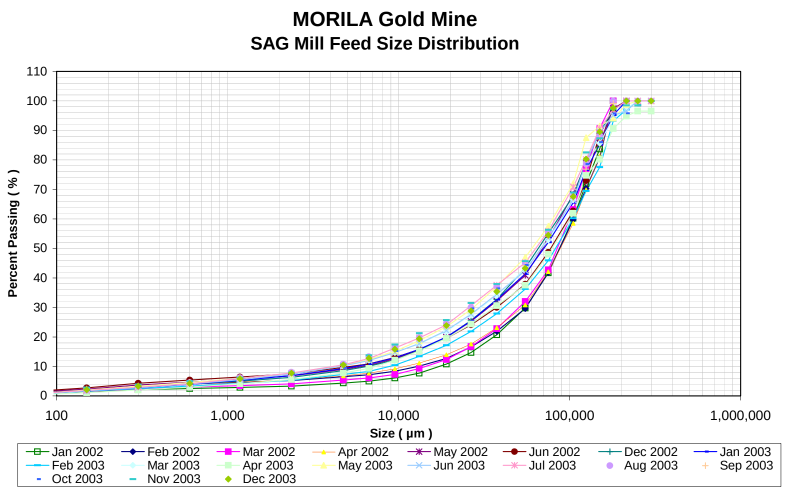


**Figure 9 – SAG Mill Feed Size Distribution - All Data**

Referring to Figure 9 on the left, by plotting the cumulative percent passing on the y-axis versus the sieve size on a logarithmic scale on the x-axis, the following graphs are realised. By examining the graphs one can clearly see that the size distributions for Jan '02, Feb '02, Mar '02 and Apr '02 are all concentrated in the lower right hand portion of the graph, indicating a more coarse size distribution

The $F_{80}$ for both the periods is comparatively similar (a variance of 6.9%), so the question to ask is "Why is the throughput for the period May '02 through to Dec '03 higher?" An increase in mill throughput from 365 tph to 399 tph was realised, a staggering 9.3% increase during the period in question. The answer is not to be found in the actual size of the top size material, as this is similar for both the periods, but the increased throughput for the May '02 - Dec '03 period can be explained in the increased generation of fines termed the *fines envelope*. Refer to Figure 10 for a summary of the data presented in Figure 9.

With reference to Figure 10, the $F_{75}$ for Jan '02 - Apr '02 was 130 mm and for May '02 - Jul '03 was 118 mm. The $F_{50}$ for Jan '02 - Apr '02 was 89 mm and for May '02 - Jul '03 was 68 mm, i.e. fifty percent of the material was less than 89 mm for the period Jan '02 – Apr '02 and fifty percent of the material was less than 68 mm for the period May '02 - Jul '03. The $F_{25}$ for Jan '02 - Apr '02 was 43 mm and for May '02 - Jul '03 was 23 mm.

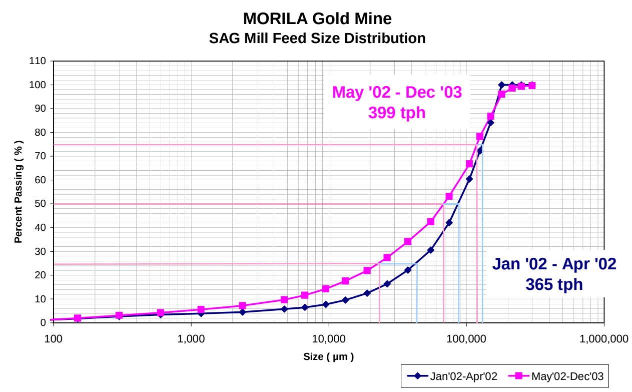


**Figure 10 – SAG Mill Feed Size Distribution - Summary**

### *4.3.2 Primary Crusher*

It is understood that *"blasting produces the ultra-fine material beneficial to mill throughput, [and that] the crushers are used to control top size and critical size material in the blast".* (Dance, 2001). See Figure 11 to view the data generated at Morila during the month of March 2004.

