Measuring and Modelling the Impact of Primary Crusher Stockpiles on Autogenous/Semi-Autogenous Mill Feed Size

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ABSTRACT

Comminution circuits generally have a coarse ore stockpile after the primary crusher to decouple the relatively uneven mined ore supply from the downstream comminution and concentrator circuits which require a stable feed. Whilst this assists with evening-out throughput disturbances, it can influence the size distribution presented to the downstream comminution circuits through size-segregation effects. This is particularly important in Autogenous (AG) and Semi-autogenous (SAG) circuits. What is not often considered, however, is that the stockpile may also cause size reduction. Size distribution data from stockpile feeds and stockpile products were collected from 13 different plants around the world. Analysis of these data supports the assertion that size reduction occurs in the stockpile. A simple power-based model is described which accurately predicts the degree of size reduction measured. Worked examples are provided which show how the model can be used to predict the stockpile product size given data on stockpile geometry and ore hardness.

INTRODUCTION

It is usual in comminution circuits to have a coarse ore stockpile after the primary crusher to decouple the effects on the comminution/concentrator circuits of the relatively uneven supply of material from the mine to the primary crusher. As primary crusher availability is generally lower than the downstream comminution circuits the stockpile additionally enables feed to continue to be delivered to the downstream comminution circuits when the primary crusher is under maintenance. The coarse ore stockpile assists with evening-out throughput disturbances; however, it can influence the size distribution presented to the downstream comminution circuits through size-segregation effects. In the case of Autogenous (AG) and Semi-autogenous (SAG) circuits this is particularly important (Morrell and Valery, 2001). However, what is rarely considered is that the stockpile not only can cause size-segregation it may also cause size reduction. This paper presents size distribution data from a number of coarse ore stockpiles in an attempt to measure the extent to which size reduction may occur. A mathematical model to describe this size reduction is also presented together with worked examples.

MEASUREMENT APPROACH

The stockpile feed (primary crusher product) and stockpile product (typically AG or SAG mill feed) size distributions used in this analysis were collected during plant surveys conducted for optimization projects. The collected size distributions cover several different commodities from operations globally.

The size distributions were measured by conducting sieving of belt cut samples. Due to the coarseness of the primary crushed material, a very large sample size is required to provide a representative sample (Barbery, 1972; Gy,1976). This presents a challenge with regards to the material's handling and sieving due to the quantities of material that must be handled. To minimize the sample size but achieve a representative sample, two separate samples were collected from the conveyor belts:

1. All the material collected from a measured length of belt (L1).
2. All material coarser than 75 mm collected from a second adjacent and much longer measured length of belt (L2).

The necessary belt length for the two samples, L1 and L2 (as shown in Figure 1), depends on the belt width, speed, and loading. In general, several full 200 litre drums of rock were collected, the material from L1 being kept separate from the material from L2. When collecting the +75 mm sample, all rocks that look big enough were initially checked using a hand-held trial sieve and selected if they are of the right size. Later, full sieve analysis removed any -75 mm particles that were accidentally collected. The data from sieve analysis of the L1 and L2 samples were analysed in terms of mass/belt length then combined into a single size distribution. An example of belt sampling is shown in Figure **2**. A detailed description of the procedure can be found in Napier-Munn et al (1996).

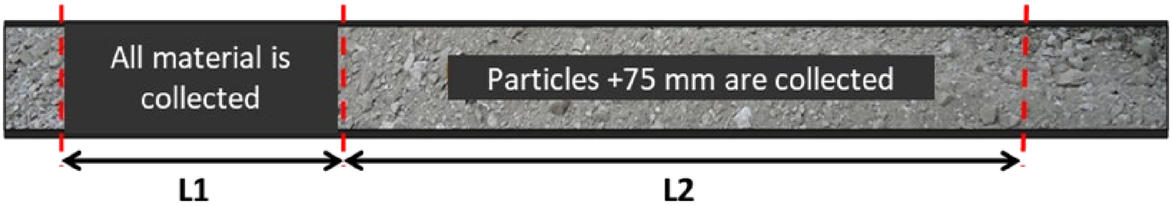

Figure 1 - Schematic for Collecting Belt Cut Samples

Figure 2 - Example of Belt Sampling - L1 (all material) on left and L2 (+ 75 mm) on right.

