The impact of ore characterization and blending on metallurgical plant performance

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Synopsis

Black Mountain, a polymetallic base metal mine located in the Northern Cape territory of South Africa has been blessed with high grade economic ore for many years. Now that the Broken Hill orebody is nearing the end of its life, a lower grade orebody (Swartberg) is being exploited to supplement production from Broken Hill. In addition, ore from stopes in the upper levels of the Broken Hill orebody is considerably tarnished (oxidized). As a natural result, the remaining ore contains a higher percentage of magnetite, pyrrhotite and pyrite than in earlier years. Bismuth, cadmium and cobalt also occur in the ore in higher quantities. The life of the mine has been lengthened with the discovery of further deeper-lying ore reserves extending from the Broken Hill orebody suitably called the Deeps section. However, to bridge the gap between development of the new reserve and the exhaustion of the old reserve, these lower grade and tarnished (oxidized) ores have to be treated. The mine has had reasonable success in identifying unfavourable or problematic ore types and stockpiling them separately to blend in small quantities with more favourable ore. Bench and plant scale tests have aided identification of the various ore types and significant improvements in flotation metallurgical results have been seen. Copper, lead and zinc metal occur as sulphides in the ore. The mill consists of a differential and sequential flotation process that extracts copper concentrate, lead concentrate and zinc concentrate respectively with silver metal recovered in the copper and lead concentrates. This paper describes the ore blending strategy employed at the mine to maximize the econo-metallurgical efficiency of the operation.

Mine plan and background

Black Mountain, a division of Anglo Operations Limited is a base metal mine located 110 km east of Springbok in the Northern Cape. It operates complex ore mining copper, lead, zinc and silver. The mine has been in existence since 1978 and is the only mine in South Africa producing copper concentrate, lead concentrate, zinc concentrate and silver contained in copper and lead concentrates. The mine processes 1.56 Mt/a and is one of the largest employers in the area. In September 1998 Black Mountain became a wholly owned subsidiary of Anglo American plc. The mine has undergone significant change in the restructuring process initiated by Anglo led by various benchmarking exercises relating with sister mines across the globe. This process has seen significant productivity improvements and a redesign of work practices and structures. The primary goal was to ensure the survival of the mine by securing the Deeps Project. By virtue of this project the life of mine was extended by at least 13 years. The project required the sinking of a new vertical shaft and an upgrade within the processing plant.

The Broken Hill orebody has served Black Mountain mine well over the years. However, while under the joint ownership of Gold Fields of South Africa Ltd and Phelps Dodge exploration activities were scaled down when it appeared that the Broken Hill ore reserve was reaching its end. In fact, the orebody had pinched out and when Anglo American plc acquired the mine, exploration activity was reinitiated and it was confirmed that the Broken Hill orebody extended wider and deeper. An intensive feasibility study was undertaken by the mine to prove the reserve and investigate a workable mining solution. By mid-2000 the Deeps Project was approved consisting of a new 1 750 m vertical shaft and development of the existing Broken Hill decline to reach the deeper-lying reserve.

Figure 1 shows the dilemma faced by the mine. Production from the Broken Hill orebody would taper down to 1.12 Mt/a from 1.56 Mt/a and be depleted by mid-2002 at current production rates, while the Deeps section would only come into production by mid-2003. With this scenario, the mine would have closed down in 2003. However, the Board of Anglo American plc took the decision to mine an alternative orebody at Swartberg, to supplement the downscaled production from Broken Hill. The Swartberg orebody was originally earmarked as the primary reserve

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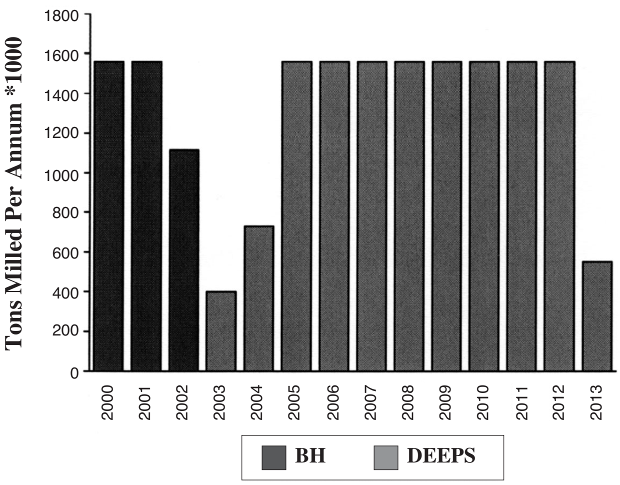

Figure 1—The dilemma facing the mine post-discovery of the 'Deeps' section

and the mine bears its name from this reserve, but was abandoned prior to commencement of production when geological studies indicated that higher metal grades could be exploited from Broken Hill. In late 1995 decline development at Swartberg was commissioned. The Swartberg mine is located 7 km from Broken Hill and ore is hauled by truck to the surface ore pad at Broken Hill. In this way, Black Mountain can maintain an annual production rate of 1.56 Mt/a (Figure 2). However, the blend of Swartberg ore with Broken Hill ore poses specific processing problems. The concentrator plant was designed to handle a specific range of ore metal grades. In addition to the lower metal grades of the Swartberg orebody, both the upper and lower orebodies are oxidized near the surface, while the upper orebody is rich in contaminants such as bismuth and cobalt.

At present the Deeps section consists of a proven reserve which extends the current life of mine to 2013. Exploration activity is continuing and it is expected that more reserves will be proven especially when exploration can be pursued with drilling from underground rather than from surface.