Comminution commences in the pit and efficient upfront fragmentation is more cost effective to implement to increase SAG mill throughput than a secondary crushing plant (Powell & Valery, 2002).

The capex cost to implement the p2p programme is approximately US$ 250,000 taking the production from 365 tph (250 ktpm[1]) to 400 tph (275 ktpm) and to establish a partial secondary crushing plant, US$ 9.8 million to increase the throughput from 400 tph to 511 tph

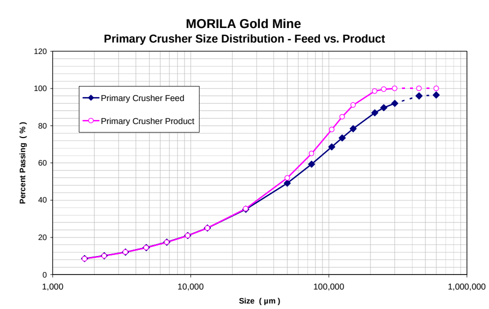


**Figure 11 – Crusher Feed vs. Crusher Product**

[1] 250 ktpm = 250 thousand tonnes per month or 250,000 tpm.

(350 ktpm).

### 4.4 Split-Online® versus Plant Laboratory Data

The Split-Online® camera system was commissioned from the 24th of June to the 1st of July 2003. During the commissioning of the Split-Online® system the average absolute difference between the digital data and the laboratory physical screening was 8.3% for the SAG mill feed material on 02-CVR-01. The size distributions of the calibration exercise from material on 02-CVR-01 can be viewed in Figure 12 – Calibration Results below. As can be seen from the December 2003 (Figure 13) screening analysis a good correlation still exists between the digital and manual screening analyses.

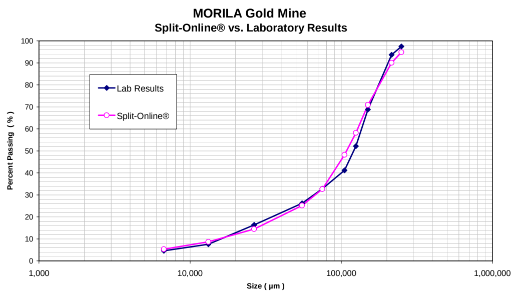


**Figure 12 – Calibration Results**

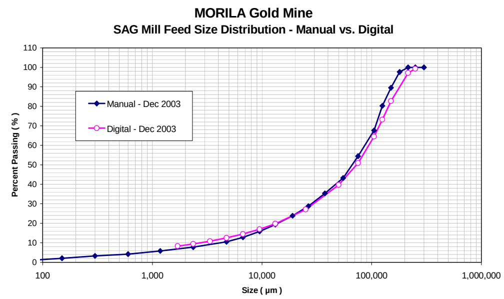


**Figure 13 – Manual Screening vs. Digital Analysis**

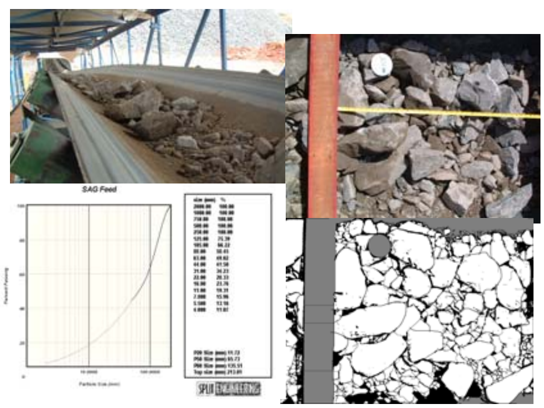


**Figure 14 – Blind Test Graphics**

Figure 14 – Blind Test Graphics; on the left is a collage of images pictorially describing the sequence of the evaluation exercise. A photograph (top right) of a one metre belt cut was taken and the photograph sent to Split Engineering for analysis. The white disc in the photograph measures 100 millimetres in diameter. The image on the lower right is a copy of the delineated image with the final size distribution positioned on the bottom left. The absolute difference of the “blind” test between the actual screening analysis performed on site and the digital analysis was 3.2% for the plus 105 mm fraction.