RESULTS

A summary of the results obtained is given in Table 1. In all cases the stockpile product 80% passing size (D_{80}) is finer than the stockpile feed D_{80}. Differences between individual results vary significantly and are believed to be due to differences in rock hardness and stockpile height. This former effect can be seen in Table 1 in which the data has been grouped into those with M_{ic} (SMC Test® crusher hardness parameter) values < 5 kWh/t and those with values > 5 kWh/t.It can be seen that the < 5 kWh/t group has a D_{80} difference between stockpile feed and stockpile product almost double that of the > 5 kWh/t group. Details of the individual size distributions are given in Appendix 1. To help visualise the consistent trend seen at all sites the feed and product size distributions have all been combined and are presented in Figure 3 in terms of the overall mean stockpile feed and product.

Table 1 – Results Summary

Site	Commodity	M_{ic}	D_{80} (mm)		Difference		
		kWh/t	stockpile feed	stockpile product	mm	%	
1	iron	2.0	148	106	42	28	Mean difference in D_{80} for materials with M_{ic} of < 5 kWh/t: 37 mm
2	copper	2.7	113	72	41	36	
3	iron	2.9	170	129	41	24	
4	copper	4.5	220	165	55	25	
5	copper	5.0	121	115	6	5	
6	gold	5.1	83	75	8	10	Mean difference in D_{80} for materials with M_{ic} of > 5 kWh/t: 20 mm
7	gold	5.9	83	69	14	17	
8	gold	6.0	167	140	27	16	
9	gold	7.9	128	110	18	14	
10	gold	8.0	117	101	17	14	
11	gold	8.4	247	202	45	18	
12	gold	9.4	131	128	3	2	
13	gold	10.1	163	135	28	17	
Mean		6.0	145	119	27	18	

