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The analysis of the mine call factor in gold mining, with specific reference to Western Holding Mine

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5.3.5.2. Vacuuming the footwall of a previously swept and vamped stope.

Objective.

It is generally believed that a significant quantity of gold remains after a mined-out area is passed as swept and vamped. Areas of potential gold lock-up include: on the footwall through poor quality control on passed sweepings; trapped in footwall cracks especially on the up-dip side of a strike gully; accumulated in the packs; reef in/on footwall in open stoping and hanging wall in under-cut stoping operations.

An experiment was conducted to quantify the amount of gold remaining in a stope after it was swept and vamped, and subsequently passed as clean by the Grade Department.

Experimental procedure.

i. Estimate the amount of ore in a stope remaining after it was passed as swept and vamped.
ii. Take representative samples to determine the gold content of the remaining ore.
iii. Remove the ore from the stope using a vacuum cleaner.
iv. Repeat the operation using brushes and scrapers to remove the fines from the footwall.
v. Break up the footwall with a percussion machine, do waste sorting and vacuum the fines.
vi. Determine the cost effectiveness of the operation.

No water was to be used in the area being cleaned up during the experiment.

Experimental results.

A site was selected at Western Holdings Mine 4/47E-30 stope to conduct the experiment(Diagram 2). The stope was mined out on open stoping and passed as swept and vamped. The Basal reef, Steyn facies was mined in the stope. Dry sweepings were practised and the average grade mined in the stope was approximately 20g/t.

On investigation it was estimated by the survey department that the stope contained approximately 60tons of fines at a grade of 30g/t in an area of ± 1 000m². It was already obvious at this stage that the stope did not meet the quality standards to have been previously passed as swept and vamped.

A contractor was hired to supply and install a vacuum cleaner, operate it and retrieve the ore in bags so that it could be transported to the plant for gold extraction.

Vacuum cleaner.

A 55kW liquid ring vacuum pump capable of creating -85kPa, using a 100mm suction hose, was installed in the stope. The collector bin, capturing the ore in 20kg size bags, was advanced to remain within 30m of the suction point, using 100mm HDPE pipes. A dust scrubber was installed

between the collector bin and the vacuum pump to remove the dust liberated during suction. Heat created by the pump was dissipated by means of water which was drained from the working place.

Minimal mechanical breakdown was experienced and the vacuum unit availability and suction performance was acceptable. A maximum of 0.17ton per hour of ore smaller than 100mm diameter was achieved (1755kg in 10 hours, 23 Feb. 1995)(Table 02). The average figure was 0.11ton per hour (21.42ton, 39 shifts @ 5 hrs.) A maximum of 5.34tons was vacuumed during a 12 shift period (6 days).

Description.

Period	**Duration**	**Activity**
Period 1	1995 February 13 to 20	Bulk ore and visible fines vacuumed
Period 2	1995 February 20 to 24	Bulk ore, reef in footwall and visible fines vacuumed.
Period 3	1995 February 25 to March 3	Bulk ore and visible fines vacuumed.
Period 4	1995 March 4 to March 10	Started to vacuum from footwall cracks
Period 5	1995 March 11 to March 16	Fines only, brushed from the footwall
Period 6	1995 March 17 to March 25	Fines only, brushed from the footwall

Period 7	1995 March 25 to April 7	Fines only, brushed from the footwall, and special tests
Period 8	1995 April 8 to April 14	Fines only, brushed from the footwall

Table 5.5. Description of the gold loss experiment.

Results.

Period	**Tons of ore**	**Grade of ore (g/t)**	**Gold content (g)**
Period 1	3.76	24.7	92.87
Period 2	4.2	437 (REEF IN FOOTWALL)	1835
Period 3	4.08	22.8	93.02
Period 4	5.34	29.5	157.5
Period 5	4.04	54.1	218.56
Period 6	2	26.1	52.2
Period 7	3.08	35.61	109.68
Period 8	5.48	41.54	227.64

Table 5.6 Results of the gold loss experiment.

Weighing and Sampling procedure.

Once the bagged ore reached surface, it was transported by truck to the weigh-bridge at Western Holdings plant. The net **weight** of the ore was subsequently determined.

The **sampling procedure** for weeks 1 and 2 were performed as follows: Five bags of ore (100kg) were mixed with a shovel and one sample of 300g was extracted. The remainder of the ore was treated in the plant for gold extraction. The sample was sent to the Assay Department for grade determination.