Split-Online® is able to determine the size distribution of the “visible” rock fragments as well as accurately estimate the percentage fines (BoBo, 2002). When comparing actual size distribution data from conveyor belts to digital analysis the average error of each size fraction is less than 10% (La Rosa et al, 2001). The results obtained at Morila have been able to validate the previous two statements above.

### 4.5 Real Time Data

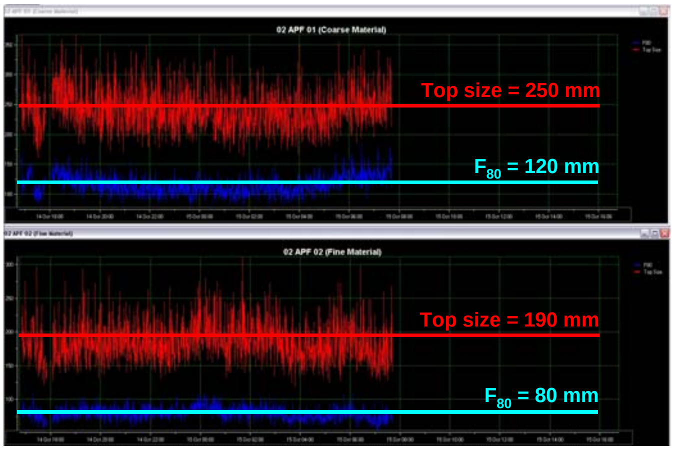


**Figure 15 – Real Online Data and Graphics**

Real time, Split-Online® data is graphically depicted in the actual "screen grabs" in Figure 15 – Real Online Data and Graphics. Apron feeder 02-APF-01 is located at the entrance of the stockpile tunnel and withdraws material closer to the periphery of the conical stockpile. Apron feeder 02-APF-02 is located in the centre of the stockpile tunnel and therefore withdraws material from more-or-less the centre of the stockpile. The $F_{80}$ of material recovered from apron feeder number one and apron feeder number two is 120 mm and 80 mm respectively.

In order to negate or at least minimise the natural effect of material segregation as a consequence of stockpiling, each apron feeder is monitored and controlled for enhanced blending purposes to optimise mill throughput. The apron feeder drives are variable speed and withdrawal rates from each unit can be individually controlled. Currently the setting of the rates is a manual operation and it is planned to improve the apron feeder control through automation using a form of expert based mill feed control.