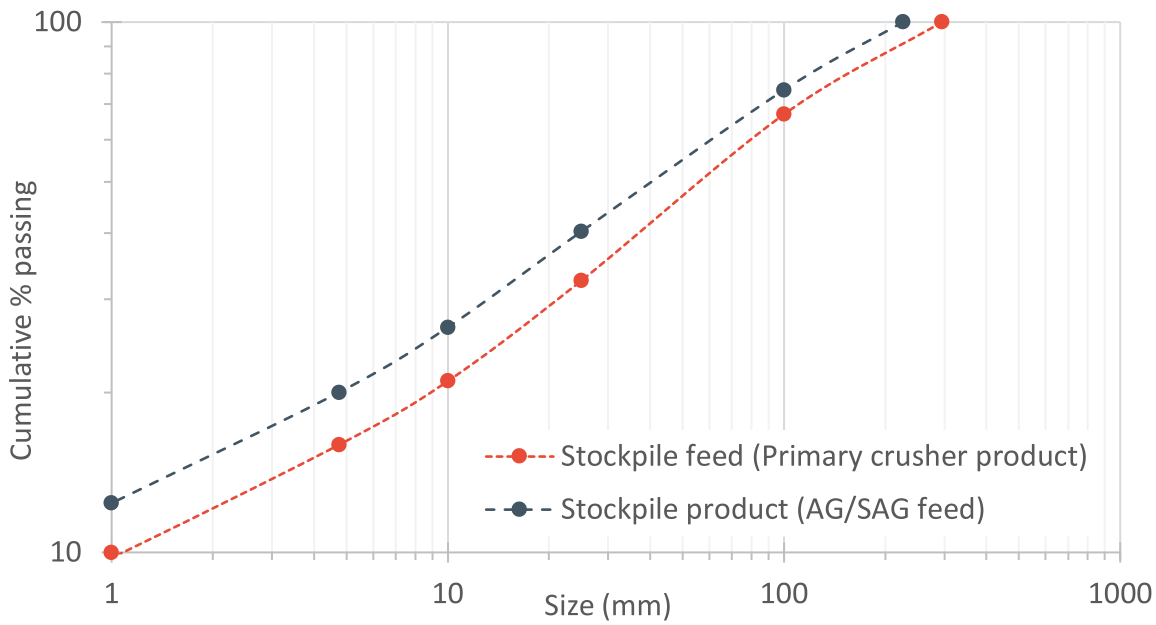

Figure 3 – Mean Stockpile Feed and Product Size Distributions

MODELLING

From a modelling perspective the size reduction that occurs in stockpiles has been assumed to be principally caused by breakage as rocks fall from the stockpile feed conveyor head-pulley and impact on the surface of the stockpile below. This (ballistic) mechanism is similar to that found in Vertical Shaft Impact (VSI) crushers. In the case of VSI crushers the energy fuelling breakage is provided by a rotor which propels the feed rocks at high speed to impact against stationary anvils or rock boxes. In the case of stockpiles gravity provides the energy, potential energy being converted to kinetic energy as the rocks fall. In recent research (Lewis-Gray and Rasmussen, 2019) it was found that the use of the SMC Test®'s crushing parameter, M_{ic}, in conjunction with Morrell's energy-size equations best described the size reduction performance of VSI machines. Hence it was decided to use the same approach to model size reduction in stockpiles.

The specific energy required for size reduction can be estimated using Morrell's energy-size relationship (Morrell, 2004; GMG Group, 2021). The general form of this equation is:

$$W = M_i \times 4 \times \left({P_{80}}^{f(P80)} - {F_{80}}^{f(F80)} \right) \qquad (1)$$

Where:

W	=	predicted circuit net specific energy (kWh/t)
M_i	=	hardness parameter (kWh/t)
$f(x)$	=	$-(0.295 + x/1000000)$
x	=	80% passing size in microns
P_{80}	=	80% passing size of the product
F_{80}	=	80% passing size of the feed

As applied to crushing circuits the form of equation 1 is as follows:

$$W_c = M_{ic} \times S_c \times k_3 \times 4 \times \left({P_{80}}^{f(P80)} - {F_{80}}^{f(F80)}\right) \quad (2)$$

Where:

W_c	=	predicted net specific energy of the crushing circuit (kWh/t)
M_{ic}	=	SMC Test® crusher hardness parameter (kWh/t)
S_c	=	coarse feed factor
	=	$55 \times (F_{80} \times P_{80})^{-0.2}$: $(0.5 < S_c < 1)$
k_3	=	open/closed circuit factor
	=	1.19 for open circuit
	=	1.0 for closed circuit

Equation 2 is normally used to estimate how much specific energy is required to be delivered by a crusher to reduce a given feed size (F_{80}) to a given product size (P_{80}). As applied to stockpile comminution and the aims of this paper, equation 2 is required to be used to estimate what the stockpile product P_{80} is likely to be given a feed size (F_{80}) and the height of the drop to the stockpile. To do so W_c, M_{ic} and k_3 need to be known. W_c in this case is the specific energy that rocks, falling from the stockpile feed conveyor head-pulley absorb as they impact the stockpile surface below, this absorbed energy being what causes breakage. Implicit in this approach is that the falling rocks on impact are broken not those already residing in/on the stockpile.

In general, for a rock of mass, m, falling from a height, h, under the influence of gravity, g, its potential energy, PE, is given by:

$$PE = m \times g \times h \quad (3)$$

As the rock falls it progressively converts this potential energy to kinetic energy, until on impact some is absorbed by the rock. If what is absorbed exceeds the yield stress, breakage will occur. How much kinetic energy is available on impact will depend on how much is lost to drag resistance, whilst how much is subsequently absorbed by the rock depends on the nature of the impact surface. If the impact surface is hard, rigid, flat and perpendicular to the impact velocity (say a flat concrete floor), the proportion of energy absorbed will be relatively large. However, if the impact surface comprises, say, an inclined loose bed of material the proportion will be lower. This latter situation is more likely to be what rocks falling onto a stockpile will encounter. To accommodate drag losses and the influence of the impact surface conditions a fitted lumped "efficiency" factor, k_{eff}, is incorporated into equation 3. Its value can vary in the range 0-1. Hence the potential energy absorbed by a rock, $PE_{absorbed}$, is given by:

$$PE_{absorbed} = m \times g \times h \times k_{eff} \quad (4)$$

There are also other factors which can influence the product size including segregation and attrition and abrasion breakage which are not accounted for in the model; these are also lumped into the k_{eff} efficiency factor. The proposed model is simple by necessity, based on impact breakage only, using data that can practically and reliably be collected. Regardless of the necessary simplification and assumptions the model achieves good accuracy as will be demonstrated in the results presented later in Table 2.