During weeks 3 and 4 all the ore were crusned after which a 10kg sample from every 100kg of ore was taken to be pulverised. A 300g sample was subsequently taken from the 10kg sample and sent it to the Assay Department for analysis.

Due to the excessive number of samples taken, and assay constraints, the samples were taken randomly as from week 5. One bag was randomly selected from every ten bags of ore. It was crushed and pulverised and a sample of not less than 300g was taken from the aforementioned for assay purposes.

Special tests conducted.

Several footwall slabs were lifted to determine the gold content trapped underneath. The area selected was in block B2, up-dip from the strike gully. The results were as follows:

POSITION	DESCRIPTION	VALUE (g/t)
2G11	Underneath pack gully +0.7 m	1.5
2G1	Gully +0.8 m, 1.0m deep	13.2
2G2	Gully +0.5 m, 1.0m deep	11.4
2G3	Gully +0.7 m, 1.0m deep	8.1
2G4	Gully +1.0 m, 0.9m deep	84.6
2G5	Gully +5.0 m, 0.5m deep	10.3
2G6	Gully +3.0 m, 0.6m deep	trace
2G7	Gully +1.6 m, 0.8m deep	2.2
2G8	Gully +4.0 m, 0.5m deep	6.6
2G9	Underneath pack up-dip of gully	22.6
2G10	Gully +0.2 m, 1.1m deep	4.1
2G11	Gully +0.8 m, 1.0m deep	19.8

Table 5.7. Results of special tests.

The volume of fines associated with these values were in all cases insignificant.

Volume vacuumed in 2mx2m grid.

Block 4 was divided into a 2x2m grid and vacuumed for ±15 minutes. The 100mm suction hose from the vacuum cleaner was split into three separate 25mm pipes so that fines could be vacuumed simultaneously from three

different areas within the grid. The area was vacuumed to a depth of 15cm and waste sorting was done.

The fines were collected into bags for each 15 minute period. The bags were marked and the vacuuming system flushed after each 15 minute cycle. These bags were individually weighed and individually valuated. The results were as follows:

Position	Duration (min)	Weight (kg)	Grade (g/t)	Gold Content (g)
Block 1	15	3.4	13.2	0.0449
Block 1	15	6.9	16.9	0.117
Block 2	20	7.0	4.5	0.0315
Block 4	15	11.6	21.9	0.254
Block 4	15	3.6	9.2	0.0331
Block 8	15	8.4	11	0.0924
Block 8	15	8.2	3.0	0.0246
Block 8	20	12.0	3.8	0.0456
Block 11	15	8.9	5.6	0.05

Table 5.8. Grade results of individual vacuumed blocks.

It is obvious from the aforementioned that gold is not concentrated in the cracks of this particular stope.

Grade distribution in the stope.

With reference to Table 77 to 79, the results were as follows:

Period	No of samples	Average Grade (g/t)	Standard Deviation	Variance	Highest and Lowest value (g/t)	Tonnage
1	38	24.81	10.45	109.3	59.2 / 5.7	3.76
2	27	437	487	237592 *	2251 / 13.8	4.2
3	78	22.8	18.28	334.4	88.4 / 1.6	4.08
4	77	29.5	8.48	72	57 / 14.6	5.34
5	40	54.1	30.96	958	197.6/ 17.2	4.04
6	35	26.1	30.02	901	99.5 / 0	2
7	19	35.61	13.3	176.9	58.5 / 11.4	3.08
8	38	41.54	14.76	217.9	94.5 / 13.1	5.48
Total	352	62.7	174	3027.9	2251 / 0	31.9

Table 5.9. Grade distribution.

* Reef in footwall was found.