Mineralogy

Swartberg ore

Mineralization at Swartberg is hosted by the Aggeneys Ore Formation which comprises an upper and a lower orebody. In the structural hangingwall of the upper orebody copper mineralization reaching ore grade proportions is present. Polyphase deformation and metamorphism complicate the geology of the Swartberg orebody.

Table I

Predicted head grades

	%Cu	%Pb	%Zng	/tAg	g/t Bi	g/t Co	%Mgt
Broken Hill	0.4–0.9	4.5–6.5	2.0–3.0	76–85	66–106	120–160	25–35
Swart-berg	0.4–1.3	2.6–5.1	0.5–1.1	21–107	122–282	152–206	40–55

The grade of Swartberg ore compares with Broken Hill as shown below.

Swartberg upper orebody

This orebody is comprised of an iron formation which consists of the following lithological units: magnetite quartzite, magnetite amphibolite and magnetite barite rock. The former two units host the sulphide mineralization. The magnetite content in the magnetite quartzite varies between 10 and 97% with a mean of 55%, whereas in the magnetite amphibolite it varies between 10–87% with a mean of 59%. In the magnetic quartzite, pyrite is the dominant sulphide and is accompanied by pyrrhotite. Galena is the dominant economic sulphide, followed by chalcopyrite and accessory sphalerite. In the magnetic amphibolite, pyrrhotite is dominant over pyrite, while galena and sphalerite are the major economic sulphides. In both rock types the higher-grade ore occurs as coarse re-crystallized sulphides. In the well-mineralized areas bismuth content is high, varying between 50 and 300 ppm.

Swartberg lower orebody

The orebody is situated structurally, approximately 20 metres below the upper orebody and consists of four dominant rock types namely sulphidic quartzite, garnet quartzite, sulphide schist and magnetic amphibolite. One or two discontinuous massive sulphide lenses are developed in the sulphidic quartzite and sulphidic schist. In the south-eastern part of the lower orebody, a relatively thick, high-grade ore shoot is developed. Pyrite is the dominant sulphide mineral whereas the dominant economic sulphides are comprised of finely disseminated galena, sphalerite and minor chalcopyrite. The average magnetite content of the lower orebody is 4% while bismuth levels are considerably lower than in the upper orebody.

High grade copper lenses

Two separate high grade copper lenses, labelled 'main' and 'hanging'-wall copper lenses, are developed in quartzite and garnet quartz schist in the hangingwall of the upper orebody. The sulphide content is variable with pyrite and chalcopyrite being the dominant sulphides. Minor galena, sphalerite and pyrrhotite are encountered near the contact with magnetite quartzite of the upper orebody. Magnetite content increases

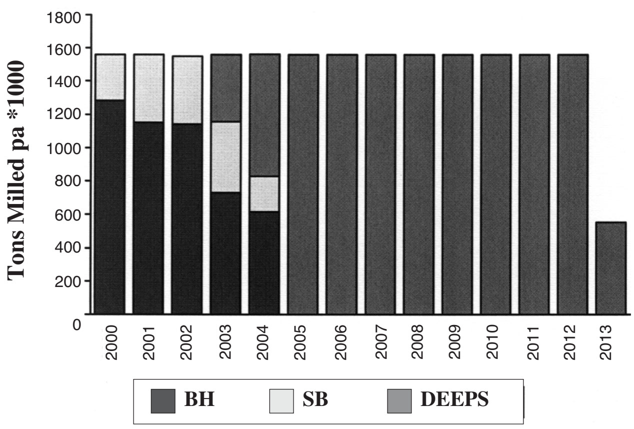

Figure 2—The solution securing transition into the 'Deeps' section

close to the contact with the upper orebody. Bismuth content in the copper lenses is very high, varying between 200 and 1 500 ppm Bi.

Mineralogical observations of bulk samples of the Swartberg orebody

A total of 12 flotation concentrates taken during the second bulk Swartberg plant trial and three of the resultant concentrate samples were sent for mineralogical investigation (February 1997) to determine:

- The bismuth minerals present and their mode of occurrence
- The reason for the lead and zinc contamination in copper concentrate of the 'Swartberg' samples, and
- The mode of occurrence of cobalt in the zinc concentrates.

This study indicates that cosalite [$Pb_2Bi_2S_5$] is the major bismuth phase and occurs either intergrown with galena in chalcopyrite, or as isolated inclusions in chalcopyrite. Bismuthinite [Bi_5S_3] is the second major bismuth phase and occurs as free, mostly rather small (<20 μm) independent grains or as crack filling and inclusions in chalcopyrite, or enclosed in galena.

The concentrate samples (from the Swartberg orebody plant trials) showed similar mineralogical and liberation characteristics. The average grain sizes varied between the samples, and those with the higher lead content were much finer. Cosalite and bismuthinite were found closely associated with chalcopyrite in the copper concentrate samples and in many cases some galena was present as well. The high bismuth content of the lead concentrate from the Swartberg ore was due mainly to bismuthinite (and probably cosalite) which are intergrown or in contact with galena. In the zinc concentrate, chalcopyrite and galena occur as intergrowths and contacts with sphalerite, and certain amounts of chalcopyrite occur as exsolution blebs inside the sphalerite. The cobalt contamination of the Swartberg zinc concentrate was due mainly to the presence of discrete cobalt-bearing minerals such as cobaltite, gersdorffite, löllingite, safflorite, etc. These occur as liberated grains, as well as attached to sphalerite.

Oxides

The Broken Hill upper orebody is oxidized in various proportions to a depth below surface of about 60 m. In addition, ore from the Swartberg mine and the Broken Hill upper levels stockpiled on the surface ore pad is exposed to weathering. This ore will, depending on the period of exposure, also oxidize.