The value of h (height of drop) in equation 4 will be related to the maximum stockpile height and the operating level. Assuming that the operating level provides an impact surface which has a height relative to an empty stockpile of h_{op} and representing the maximum stockpile height by h_{max}, the ratio h_{op}/h_{max} can then be defined as a relative operating level L_{op} where L_{op} varies in the range 0-1, 0 being the condition of an empty stockpile and 1 when it is full. Equation 4 can now be written:

$$PE_{absorbed} = m \times g \times h_{max} \times (1 - L_{op}) \times k_{eff} \tag{5}$$

As specific energy is energy/mass then, using equation 5, the specific energy absorbed by rocks falling on to a stockpile, W_{sp}, can be represented by:

$$W_{sp} = g \times h_{max} \times (1 - L_{op}) \times k_{eff} \tag{6}$$

The data in Table 1 were not originally collected with the view to developing a stockpile size reduction model and hence there is limited information concerning stockpile heights (h_{max}) and operating levels (L_{op}). To overcome this, stockpile heights were estimated based on assuming a typical live capacity in tonnes, C_l. From a design perspective this is often calculated using the number of hours that the live capacity is required to keep the downstream comminution operating at its nominal throughput capacity during periods where there is no material being fed to the stockpile. Hence, for example, in the case of Newcrest Cadia operation (Dunne et al, 2001), it had a stockpile live capacity designed to last 20 hours, which at its original nominal comminution circuit throughput of 2000 t/hr gives a live capacity of 40,000 tonnes. In contrast, Newmont Boddington operation (Hart et al, 2011) has a live capacity designed to last 8 hours, though as its nominal comminution circuit throughput is 5000 t/hr it also has a live capacity of 40,000 tonnes. With reference to Table 1, the mean live capacity in terms of hours, for those sites where it was known, was 14. For those sites where it was not known, 14 hours was assumed. Multiplying the live capacity hours by the throughput of each comminution circuit, the live capacity of each stockpile in terms of tonnes was estimated. From simple geometry it can be shown that for a conical stockpile with a central offtake, the live capacity is 25% of the stockpile total volume (see Appendix 2). This figure agrees with that quoted by Zamarano (2006). Hence the total stockpile capacity is 4 times the live capacity:

$$Total\ stockpile\ capacity\ (tonnes) = 4 \times C_l \tag{7}$$

Assuming a void fraction of 0.3 and using the measured rock sg, the volume of the stockpile, V_{sp}, is given by:

$$V_{sp}\ (cu.m) = 4/(1 - 0.3) \times C_l/sg \tag{8}$$

Which reduces to:

$$V_{sp}\ (cu.m) = 5.714 \times C_l/sg \tag{9}$$

Having estimated the total stockpile volume, an angle of repose is required to estimate the maximum stockpile height, h_{max}. From Froehlich (2011) the mean angle of repose of graded rocks in the same size range as those in Table 1 is 38 deg. Hence h_{max} can be calculated as follows:

$$h_{max} = \left(\frac{3 \times V_{sp} \times (Tan\ 38)^2}{\pi}\right)^{1/3} \tag{10}$$

Which reduces to:

$$h_{max} = 0.836 \times V_{sp}{}^{1/3} \tag{11}$$

Combining equations 9 and 11 we get:

$$h_{max} = 1.5 \times \left({C_l}/{sg}\right)^{1/3} \tag{12}$$

Having defined h_{max}, then with reference to equation 6, this leaves L_{op} and k_{eff} as unknowns. It is considered good practice to operate with high stockpile level (70% or higher) as much as possible to avoid issues with segregation (Morrell and Valery, 2001). Therefore, it was decided to assume that L_{op} was on average equal to 0.5 which is equivalent to assuming that on average the stockpiles in Table 1 were operated with 87.5% of the live capacity available (see Appendix 2). The parameter k_{eff} was initially left as a fitted parameter. Equation 6 can now be reduced to:

$$W_{sp}(kWh/t) = 0.00204 \times \left({C_l}/{sg}\right)^{1/3} \times k_{eff} \tag{13}$$

With reference to equation 2, P_{80} is the 80% passing size of the stockpile product, F_{80} is the 80% passing size of the stockpile feed, W_{sp} replaces W_c, M_{ic} is directly measured by conducting SMC Tests on representative samples of stockpile feed, whilst k_3 is set to the open circuit value of 1.19 hence giving equation 14.

$$W_{sp} = M_{ic} \times S_c \times 1.19 \times 4 \times \left(P_{80}{}^{f(P80)} - F_{80}{}^{f(F80)}\right) \tag{14}$$