Conclusion.

i. The fines that remained in this stope was largely due to unacceptable mining practice and not due to enrichment of fines in the footwall cracks or packs.

ii. Dry sweepings was practised in this stope, obviously without the correct tools like brushes and scrapers being used.

iii. The sweepings in this stope should not have been passed by the Survey Department.

iv. Reef left in the footwall (n : picked up in previous records) accounted for the majority of the gold recovered from this working place. (± 66 percent = 1.8kg)

v. Gold does not seem to concentrate in footwall cracks as popular belief suggests. The experiment conducted in a carbonaceous stope where a waterjet was used proved the same. To date no place has been found where this supposed concentration of gold has taken place.

vi. Although an amount of approximately 2.5kg of gold in a volume of approximately 30ton of ore was vacuumed in an area of 660m², the project at best covered it's cost (Table 80&81). A higher volume of extraction is needed to make it viable. It is suggested that "high volume" of approximately 1ton per hour at a grade of at least 30g/t must be vacuumed to be profitable.

vii. If it assumed that a 10 hour effective vacuuming shift is obtained per day and productivity remains at 0.1ton per hour, the average grade required is approximately 60g/t recovered. At this stage it

seems that if it was not for the reef in the footwall that was extracted, this project would have made a loss.

viii. The grade of footwall fines to be vacuumed can be expected to be approximately 150 percent of the original face value. The amount of fines vacuumed was approximately 0.048t/m2.

Western Holdings cost per month	R 6 960
Contractors cost per month	R 36 000
Total cost per month	R 42 960
Say	R 43 000
Current gold price (R/kg)	R 43 000
Break-even gold (g)	1 000
Current vacuum volume (ton/month)	17.38
Break-even grade (g/t)	57.5

Table 5.10. Break-even analysis.

5.3.6. Western Holdings Mine 2 shaft gold reconciliation experiment.

5.3.6.1. Background to the problem.

2 Shaft was performing unacceptably from a profit viewpoint for a number of years. Mining took place in the Basal reef horizon in the Steyn Facies. The reef dips from west to east stretching from 22 to 52 levels. The shaft stretches over a length of 3.5km north/south. Approximately 1

700 persons were employed at the shaft which had a production target of 9 000m² during 1994 which has been reduced to 6 300m² in September 1995. The shaft was facing imminent closure as from October 1995. The reason for the aforementioned was that it was responsible for a R2 million loss monthly which could no longer be sustained.

The lower portion of 2 Shaft has steep stopes dipping at +37°, with a layer of shale above the reef at a thickness of up to 2m. Besides the fact that steep mining is challenging, the reef also had to be undercut to make the area payable. Mining in this area ceased during 1995 after operating it unprofitably through a sub contracting firm.

A qualitative analysis done by the mine's Geology department on the shale found at No.2 Shaft, using a "PIMA" (portable infrared mineral analyser), revealed that the dominant phylosillicates present were pyrophyllite, sericite and rectorite. The highest level of pyrophyllite was found in the shaft pillar area which was being removed. It was thought that pyrophyllite had an adverse effect on the gold recovery process. The reef mined at 2 Shaft pillar was of a carbonaceous nature where the gold is concentrated mostly in the contact of the reef band.

The Mine Call Factor decreased wherever No.2 Shaft's ore was sent to be processed. No.2 Shaft sent its ore traditionally to the Welkom plant but during 1993 it was decided to send the ore to Western Holdings plant as it was expected that an improve Mine Call Factor would be obtained. The Mine Call Factor decreased in that plant and increased in the Welkom plant (Graph 4). It was then decided to redirect 2 Shaft's ore once again

to the Welkom plant. The Mine Call Factor dropped again and put 1 Shaft under financial pressure as well, as the gold apportionment was based on gold broken. It was then concluded that the ore of No.2 shaft had a detrimental affect on the metallurgical process.

5.3.6.2. Experimental procedure.

As from 8 September 1995 to 23 November 1995, the ore from No.2 shaft was sent by rail to a independent milling circuit in the President Brand carbon-in-pulp plant. The objective was to determine for once and all whether this ore affected the metallurgical process in the plant.

The ore mined underground was delivered to surface where the tonnage was measured at the shaft head using a belt-weightometer. The ore was delivered was sampled using the go-belt sampler which was mounted on the conveyor belt. The sampling interval was set at 50ton to ensure that a statistically significant result could be obtained. The resultant product of tonnage and grade were taken as the gold hoisted and therefor as the **gold called for** from the plant.

The ore was transported by rail to the plant where the number of hoppers was accounted for and a hopper tonnage factor applied for auditing purposes. The ore was tipped onto a belt where it was weighed again and subsequently tipped into a 3 000ton capacity silo. The silo delivered onto another belt which fed an independent milling circuit. The tonnage throughput was once again weighed using a belt-weightometer.

Samples of the pulp delivered from the milling section were taken by means of 4 cutters in the pulp stream, before the milled product was delivered to the thickeners. These samples taken were used to determine the pulp grade. The **gold accounted for** was determined as the product of the pulp grade and the tonnage milled.