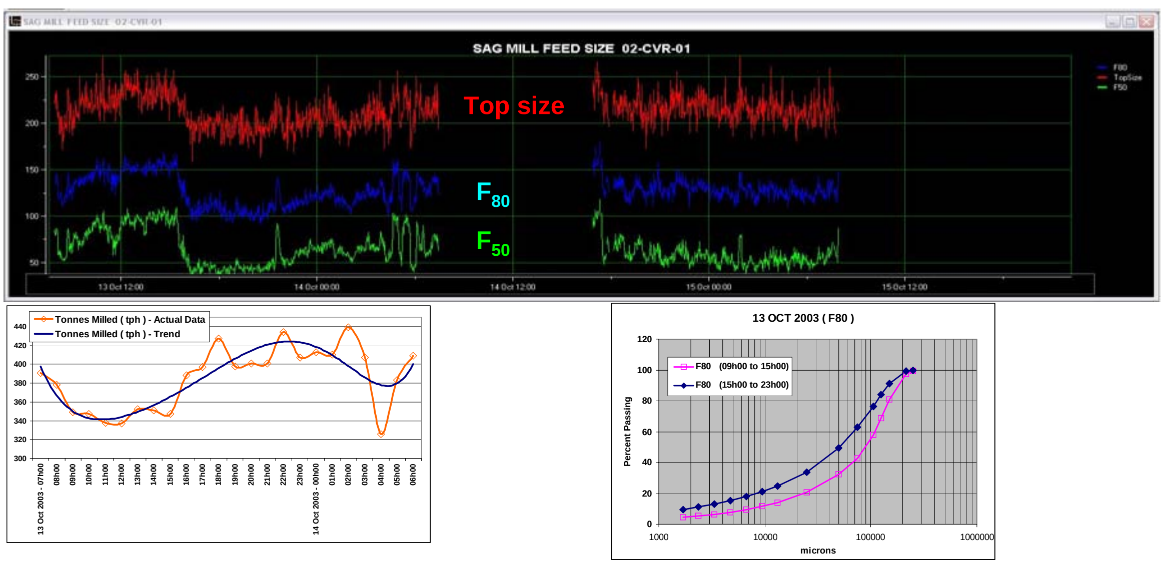


**Figure 16 – Confirmation of Size Distributions vs. SAG Mill Throughput**

Plotting the hourly throughput on a separate Excel® graph over the corresponding period as the online data and then compressing the scale on the Excel® graph to that of the real time graphic, Figure 16 is produced. As $F_{80}$ increases the mill throughput drops off and as the $F_{80}$ decreases mill throughput increases. The corresponding size distribution S-curves, also attached, once again support the significance of the fines envelope to enhanced the mill throughput.

### 4.6 Grade Control

As stated previously, for the p2p programme to be a success increases in throughput must not be at the expense of grade control.

Dilution may be subdivided into two components, namely "internal dilution" and "external dilution". Internal dilution is the waste material within the orebody that cannot be physically extracted from the ore due to mining constraints. It is typically not quantified but included in the ore grade estimates. External dilution is related to the material outside of the in-situ, pre-blast ore block boundaries, which is not included in the ore grade estimates. Typically this can be tracked by reconciling truck counts and average truck tonnage factors to the in-situ ore block tonnes. Mine planning budgets, as per the LOM plan allow for 10% “mining dilution” and a 5% “ore loss” during the mining process. Dilution and ore loss is estimated to be within these parameters to date. The estimation of long-term dilution has been complicated by the very high grades experienced at Morila and the subsequent revisions in the modelling process (Poulin et al., 2004).

Using the Mine Call Factor (MCF) as a measure to quantify the quality control effectiveness of the p2p programme and bearing in mind that the MCF is in part a measure of ore body dilution, amongst other variables, but also includes less objective factors for example interpretation of the orebody model. The MCF (grade control vs. metallurgical plant check out) for the 2003 year with reference to tonnes, grade and ounces is 102%, 96% and 98% respectively. In other words with the MCF based on ounces at 98% the process plant “saw” 2% less gold than that predicted by the orebody model; a reasonably good correlation.

### 4.7 Blasting and Fragmentation

In summary (Day, 2004), the improvements in fragmentation observed over the Pit-to-Plant programme can be assigned basically to the application of universal good blasting practice. Supervision and quality control were tightened up, with “buy-in” being achieved from the mining contractor.