Given that the objective is to rearrange equation 14 such that it can be solved for P80, the fact that the exponent of the P80 term is a function of the P80 and the S_c term also contains the P80, makes an analytical solution impossible. However, numerical approximation techniques were found that provided very accurate estimates and gave rise to equations 15 and 16. Alternatively, the excel "solver" function can be used to search for a P80 value which satisfies equation 14.

$$P_{80} = -65500 \times \left(\left(\ln\left(\frac{W_{sp}}{(4 \times M_{ic} \times 1.19 \times S_c)} + F_{80}{}^{-\left(0.295 + \frac{F80}{1000000}\right)}\right)\right) + 3\right) \tag{15}$$

$$S_c = 57.4 \times F_{80}{}^{-0.4} \qquad : (0.5 < S_c < 1) \tag{16}$$

Using equations 15 and 16 on the data in Table 1, the best fit for k_{eff} was found to be 0.7, resulting in the predictions detailed in Table 2 and plotted graphically in Figure 4. The relative error standard deviation was found to be 6.5% indicating an accuracy at the 90% confidence level of just over 10%.

It is emphasized that the model predicts changes in the 80% passing size. Given that it uses a power-based equation to do so, an underlying assumption is that the gradient in log-log space with respect to the 80% passing size of the stockpile product will be the same as the stockpile feed. Inspection of Figure 3 indicates that there is a slight deviation from this assumption and that the stockpile product tends to have slightly more fine material than would otherwise be expected. It is known (Morrell and Valery, 2001) that as stockpiles are drawn down the stockpile product tends to become coarser and as it builds up will tend to be finer. In addition, due to size segregation, certain feeders tend to produce finer material than other feeders. As control of stockpile levels and feeder choice were not explicitly controlled when collecting the data in Figure 3, it is possible that the stockpile product size distributions were influenced by these phenomena. Hence the slight increase in finer material may be due to a slight overall bias in the operation of the stockpiles and feeders in the data base favouring the production of finer material. However, an alternative hypothesis is that the additional fines are caused by comminution not accounted for in the model. As it stands, the model considers only impact breakage and further assumes that on average 50% of the total available potential energy is used for this purpose (by virtue of the assumption that L_{op} = 0.5). Once the rocks falling from the head-pulley have impacted on the stockpile below they continue to move further down in a "shuffling" motion until they finally exit the feeder, thereby effectively using up the potential energy remaining after the initial impact. This "shuffling" motion occurs under the progressively increasing pressure of the stockpile above. Such conditions are ideal for abrasion and attrition breakage, both of which are known to create relatively fine products. At this stage this remains conjecture as there is insufficient information to prove its validity. It is suggested that a DEM study may well be able to shed further light on this subject.

Table 2 – Observed vs Predicted Stockpile P_{80}

Site	Metal	M_{ic}	stockpile product P_{80} (mm)	
		kWh/t	observed	predicted
1	iron	2.0	106	110
2	copper	2.7	72	76
3	iron	2.9	129	140
4	copper	4.5	165	154
5	copper	5.0	115	98
6	gold	5.1	75	75
7	gold	5.9	69	77
8	gold	6.0	140	143
9	gold	7.9	110	112
10	gold	8.0	101	107
11	gold	8.4	202	195
12	gold	9.4	128	124
13	gold	10.1	135	139
Mean		6.0	119	119