Other parameters that influence metallurgical performance

Apart from ore mineralogy and oxidation many other ore characteristics influence plant performance.

Magnetic content of the ore

The magnetic circuit was designed to remove ± 30% of the ore prior to flotation. This means that 70% of the total ore reports to flotation at the designed head grades. Higher magnetic content of the ore has two implications:

- Capacity of the magnetic circuit must be increased
- Fewer tons of ore will report to flotation which can have both positive and negative effects.

Lower magnetic ore content reduces retention time in the flotation circuit with adverse effects on grade and recovery. It was found that in total 33% magnetic material could be removed without significant metal losses via the magnetic tails. Since the magnetic content of the Swartberg upper orebody is high and variable, the magnetic content influences crushing, milling and flotation performance significantly.

The geology of the orebody renders it impossible to match constant magnetic content to the required metal head values. Hence varying magnetic removal and especially magnetic scavenger flotation accommodates the variation in the magnetic content.

Head grade of the ore and metal ratios

The flotation circuits were designed to handle a certain range of head grades. If the head grade of any single mineral species is increased, more rougher concentrate is produced, higher circulating loads result and the circuits cannot cope. This is particularly the case for higher copper head grades. The ratios of the different head grades relative to each other (i.e. Cu:Pb, Cu:Zn and Pb:Zn) also influence flotation circuit performance and can invoke cross-contamination of concentrates.

Oxidation of the economic sulphide minerals

Oxidation of the ore results in poorer selectivity (increased lead and in particular zinc contamination of the copper and lead concentrates) during flotation and an increase in circulating loads. This has a detrimental effect on capacity and in severe cases, the milling rate is forcibly reduced and out-of-specification concentrates are produced.

The analytical method used to determining oxidation levels in the ore is not exact, but it is used on a comparative basis. Analysis of daily plant data indicated that an oxidation level of up to about 13% for copper and lead oxides could be tolerated without severe effects on grade and recovery provided that blend ratios are kept to an absolute minimum. Bench flotation test work also showed that blending small amounts (5% w/w increments) of highly oxidized ore with non-oxidized sulphide ore adversely affects concentrate grades and recoveries. It is thus unacceptable to blend oxide ore with sulphide ore to get to the 13% oxidation level boundary. The 13% oxidation level is therefore for tarnished sulphide ore and not for oxidized ore blended with non-oxidized sulphide ore.

Hardness of the ore

The hardness of the ore is mainly linked to the magnetic content of the ore. Results from grab samples sent to Mintek for Bond indexing are as follows:

Bond Work Index (kWh/t)

	Broken Hill	*Swartberg*
Rod Mill	13.00	16.10
Ball Mill	13.09	17.43

This indicates the range of hardness of the material. High hardness causes inadequate liberation of valuable mineral that adversely influences both grade and recovery of minerals and in extreme cases the ore throughput needs to be reduced.

Oil contamination of the ore

Laboratory test work has irrevocably shown that oil acts as a totally unselective collector yielding a bulk sulphide-type float. Galena floated by oil is virtually impossible to depress using sulphurous acid and at best difficult with dichromate. Excessive doses of dichromate would be required which is detrimental to the subsequent flotation of lead. Plant problems with control of the float reflect this clearly. Generally, oil at any concentration in the ore is detrimental to copper flotation, but diminishes with reduced concentration. Every effort is therefore made to eliminate oil (and grease) from the ore.

Blending approach

Owing to limited ore reserves there arose a need to supplement high grade good quality ore from Broken Hill with low grade and tarnished (oxidized) ore to extend the life of mine till the Deeps comes into operation. The options with the tarnished (oxidized) ore were three-fold:

- Not to treat it, which would mean closing the mine
- To treat it separately by batch processing, which was impractical as it would involve having to stop the plant periodically as the mine would not be able to provide sufficient tonnage for the plant to remain running due to underground development work and having to mine various sporadic areas
- Blend the ore, to secure the life of the mine.

The last option thus appeared to be the only long-term solution. The first step in the approach to blending was to identify precisely which ore was problematic. Analytical determination was performed on grab samples and silo samples. The samples were analysed for oxides, cobalt, bismuth, copper, lead, zinc, and silver. Bench test work was carried out under varying conditions of pH, flotation reagent dosing and grinding. On plant scale, the Courier on-line X-ray analyser was used to determine what ore was problematic by monitoring the fluctuations in feed ore grade. More recently, ore types are being identified by virtue of their calculated reactivity number, which is a dimensionless quantity relating the chemical oxygen demand of the sulphide ore to the degree of aeration required. Thus a semi-'realtime' quantitative assessment is almost feasible. Grab samples of the feed ore are also crushed and ground at various levels to determine the impact on flotation in the plant. As a 'tracing' method, coloured metal discs were dropped into stopes that had problematic ore. When the metal detectors on the ore conveyors detected these discs, it signalled the problematic ore early enough to treat it separately in the plant. Once problematic ore was identified, it could be kept separate at the surface ore pad or in the plant ore silos and could be blended at optimal rates as pre-determined by bench test work. However, it must be borne in mind that blending is limited by the mine production schedule. Owing to low availability of ore and large amounts of ore from certain problematic stopes that must be mined in order to ensure the long-term future of the mine, the schedule does not always accommodate proper blending.

Swartberg ore plant trials

All of the plant trials described in this section were preceded by bench flotation test work as a normal matter of course.

Swartberg lower orebody plant trial (1995)

During this plant trial the Swartberg ore was blended with Broken Hill ore at the surface ore pad i.e. prior to primary crushing. A total of 4 176 tons of Swartberg lower orebody ore was treated (December 1995). This represented 13% of the total tonnage treated over the 8-day period.