The powder factor was steadily increased over the period from 0.65 kg/m$^3$ to 0.85 kg/m$^3$. Burden to spacing ratio was kept as near as practical to 1.25. A staggered pattern (as used before the p2p period) was maintained. During this process stemming height was adjusted and refined in order to contain the blast without creating blocky ground within the stemmed section of the blast. Where possible blasts were orientated to throw perpendicular to the strike of identified local joint sets. In addition the crest row of shot holes were carefully positioned to ensure that the burden was correct in this row. The size of the individual blast blocks was maximised. Videos were taken of each blast, and critical analysis was undertaken. The need to maintain loading selectivity and reduce dilution led to the need to choke blast all ore blasts. The use of the normal 80%:20% emulsion:ANFO blend resulted in too much back-break within choke blasts and this led to poor drilling performance in subsequent adjacent blasts. In order to counter this, straight emulsion was used as the blasting agent. This having a higher shock energy, and lower heave energy resulting in a shattering effect on the blast block. The desired result was achieved. All initiation was undertaken using “Nonel” shock tubes. A trial was undertaken using electronic detonators, and no discernable difference was observed. This trial was however undertaken prior to the installation of Split Engineering’s Split-Online® fragmentation analysis system being installed. Consideration is being given to a further electronic detonator trial in the near future (Day, 2004).

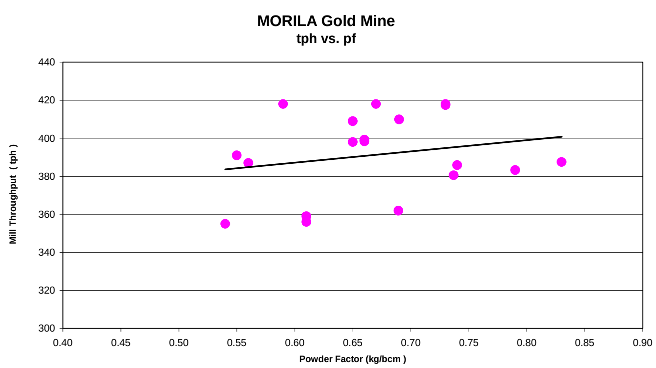


**Figure 17 – Tonnes per Hour vs. Powder Factor**

Morrell (2003) states that the powder factor is the largest driver in generating a more favourable feed size distribution although this relationship may not be linear. Morrell’s data ranged from a powder factor of 0.6 to 1.3 kg/m$^3$ whereas only small incremental changes to the powder factor within a range of only 0.2 kg/m$^3$ have been made at Morila.

An increase in SAG mill throughput has been attributed to an increase in powder factor at Morila as can be seen in Figure 17 – Tonnes per Hour vs. Powder Factor.

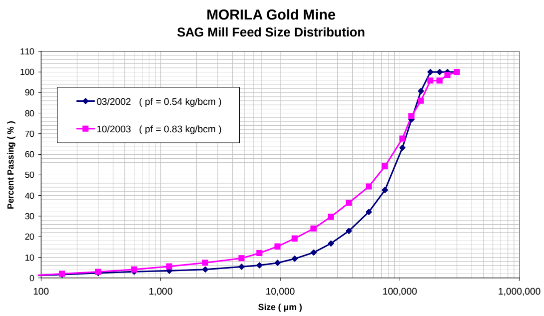


**Figure 18 – Varying Powder Factors vs. Throughput**

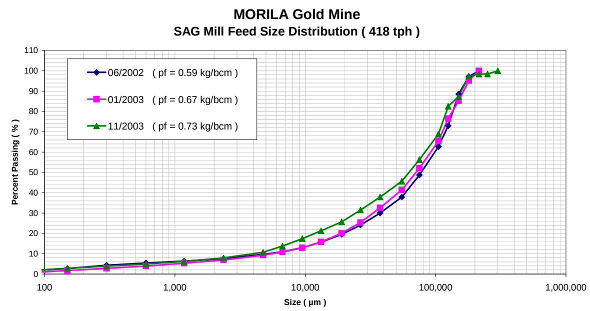


**Figure 19 – Varying Powder Factors @ 418 tph**

Although Figure 18 – Varying Powder Factors vs. Throughput[2] confirms the relationship between powder factor and size distribution and hence mill throughput it is important to note with reference Figure 19 – Varying Powder Factors @ 418 tph[3] that many other factors for example, and not limited, to percent critical mill speed (Morila is in a fortunate position to have a variable speed SAG mill operating from $N_C$ 65-85%), mill load, percent steel filling and mill liner condition, as well as the ore characteristics such as the various work indices will also effect the SAG mill throughput and therefore solely not size distribution dependent.