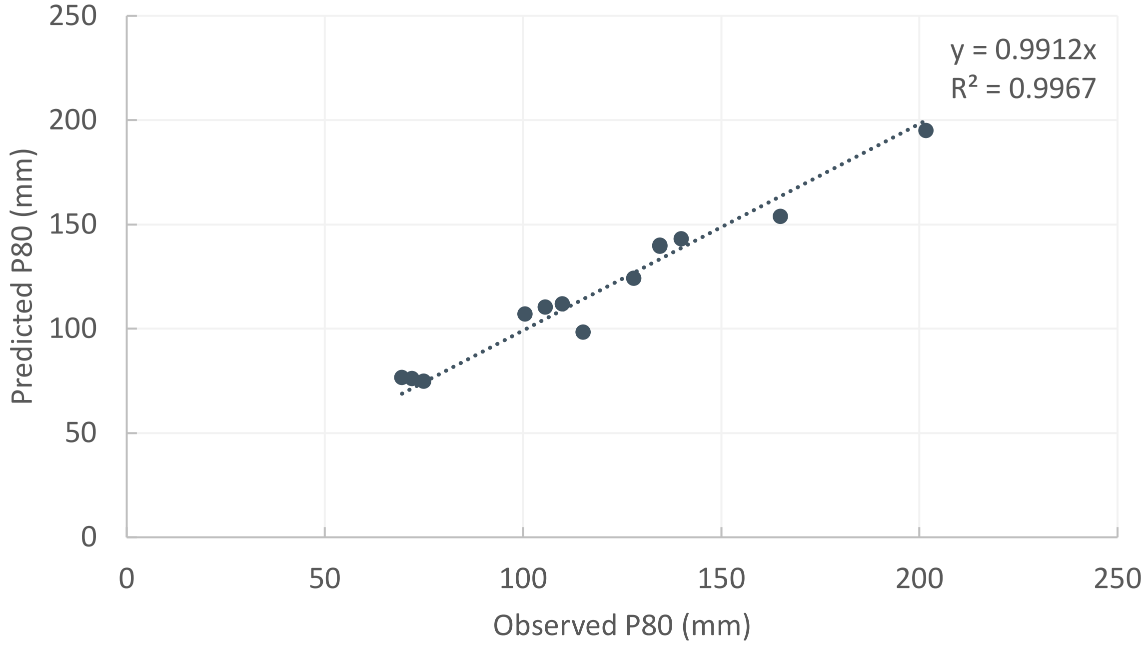

Figure 4 – Stockpile Size Reduction Model Accuracy (Observed vs Predicted Stockpile P80)

Using 0.7 for k_{eff} in equation 13 and substituting for W_{sp} in equation 15, equation 15 can now be reduced to:

$$P_{80} = -65500 \times \left(\left(\ln\left(\frac{0.0003 \times \left({}^{C_l}/_{sg} \right)^{1/3}}{(M_{ic} \times S_c)} + F_{80}^{\;-\left(0.295 + \frac{F80}{1000000}\right)} \right) \right) + 3 \right) \qquad (17)$$

Equations 16 and 17 are best used in situations such as greenfield designs where the stockpile height has yet to be finalised. However, in cases where the stockpile height is known then by combining equation 6 with equation 15, the resultant equation 18 should be used instead of equation 17.

$$P_{80} = -65500 \times \left(\left(\ln\left(\frac{0.0002 \times h_{max}}{(M_{ic} \times S_c)} + F_{80}^{\;-\left(0.295 + \frac{F80}{1000000}\right)} \right) \right) + 3 \right) \qquad (18)$$

CONCLUSIONS

Analysis of size distribution data from coarse ore stockpile feed and product size distributions supports the assertion that size reduction occurs in stockpiles. A simple power-based model assuming impact breakage to be the sole source of size reduction was developed. It was found to predict the degree of size reduction, measured in terms of the 80% passing sizes, to within an accuracy of just over 10% at the 90% confidence level. Worked examples which show how the model can be used to predict the stockpile product size given data on stockpile geometry and ore hardness are provided in Appendix 3.

On average the size distribution data collected from the 13 plants included in this study indicate that the stockpile products may contain slightly more fine material than a simple impact comminution model would suggest. Whilst biases in the way that the data were collected cannot be ruled out, it is possible that the apparent additional finer material may have been the result of abrasion and attrition breakage not accounted for in the model. DEM studies of stockpile dynamics may be able to shed light on this phenomenon.