The **gold called for** and the **gold accounted for** were to be compared to determine whether a significant 'loss' occurred between the shaft and the plant.

5.3.6.3. Results.

Approximately eighty-five thousand tons (85 000tons) of ore was hoisted to surface during this period (Table 85, Graph 96). Six hundred and eighty-eight, 50kg samples were taken with the go-belt sampler, mounted on the surface conveyer belt, to determine the grade delivered to the plant. The grade of ore delivered amounted to 6.53g/t. The total amount of **gold called for** from the plant totalled to 555.9kg.

The **gold accounted for** in the President Brand plant amounted to 636.3kg which was 80.4kg (+14.4 percent) additional to that called for from the shaft (555.9kg). The tonnage milled amounted to 82 414tons which was understated by 2 759 tons (-3.2 percent) when compared to that measured at No.2 shaft. The average pulp grade amounted to 7.72g/t (Graph 98).

Both the grade distribution of the go-belt and pulp samples amounted quasi log-normal distributions (Graph 99&100), which is similar to that of

the grade in situ underground. This relationship can also be seen on the No.2 shaft go-belt sampling grade distribution over the period October 1995 to January 1996 (Graph 55).

5.3.6.4. Conclusion.

The experiment lasted for a seventy-five day period and conclusive results were obtained.

The **gold accounted for** in the metallurgical plant amounted to 636.3kg during this period exceeded the **gold called for** at 555.9kg (Graph 97). The results indicated that the gold hoisted contained more than the gold estimated as delivered. It was therefore concluded that the ore from No.2 shaft did not detrimentally affect the metallurgical process as was previously believed to be the case. This experiment saved the shaft from closure in the short term.

Analysis of the results indicated that the tonnage discrepancy increased throughout the experimental period and assisted to re-establish No.2 shaft as a profitable concern (Graph 101&102). The theoretical hoisted grade determined during the experimental period ranged from 7.82g/t to 8.94g/t. The final pulp grade amounted to 7.22g/t and the shortfall of 0.6g/t and 1.7g/t is unaccounted for. In this instance a positive tonnage discrepancy was included in the total tonnage hoisted. Although during this experiment it was concluded that the gold is contained in the hoisted grade, the additional tonnage hoisted attenuates the problem of the underground **theoretical gold loss** (Table 5.11).

	Sept 95	Oct. 95	Nov. 95
Tons Hoisted	34 322	34 056	32 265
Discrepancy	180	2958	2728
% Discrepancy	0.5	9.5	9.2
Au content(g)	1 524	23 131	24 470
Au called for(g)	288 090	307 100	303 780
% difference	0.53	7.5	8.1

Table 5.11. Influence of positive tonnage discrepancy on gold called for.

CHAPTER 6.

CONCLUSIONS AND RECOMMENDATIONS.

The following conclusions and recommendations were discussed in the previous chapters:

6.1. Apparent Gold Loss.

6.1.1. In-situ sampling.

The rudimentary method used during in-situ sampling of the underground panels needs to be improved upon. The use of a compressed air-powered 100mm diamond blade angle grinder is recommended to cut the outer perimeter of samples. Experimentation to compare the effectiveness of the aforementioned and conventional chip sampling proved fruitless as the grades are extremely variable in the x,y and z planes. It was, however, concluded by underground observations that diamond blade cutting was more accurate. Western Holdings Mine will continue to introduce these units to ensure more accurate sampling methods.

Current indications are that the gold in-situ is over-estimated which results in an **apparent gold loss**. A task group has been formed under the auspices of the Anglo American Group Surveyor to further investigate possible solutions to this problem.

6.1.2. Specific gravity.

Experimentation indicated that the specific gravity of the ore at 2 780kg/m³ should rather be in the vicinity of 2 700kg/m³. The result of this over-estimation is that an **apparent gold loss** is created in that the grade content of the ore is over-estimated and theoretical tonnage is created.