Table II shows the Swartberg lower orebody ore was lower in magnetite, bismuth and cobalt. The Swartberg ore was found softer than the Broken Hill ore due to the lower magnetite. Flotation results (Table III) were better than target for the plant trial indicating the good quality of the lower orebody ore.

Swartberg upper orebody plant trial (1996)

In January 1996, 13 994 tons of Swartberg upper orebody ore was treated over a two-week trial period. The Swartberg ore was crushed and kept separate, partitioned among the five ore silos and blending was done on the rod mill feed conveyor by manipulating the tonnage via variable speed belt feeders. The Swartberg ore was blended with Broken Hill ore in ratios of (Swartberg to Broken Hill) 20:80 and 40:60 during the plant trial.

Table IV shows that the copper, bismuth, cobalt, magnetite and iron content of the Swartberg upper orebody ore was higher than that of the averaged Broken Hill ore, while the lead, zinc, silver, pyrite and pyrrhotite content was lower. On average, flotation results (Table V) were better than target during this plant trial mainly due to the higher-

Table II

Average plant feed grade during the trial period

	%Cu	%Pb	%Zn	g/t Ag	g/t Bi	g/t Co	%Mgt
13% Blend	0.36	7.15	3.07	103	71	146	28.1
Ave BH 1995	0.41	6.26	2.9	85	74	159	36.1

Table III

Average concentrate grades and recoveries achieved

		Assays						Recoveries	
		%Cu	%Pb	%Zn	g/t Ag	g/t Bi	g/t Co	F/cast	Actual
Copper	13% Blend	28.9	3.35	2.68	1200			71.3	73.4
	Ave BH 1995	27.75	3.27	2.97	852	48	182	70.6	74.1
Lead	13% Blend	0.61	72.5	3.68	893	726		90.4	90.8
	Ave BH 1995	0.47	73.36	3.46	785	765	17	88.5	89
Zinc	13% Blend	0.29	2.02	51.1	65		371	75.4	75.8
	Ave BH 1995	0.37	2.56	51.8	58	35	584	73.7	74.8

Table IV

Average plant feed grade during the 20:80 and 40:60 trial period

	%Cu	%Pb	%Zn	g/t Ag	g/t Bi	g/t Co	%Mgt
20:80 Blend	0.43	6.72	3.08	75	65	183	41.1
40:60 Blend	0.44	6.78	2.77	71	95	201	34.1
Ave BH 1995	0.41	6.26	2.9	85	74	159	36.1

Table V

Average concentrate grades and recoveries achieved

		Assays						Recoveries	
		%Cu	%Pb	%Zn	g/t Ag	g/t Bi	g/t Co	F/cast	Actual
Copper	20:80 Blend	27.3	3.07	2.61	773			73.1	75.5
	40:60 Blend	28.3	2.92	3.98	714			73.1	70.7
	Ave BH 1995	27.8	3.27	2.97	852	48	182	70.6	74.1
Lead	20:80 Blend	0.56	75.2	3.43	726	646		89.6	89.6
	40:60 Blend	0.82	78.4	3.1	671	741		89.6	88.8
	Ave BH 1995	0.47	73.36	3.46	785	765	77	88.5	89
Zinc	20:80 Blend	0.39	2.75	52.3	46		517	74.8	75.6
	40:60 Blend	0.52	3.54	52.4	59		626	74.2	76.1
	Ave BH 1995	0.37	2.56	51.8	58	35	584	73.7	74.8

grade ore from Broken Hill and Swartberg ore treated. However, these high grades are not considered typical of the Swartberg upper orebody. Analysis of subsequent geological samples indicated lower grades could be expected with more contaminants.

Swartberg first bulk plant trial

The first 100% Swartberg bulk ore plant trial was run between June and July 1996 (treating 26 197 tons) to determine metallurgical performance. This trial also needed to establish what process changes were required to treat Swartberg ore on its own, bearing in mind that the concentrator was originally designed to treat solely higher-graded and less-contaminated Broken Hill ore. Table VI shows the measured average feed ore grade with the flotation results achieved during the trial depicted in Table VII.

The magnetic separation circuit and the copper flotation circuit were found to be under capacity in that copper recovery and concentrate grade were both significantly lower and notably high copper contamination of the lead and zinc concentrates was observed. Bismuth content of copper concentrate was abnormally high. Mineralogical studies indicated the high bismuth was due to bismuthinite and cosalite associated with the chalcopyrite. The high lead contamination also increased the bismuth content of the concentrate since bismuthinite is often associated with galena.

Lower lead concentrate grades were due to higher copper and zinc contamination. This, too, is a natural result of the under capacity copper circuit, which had difficulty coping with the higher copper ore grade. The higher zinc contamination of the lead concentrate was attributed to the presence of oxidized copper measured as percentage copper oxide (%CuO). The very high bismuth in the lead concentrate was the most serious problem encountered. The mineralogical investigation that followed indicated intimate inter-growths between bismuthinite and galena. This phenomenon presented a problem that would be complex to solve. There is no regrind circuit for lead since galena is softer than chalcopyrite and sphalerite. Hence, it is virtually impossible to liberate galena from the associated bismuthinite. High lead in the zinc concentrate was due to the unfavourable lead to zinc metal grade ratio of the feed ore. The slight increase in cobalt content of the zinc concentrate was due to the increased amount of cobalt present in the feed ore. Zinc recovery was very low due to the lower zinc head grade and also due to zinc losses to the copper and lead concentrates.