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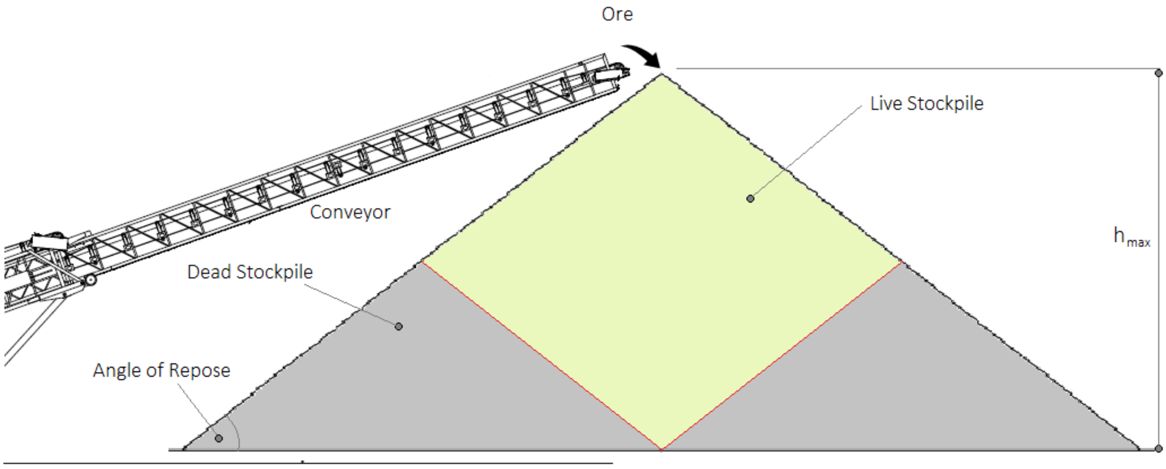
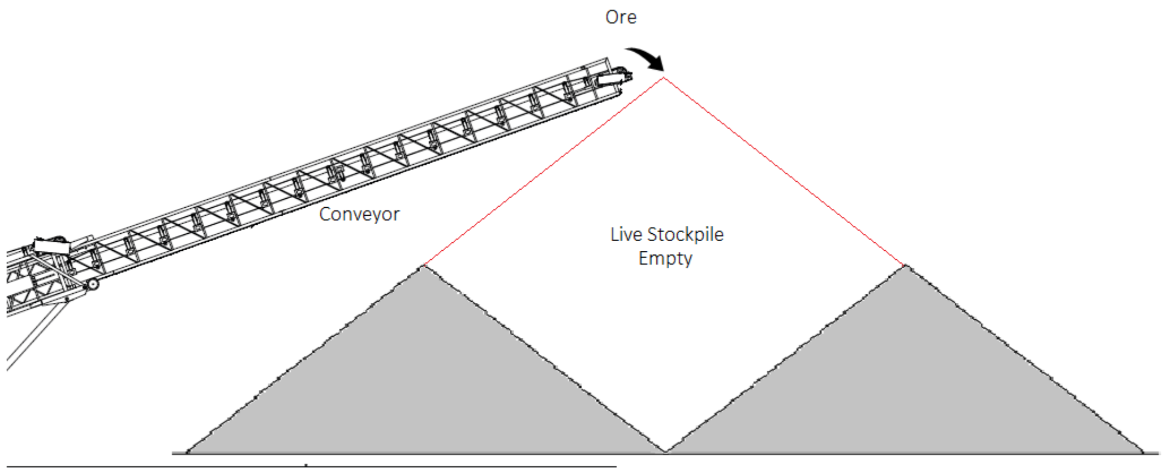
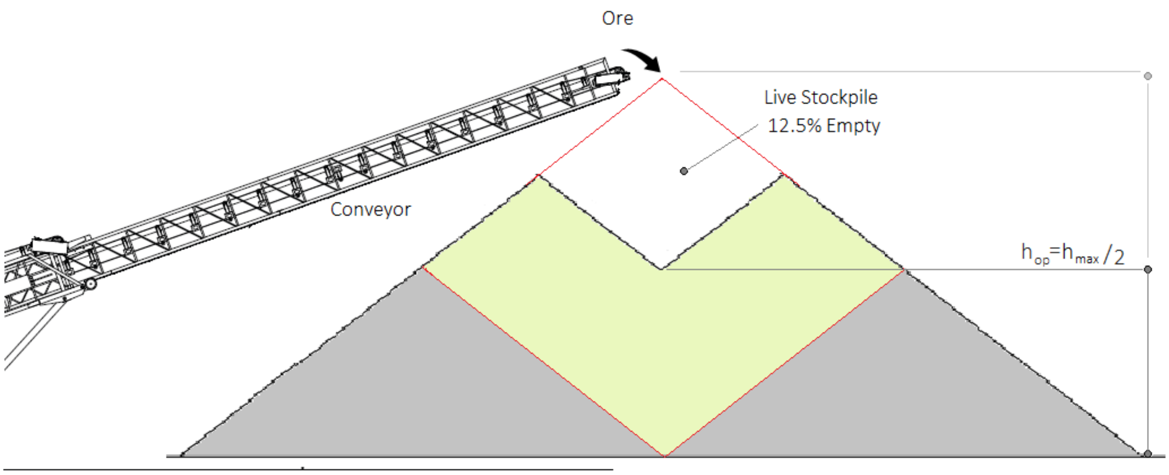
APPENDIX 1 – Measured Size Distributions of the Stockpile Feed and Product