### 4.8 Costs

The various costs per activity for the period 2002 – 2003, spanning 24 months can be viewed in Figure 20. The mining costs include mining supervision and planning, survey and drafting, diesel and all costs associated with the mining contractor. The drill & blast figure excludes the cost of “pre-split” and the total drill & blast cost including the pre-split cost is included in the mining cost. Together with the geology cost the mining cost is 32% of the total Morila cash cost (US$ per tonne treated). The drill & blast cost (excluding pre-split) is 21% and 7% of the mining cost (including geology) and the total mine cash cost respectively. The costs for geology includes mineral resource administration, grade control, geotechnical costs etcetera.

Plant costs include all costs associated with operating the metallurgical facility. All power and water supply costs for the entire mine are included in the plant costs. Additional costs allocated to the plant include costs associated with the Tailings Storage Facility (TSF), the laboratory and engineering workshops etcetera. The plant cost is 32% of the total Morila cash cost.

Costs included in the line “other costs” include elements such as, site administration, transport, asset protection, materials management, safety, health and environmental as well as the cost to run the school, the clinic etcetera. CPS and Ad Valorem refer to taxes and royalties relating to gold sales.

An incremental increase in the drill and blast cost at Morila of US$ 0.07 per tonne treated (Day, 2004) since the inception of the p2p programme is minimal (an increase of 0.2% based on the total cash cost) in comparison to the additional revenue “brought forward” as a result of an almost 10% increase in mill throughput together with the associated precious metal content therein, not withstanding *the improved load and haul productivity* (Morrell, 2003) generally encountered as a result thereof in the mine.

[2] Mill throughput for March 2002 was 355 tph and the throughput for October 2003 was 388 tph.
[3] The powder factor for June 2002, January 2003 and November 2003 was 0.59, 0.67 and 0.73 kg/m$^3$ respectively.

| Average Costs ( US$ per tonne treated ) for the period 2002 - 2003. | |
|---|---|
| Drill and Blast | 2.31 |
| Mining | 10.03 |
| Mining (incl. Geology) | 10.76 |
| Metallurgical Plant | 10.69 |
| Other | 4.26 |
| CPS, Ad Valorem | 6.01 |
| Management Fee | 0.99 |
| Refining, Freight & Insurance | 0.42 |
| **Total Cash Cost** | **33.13** |

**Figure 20 – Average Costs ( 2002 - 2003 )**

## 5 CONCLUSIONS AND THE WAY FORWARD

It is clear that the mill feed size distribution is a major contributing factor influencing mill throughput. There is a greater percentage of finer material being fed to the mills during periods of increased tonnage. Increased focus on blasting parameters, not so much for the reduction of top size material but more for the increase in the generation of fines, termed the "fines envelope" has been identified as being paramount for the increase in mill throughput.

The online, digital size analysis has made it possible to monitor and control the feed size through each phase of blasting, primary crushing and stockpile apron feeder ratio control. With the installation of the cameras at the primary crushing tip, more detailed size distributions, per blast, are now available for interpretation. With the information gained using the "Spilt-Online®" system as a tool, Morila is able to make significant progress in quantifying the effect of changing the various blasting parameters for increasing the production of "finer" material thus enhancing mill throughput. Continued focus to increase blast fragmentation remains vital to improvements in mill throughput.

In the complex comminution circuits of today implementing an "expert system" that utilises a fuzzy-logic, rules-based system to automate process decisions based on real-time and historical trends would be beneficial in further improving efficiencies (and profitability) around the comminution circuit within the process plant. Since the inception of the p2p programme an increase in throughput of approximately 10% has been achieved. Linking the Split-Online® system to the expert system is the next step in generating additional rewards.

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