Swartberg second bulk trial

In February 1997, another plant trial was run treating 30 935 tons of Swartberg bulk ore. In summary the tests showed that acceptable grades and recoveries could be obtained

Table VI

Average plant feed grade during the trial period

	%Cu	%Pb	%Zn	g/t Ag	g/t Bi	g/t Co	%Mgt
Bulk S/berg	0.72	4.89	1.18	49	188	168	43.8
Bulk range	0.63–0.89	4.19–5.37	0.96–1.66	43–55	156–228	158–190	39.5–45.7
Ave BH*	0.41	6.26	2.9	85	74	159	36.1

*The average taken was from January to June 1996

Table VII

Average concentrate grades and recoveries achieved

		Assays						Recoveries	
		%Cu	%Pb	%Zn	g/t Ag	G/t Bi	G/t Co	F/cast	Actual
Copper	Bulk S/berg	28.9	7.12	3.23	274	293		74.8	66.2
	Bulk range	26.6–30.7	3.8–12.4	2.0–5.2	216–437	196–387			
	Ave BH*	28.5	3.87	2.87	784	81			
Lead	Bulk S/berg	2.16	68.33	5.83	519	1849		87.6	85.5
	Bulk range	1.2–3.2	61.3–75.3	2.6–11.5	421–681	1472–2630			
	Ave BH*	0.59	73.5	3.64	800	914		89.1	89.2
Zinc	Bulk S/berg	0.91	8.55	47.81			518	73.6	35.1
	Bulk range	0.5–1.2	3.6–12.5	44.2–52.6			411–648		
	Ave BH*	0.43	2.58	51.71			435	74.9	74

subsequent to suitable modifications to the plant. The modifications would relate chiefly to increasing the capacities of both the magnetic separation and the copper flotation circuits. Table VIII shows the comparative feed ore grades.

Copper, lead and zinc losses to the magnetic fraction were notably high due to the higher magnetic flow rate. Table IX shows flotation results. In general, metal recoveries for copper, lead and zinc were lower but could possibly be improved by increasing the capacity of the magnetic separation scavenger flotation cells. The copper circuit flotation results were similar to that of the previous bulk test and again copper and lead concentrate contamination by bismuth was high. Though it is unlikely that this contamination can be reduced, copper grade and recovery could improve concomitantly reducing the amount of misplaced lead and zinc in the copper concentrate if the copper circuit was upgraded.

Lead concentrate grade and recovery was acceptable and while upgrading the copper circuit could reduce copper contamination, zinc contamination of the concentrate was inevitable though not a formidable problem. However, bismuth content of the lead concentrate was much worse than in the first bulk plant trial re-emphasizing the severity of this problem. Detailed analysis of plant results for Broken Hill ore show that up to 80% of the bismuth in the feed ore reports to the lead concentrate. The same was seen to hold true for bulk Swartberg ore in this trial.

The lower zinc head grade of 0.7% largely contributed to the poor performance of the zinc flotation circuit. Although it was conceivable to produce a 50% zinc concentrate, it was equally unlikely that zinc recovery would improve much beyond 35%. A serious problem was the higher cobalt contamination of the zinc concentrate that invokes higher penalties from the smelter owing to the manner in which the sales contract is structured. Analysis of plant data indicates that between 1% and 5% of the cobalt in the Swartberg feed ore reports to the zinc concentrate compared to 15% for Broken Hill ore. Mineralogical investigations show that discrete liberated cobalt grains within the zinc concentrate as well as cobalt minerals attached to sphalerite occur. It is assumed that the recovery of liberated cobalt grains to the concentrate by flotation is poor based on cobalt analysis of the flotation tailings. Regrinding of the rougher concentrate to liberate more cobalt minerals may reduce cobalt contamination. The financial implications of cobalt and bismuth contamination of the zinc and lead concentrates respectively were addressed in a marketing report generated for that purpose at the time.

Broken Hill upper ore plant trials

10% blend trial

In an attempt to model the lead contamination of the copper concentrate as a function of the percentage Broken Hill upper ore (specifically from the 1/1500 stope) fed to the mill, a randomized block trial was planned and executed to obtain other data points. This method was employed primarily to compensate for variations in feed over time but also for all sources of sample error other than the treatment (1/1500). From the data collected it was evident that only a 28.2% probability existed for an average of 0.15% decrease in lead contamination of the copper concentrate, when 1/1500 ore was blended with the other bulk Broken Hill ore from the deeper mining levels. At 20 t/h (or 10% of total feed to plant) the probability is so low that one can conclude that there is no perceivable or significant difference in the lead contamination of the copper concentrate when 10% of the feed to the mill is 1/1500 ore. The main objective of the 10% blend trial was to evaluate any increase in lead contamination of the copper concentrate when:

- no 1/1500 ore was included in the feed ore to the mill ('normal' condition), and
- the feed ore consisted of 10% 1/1500 ore (i.e. 'oxidized' treatment).

Data were extracted from the plant via the Sequel (SQL) database which captures hourly data from the Courier 30XP on-line X-ray analyser. The randomized block design[1] was used to interpret the results of the trial. By virtue of this design, dividing the test results into blocks eliminates the effect of an unwanted yet perhaps significant variable, for instance time series variations. Different levels of the unwanted variable are experienced in each block and by using this methodology variance associated with the unwanted variable can be quantified and removed. All precautions were taken to ensure that meaningful results were obtained in the most efficient manner.