Site	-1 mm (%)		- 4.75 mm (%)		-10 mm (%)		-25 mm (%)		-100 mm (%)	
	feed	prod	feed	prod	feed	prod	feed	prod	feed	prod
1	15.5	17	22	24	26	29	37	40.5	67	79
2	20.5	23	30	35	37.5	45	50	60	77.5	88
3	16	19.5	21	25	24	29	30.5	41	61	74.5
4	9	11	17	20	23	27	32	37	54	62
5	5	8.5	12.5	17.5	18.5	24	32.5	38	73	76
6	28	30	38.5	43	45	51	57.5	63	86	89
7	5	8.5	13	19	22	29	40	50	88	89
8	6	9.5	12	15.5	17.5	21.5	29.5	34	60.5	70
9	4.5	6.5	8.5	12.5	12	18	28	35	70	76
10	5.5	6	10.5	13	15	19	28	37	72	79.5
11	5	5	8	8	11	11	16	16	38	44
12	4.5	10.5	9.5	18	14	26	25	41	65	71
13	2.5	6	5	10	8	16	17.5	31.5	59	70
mean	9.8	12.4	16.0	20.0	21.0	26.6	32.6	40.3	67.0	74.5

APPENDIX 2 – Stockpile Geometry







APPENDIX 3 – Worked Examples

A. Stockpile Height Known

Equations:

$$P_{80} = -65500 \times \left(\left(\ln \left(\frac{0.0002 \ \times h_{max}}{(M_{ic} \times S_c)} + F_{80}{}^{-\left(0.295 + \frac{F80}{1000000}\right)} \right) \right) + 3 \right)$$

$S_c = \ 57.4 \times F_{80}{}^{-0.4}$: (0.5 < S_c < 1)

Input data:

Stockpile height (h_{max})	=	32m
M_{ic}	=	7.6 kWh/t
Expected primary crusher product (F80)	=	152mm

Calculations:

$$S_c = \ 57.4 \times 152000^{-0.4}$$

$$= \ 0.485$$

As 0.485 < 0.5 and S_c must fall in the range 0.5 < S_c < 1 then S_c is set to 0.5

$$P_{80} = -65500 \times \left(\left(\ln \left(\frac{0.0002 \ \times 32}{(7.6 \times 0.5)} + 152000^{-\left(0.295 + \frac{152000}{1000000}\right)} \right) \right) + 3 \right)$$

$$= -65500 \times (-5.03416 + 3)$$

$$= 133238 \text{ microns}$$

Hence the stockpile product (AG/SAG mill feed) is expected to have on average a P80 of 133mm

B. Stockpile Height Unknown

Equations:

$$P_{80} = -65500 \times \left(\left(\ln \left(\frac{0.0003 \times \left({C_l}/{sg} \right)^{1/3}}{(M_{ic} \times S_c)} + F_{80}{}^{-\left(0.295 + \frac{F80}{1000000}\right)} \right) \right) + 3 \right)$$

$S_c = 57.4 \times F_{80}{}^{-0.4}$: ($0.5 < S_c < 1$)

Input data:

Nominal comminution circuit throughput	=	2500 tph
Required time for live stockpile to last	=	12 hours
M_{ic}	=	4.2 kWh/t
Sg	=	2.7
Expected primary crusher product (F80)	=	120mm

Calculations:

$$Live\ capacity\ of\ stockpile\ (C_l) = 2500 \times 12\ tonnes$$

Hence:

$$C_l = 30000\ tonnes$$

$$S_c = 57.4 \times 120000^{-0.4}$$
$$= 0.534$$

$$P_{80} = -65500 \times \left(\left(\ln \left(\frac{0.0003 \times \left({30000}/{2.7} \right)^{1/3}}{(4.2 \times 0.534)} + 120000^{-\left(0.295 + \frac{120000}{1000000}\right)} \right) \right) + 3 \right)$$

$$= -65500 \times (-4.52935 + 3)$$

$$= 100172 \text{ microns}$$

Hence the stockpile product (AG/SAG mill feed) is expected to have on average a P80 of 100mm