Naturally, the more data collected the less the variance (s^2) that masks the effect of the treatment. However, it is also true that to collect a lot of data is time and resource

Table VIII

Average plant feed grade during the trial period

	%Cu	%Pb	%Zn	g/t Ag	g/t Bi	g/t Co	%Mgt
Bulk S/berg	0.5	4.3	0.7	26	161	190	49.3
Bulk range	0.43–0.59	3.7–5.1	0.5–1.1	21–38	122–191	152–206	40.2–53.8
Ave BH*	0.43	6.14	2.88	84	86	158	33.2

Table IX

Average concentrate grades and recoveries achieved

		Assays						Recoveries	
		%Cu	%Pb	%Zn	g/t Ag	g/t Bi	g/t Co	F/cast	Actual
Copper	Bulk S/berg	24.7	9	6.4	223	448		72.1	69.1
	Bulk range	21.9–27.8	4.5–18.0	4.2–10.4	183–431	282–557			
	Ave BH*	28.1	3.44	2.8	771	82		73	72.8
Lead	Bulk S/berg	1.2	72.3	3	448	2693			81.5
	Bulk range	0.82–2.59	69.4–73.9	2.7–3.5	369–592	1121–4030			
	Ave BH*	0.61	73.7	3.55	798	915		89	89.4
Zinc	Bulk S/berg	0.8	4.1	43.4	52		844		35.1
	Bulk range	0.4–1.36	3–4.7	36.6–50.3	41–63		547–1072		
	Ave BH*	0.44	2.7	51.7	59		483	75	73.6

intensive and therefore care was taken to gain optimal sample numbers by the following method. The general formula for sample size is:

$$n = 2\left(\frac{s.z\alpha}{D}\right)^2$$

where n = number of tests in each group
s^2 = expected variance
D = difference it is required to detect
$z\alpha$ = normal deviate for confidence level $\alpha = 0.01$

This formula is adequate for ordinary paired t-tests consisting of only the probability of the so-called type I error, namely α (i.e. wrongly rejecting the null hypothesis, H_0) but does not make provision for the probability, β for type II error (i.e. wrongly accepting the null hypothesis). The total number of experiments required in each group to detect a difference with type I and type II errors controlled at confidence levels of α and β respectively for large samples is thus:

$$n = \frac{2\left(z_\alpha + z_\beta\right)^2 s^2}{D^2}$$

where $z\alpha$ = normal deviate for confidence level α
$z\beta$ = normal deviate for confidence level β.

The use of this method is strictly speaking not true for small samples where n is less than 150 because when the null hypothesis is false the non-central or skewed normal distribution has to be used. The modified sample size from Davies's tables (1967)[2] can then be approximated by:

$$n_{\mathrm{mod}} = \frac{n}{\left(1.1 + 2.71(\beta / a)a^{1.244}\right)}$$

The result is that seven (7) blocks are necessary to ensure a meaningful and accurate result where:

D = 1.5 (historical increase in lead contamination due to upper ore)
s^2 = 5.88 (analysis of plant year-to-date 1999 actual %Pb in copper concentrate)
z_α = 1.30 (type I error—one sided 90% confidence level)
z_β = 0.84 (type II error—one sided 80% confidence level).

To produce stable and intelligible results a sampling period of twelve hours for each treatment was allowed. The interval between treatments was 3 hours after a 'normal' ore trial (i.e. Broken Hill lower orebody ore) and 6 hours after an 'oxidized' ore trial (i.e. a blend of 'normal' ore with 10% Broken Hill upper orebody ore). The average results over the period could then be used as raw data for a Two-way ANOVA (analysis of variance).

Equal numbers of treatments (or conditions) were made in each block and the order of the treatments was randomized for every block. A coin was flipped for each trial to ensure randomness of the order of 'normal' or 'oxidized' ore within each block. The results thus obtained were analysed by a Two-way ANOVA. Table X shows the order of results over the 10-day period in the Tables that follow.

The data was suitably rearranged to facilitate the requirements for a Two-way ANOVA (Table XI) and statistical parameters were generated for further calculation (Table XII). A Microsoft Excel spreadsheet was used to perform the Two-way ANOVA calculation. The F_{crit} values (Table XIII) are for a 99% confidence level. The F value for the row data at 0.14 was much less than the F_{crit} value of 5.89. Thus, the difference between the averages of the rows (between 'normal' and 'oxide') is therefore not significant at the 99% confidence level. However, it is significant at 28.2% level, but is not considered meaningful in any way. Similarly, the F value for the column data was also not significant indicating that the ore over the test period could be regarded as if being from the same source.

20% blend trial

A secondary attempt was launched a few months later to quantify the effects of blended Broken Hill upper levels ore on the copper flotation circuit. Another randomized block design statistical trial was planned and executed. It was hoped that the data derived here would complement data from the previous trial[3] and assist in formulating some correlation. From the data collected it is 98.8% certain that 1/1500 ore blended with the other Broken Hill bulk ore at 40 t/h (or 20% of total feed) would cause an average of 0.67% increase in lead contamination of the copper concentrate. This represents a relative increase of 24% higher lead contamination. Congruent to the previous trial, the main objective of the trial was again to evaluate the increase in lead contamination of the copper concentrate when:

Table X

Ordered results (copper concentrate)

	Treatment	Concentrate		
		Cu	Zn	Pb
Block 1	norm	24.51	3.27	3.81
Block 1	ox	27.58	2.93	4.13
Block 2	ox	23.12	6.50	3.99
Block 2	norm	29.01	4.21	2.81
Block 3	norm	29.41	3.23	2.20
Block 3	ox	28.55	2.72	2.63
Block 4	norm	29.31	2.63	2.42
Block 4	ox	28.42	2.36	2.10
Block 5	ox	28.45	2.21	2.48
Block 5	norm	29.02	2.58	2.75
Block 6	norm	25.53	2.48	3.82
Block 6	ox	27.47	2.60	3.58
Block 7	norm	27.43	2.89	3.60
Block 7	ox	29.66	3.03	1.45

Table XI

Two-way ANOVA input

	Block 1	Block 2	Block 3	Block 4	Block 5	Block 6	Block 7
Ox	4.13	3.99	2.63	2.10	2.48	3.58	1.45
Norm	3.81	2.81	2.20	2.42	2.75	3.82	3.60

Table XII

Statistical parameters

	Average	Var	Sd
Ox	2.91	0.76	0.87
Norm	3.06	0.44	0.66

Difference between means: $\delta_{avg.}(x)$ = - 0.15%Pb.

Table XIII

Two-way ANOVA output

Anova: Two-factor without replication

Summary	Count	Sum	Average	Variance
Row 1	7	20.37677	2.910967	1.026811
Row 2	7	21.41027	3.05861	0.452918
Column 1	2	7.938787	3.969394	0.053027
Column 2	2	6.807234	3.403617	0.695315
Column 3	2	4.832604	2.416302	0.092564
Column 4	2	4.528152	2.264076	0.050932
Column 5	2	5.232677	2.616339	0.036132
Column 6	2	7.39998	3.69999	0.027295
Column 7	2	5.047604	2.523802	2.304918

Anova

Source of variation	SS	df	MS	F	P-value	F_{crit}
Rows	0.076295	1	0.076295	0.143778	0.71761	5.987374
Columns	5.694489	6	0.949081	1.788533	0.24869	4.283862
Error	3.183887	6	0.530648			
Total	8.954671	13				

- ➤ no 1/1500 ore is included in the feed ore to the mill (‘normal’ condition or treatment), and
- ➤ the feed ore consists of 20% 1/1500 ore (i.e. ‘oxidized’ treatment).

Again, the main response or dependent factor (%Pb in copper concentrate) was monitored simply being logged continually to the SQL database from the Courier 30XP on line X-ray analyser. The absolute deviation of the analyser relative to laboratory analyses for lead in the copper concentrate is less than 5% on average which meant that any deviation between the two could be ascribed to random error.

The F_{crit} values (Table XVII) are for a 99% confidence level (α= 0.01). The F value for the row data at 13.2 is again less than the F_{crit} value of 13.7 indicating that the difference between the averages of the rows (between ‘normal’ and ‘oxide’) is not significant at the 99% confidence level. It is, however, actually significant at the 98.8% level. In other words 98.8% of the time the contamination by lead in the copper concentrate will increase by more than 0.67% when 20% of the feed is 1/1500 ore as opposed to when there is no 1/1500 ore in the feed. The F value for the column data is even more significant and would have swamped the variation due to the treatment if the data were not blocked. It also indicates that in plant conditions variations over time are so strong that any future analysis of plant data as a whole can really only be accurately analysed using this technique or perhaps one more advanced.

For comparison a simple student t-test for independent means was done:

$$\text{Pooled standard deviation}, s_{pooled} = 0.995$$

$$t = \left(\frac{\delta_{avg}(x)}{s_{pooled}\sqrt{2/n}} \right) = 0.362$$

$$t_{crit}(\text{one sided, } \alpha = 0.01, 12df) = 1.81$$

H_0: $\delta_{avg}(x) = 0$ (There is no difference between the treatments)

H_1: $\delta_{avg}(x) \geq 0.67$ (The difference is significant)

The rule states that we must reject H_0 if $t \geq t_{crit}$.

Table XIV

Ordered results

	Treatment	Concentrate Cu	Zn	Pb
Block 1	ox	27.09	3.21	4.66
Block 1	norm	27.99	1.97	4.07
Block 2	ox	26.40	2.27	5.25
Block 2	norm	27.15	3.03	3.80
Block 3	norm	29.94	2.91	1.82
Block 3	ox	27.37	2.58	2.36
Block 4	ox	29.95	2.09	3.18
Block 4	norm	30.07	1.83	3.20
Block 5	ox	29.36	2.14	3.30
Block 5	norm	29.25	2.55	2.69
Block 6	norm	28.09	3.82	1.95
Block 6	ox	29.76	1.85	2.34
Block 7	norm	28.31	3.41	2.33
Block 7	ox	25.16	3.86	3.49

Table XV

Two-way ANOVA input

	Block 1	Block 2	Block 3	Block 4	Block 5	Block 6	Block 7
Ox	4.66	5.25	2.36	3.18	3.30	2.34	3.49
Norm	4.07	3.80	1.82	3.20	2.69	1.95	2.33

Table XVI

Statistical parameters

	Average	Var	Sq
Norm	3.06	0.44	0.66

Average difference: $\delta_{avg}(x)$ = 0.67% Pb

Table XVII

Two-way ANOVA output

Anova: Two-factor without replication

Summary	Count	Sum	Average	Variance
Ox	7	24.58716	3.512452	1.199793
Norm	7	19.86554	2.837934	0.781668
Block 1	2	8.735248	4.367624	0.171406
Block 2	2	9.048608	4.524304	1.059961
Block 3	2	4.180082	2.090041	0.150433
Block 4	2	6.381884	3.190942	0.000386
Block 5	2	5.992846	2.996423	0.190098
Block 6	2	4.289295	2.144647	0.074694
Block 7	2	5.82474	2.91237	0.668489

Anova

Source of variation	SS	df	MS	F	P-value	Fcrit
Rows	1.592408	1	1.592408	13.21392	0.010899	13.74519
Columns	11.16571	6	1.86095	15.44232	0.002049	8.466031
Error	0.723059	6	0.12051			
Total	13.48117	13				

Therefore, we must accept the null hypothesis, H_0 since $t < t_{crit}$. There is no difference in the mean %Pb in the copper concentrate from the evidence collected. This was proven untrue in the Two-way ANOVA as there is no difference between the treatments here because we have not compensated for sample variance due to time progression.

Similarly a single-factor analysis of variance, analysing only the variance between and within the two treatments delivers similar results.

The tested confidence limit is 98.8% using *F* distributions. The single-factor analysis (Table XVIII) shows that even at the F_{crit} of 3.18 corresponding to (1,12) degrees of freedom *df* and a 90% confidence level there is no significant difference since $F < F_{crit}$. In fact, one *can* be 77% confident but this is *not* deemed sufficient. It is clear that 1/1500 ore when fed to the concentrator at 40 t/h (which is 20% of the total feed to plant) increases the lead contamination in the copper concentrate by 0.67% from 2.84%Pb to 3.51%Pb on average, a relative increase of 24%.

As the 20% level already represented a 24% relative increase in lead contamination of the copper concentrate it was clear that separate stockpiling of the 1/1500 ore be continued so that blending could be done to control the contamination level.

Metallurgical results without blending

In January 2000, proper ore blending could not be done due to the low availability of ore. Poor metallurgical results were experienced and an investigation was launched to determine possible causes specifically related to ore types treated. A study of historical data indicated that an increase in the percentage of upper level ore treated especially a relatively high percentage of ore from stopes in 1/1300 and 1/1500 levels. It was known by that stage that ore from these stopes even in inordinate percentages had detrimental effects on plant metallurgical performance. It had also been shown that apt blending of the ore to prevent excessive variation or high percentages of tarnished ore from entering the concentrator assisted the achievement of the desired flotation results. Figures 3 and 4 show clearly that the recoveries of copper (indicative of results as a whole in this instance) went off target at the same time that the pH, a fairly reliable indicator of poor ore conditions dropped from approximately 22 December 1999. This coincides with the date when the first 1/1300 and 1/1500 ore was mined. Handling such high percentages of surface level ore is difficult and inevitably increases concentrate contamination.

Blending strategy/conclusions

Current strategy

It is clear that the flotation results can be optimized for the various problematic ore types with the use of smart instrumentation such as the Courier on-line X-ray analyser and the Proscon 2100 monitoring system. The test work has proved that much better results are obtainable with a consistent feed to the mill by means of proper blending. The strategy from here on is to continue bench test work on ore from suspect stopes. Naturally, the blending strategy will change according to the ore types mined and the availability of ore from the mine. Often mixes of Swartberg, Broken Hill upper levels and hoisted Broken Hill ore are supplied in ratios contrary to those required for blending. At other times, it is possible to keep the various ore types separate, but they may still not be blended optimally because of the higher amounts of problematic ore.

It cannot be over-emphasized how essential good communication between the mining and metallurgical personnel is. A mining schedule that accommodates a blending regime will ensure that in-specification concentrates are produced more consistently.

Future strategy

It is expected that the numerous bench floats will in time yield a useful database from which to plot flotation strategies to be used in the plant. Ongoing development test work relating to on-line ore characterization may also yield feed-forward control loops. These loops can be nested in the expert control system already in its initial stages of use at the plant.

Table XVIII

ANOVA output

Anova: Single factor (Summary)

Group	Count	Sum	Average	Variance
Ox	7	24.58716	3.512452	1.199793
Norm	7	19.86554	2.837934	0.781668

Anova

Source of variation	SS	df	MS	F	P-value	F_{crit}
Between groups	1.592408	1	1.592408	1.607306	0.228918	3.176552
Within groups	11.88877	12	0.990731			
Total	13.48117	13				

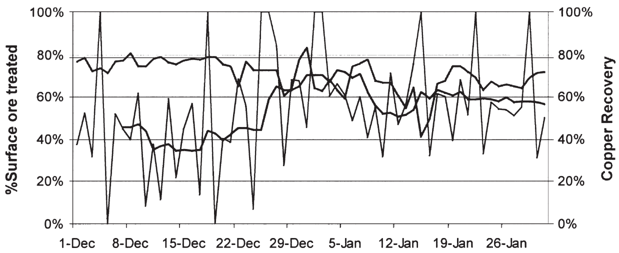

Figure 3—Surface ore as a percentage of total ore treated in the concentator

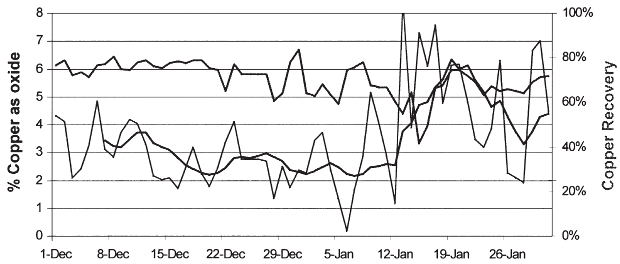

Figure 4—Oxide as a percentage of total ore treated in the concentator

The technical department is developing an economic efficiency control strategy to find the optimum grade-recovery combination for each ore scenario entering the plant. This control strategy combines plant metallurgical performance with the marketing environment in which the mine finds itself. The monetary values of concentrates are calculated by using actual concentrate grades and recoveries combined with metal prices, treatment charges, refining costs, transport costs and penalties. These are compared with forecasted figures based on historical data. The forecast figures represent a 100% economic efficiency. This generates a picture of how the plant is performing on an economic basis. At times when metal prices drop, the economic efficiency can signal the benefit of higher recovery in respect of lower concentrate grade or vice versa depending on how the sales contract is structured. Thus the economic efficiency model can indicate which direction to take during concentrate production as it incorporates the interaction of most of the significant off-mine costs against probable revenue. When fully developed, the model will be used to optimize blending ratios and grade-recovery for different ore types.

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