The grade as determined by skin-sampling should be stated at a specific density and not be assumed to be the same as that of the surrounding rock. When the sampled grade is extrapolated to the remainder of the stoping width, the sampled grade must be adjusted for the density of the surrounding rock to ensure that the grade estimate is accurate. In the instances cases where the specific gravity of the sampled portion and the remainder of the ore is at different densities the grade content will be over-stated. This over-estimation of the grade in-situ will be biased towards reef types where the gold content is restricted to narrow depositions such as carbonaceous reefs. In this instance the density of the sampled portion is lower (i.e. ±2 663kg/m³, Table 1.2) than the normally assumed 2 780kg/m³ of the remainder of the rock, resulting in the over-estimation of the grade (4.4%). In the instance of bigger conglomerate reef types, such as Leader reef, this problem will be minimised as the density of the sampled portion and that to which it is to be extrapolated to are similar.

It remains for the mines to introduce the reduced specific gravity to decrease the **apparent gold loss** which can amount up to approximately 7.5 percent (i.e. 2 663/2780 grade, 2700/2780 tons, combination = 7.3%) of the **theoretical gold loss**.

6.2. Real Gold Loss.

The legendary gold accumulations due to previous losses underground were found in insignificant proportions underground.

6.2.1. Gold in footwall cracks.

Gold accumulations in the footwall cracks were found in insignificant quantities. It was found uneconomical to remove from underground. Vacuum cleaners were used to vacuum the areas in question. It was also attempted to seal a gully using a cementicious based product. Although the seal withstood the abrasion of the rocks and scrapers remarkably, it did not appear to make any significant difference to 'gold loss'.

6.2.2. Sweepings.

Gold remaining on the footwall due to sub-standard dry sweepings was found to be uneconomical to recover. It is suggested that sweepings per sé is merely an indicator that most of the ore was removed from the previously worked areas and that the gold is not enriched t, disproportionately in the fines.

6.2.3. Reef in hanging and footwall.

Previously unaccounted for reef remaining in the footwall was found during an experiment. This type of gold loss is avoidable and is classed as 'dirty mining'. A similar occurrence of reef remaining in the hanging wall after a panel was undercut, is also avoidable. The accounted for areas where this type of avoidable gold loss occurs is of a relatively small proportion.

Reef remaining in the hanging wall due to undercutting of some stopes has been found but it also does not explain the real gold loss to date.

6.2.4. Waterjets.

Waterjets does increase the productivity underground, provided that the associated infra-structure is adequate. It was found that the use of waterjets does not have a significantly different influence on gold loss when compared to conventional mining. It was observed in underground situations that the footwall was significantly cleaner with the use of waterjets when compared to a stope where dry sweepings was done.

A vacuum cleaner was used to vacuum the footwall of a stope where a waterjet was used, but an insignificant volume of fines were found. It was surprising to find during this operation that the grade of the fines accumulated in a gully of a stope where a waterjet was used, was 50 percent of that mined in the stope.

6.2.5. Mud.

The grade of mud remaining underground in the footwall infrastructure was found to be similar to that of the ore mined in that particular area. The loading of mud will make a difference to the **gold accounted for**, only if it is done in significant volumes in relation to the tonnage mined.

6.2.6. Other gold losses in stopes.

Several experiments were conducted in stopes to account for all the gold from the blasted face to that finally delivered in the hoppers. In just about all the cases, approximately 30 to 40 percent of the gold was unaccounted for by the time it reached the hoppers. The unaccounted for gold could not be found in the stopes. Although the method of grab sampling is not an accurate representation of the values assigned to underground panels, it can still be used as a diagnostic tool. The errors associated with this method remains mutually inclusive when used for comparative purposes. Notwithstanding the aforementioned facts it is suggested that a portion of the **gold called for** was probably not there in the first instance. The estimation of in-situ grade remains inaccurate as explained under 6.1.

6.2.7. Gold loss in the metallurgical plant through residues.

It is a most convenient option to blame the metallurgical plant when a low Mine Call Factor is achieved. However, if the residue values are

measured correctly, it cannot have an influence on the Mine Call Factor at all. (Although this is a well known fact, it is not always understood by all when a poor MCF is prevailing). This is the case as the gold-in-residue is part of the **gold accounted for** equations. The metallurgical process in a plant will dictate what residue values are acceptable to that particular plant.

Gold theft was not considered in this thesis.

6.3. Other Recommendations.

6.3.1. Tonnage from old areas.

It is recommended that at least until the estimation of grade in-situ has been improved upon, additional tonnage from old areas with sufficient grade must be extracted at Western Holdings Mine to make up for **unaccounted for** gold being experienced. This strategy was practised for the period April 1989 until October 1994 when it was scaled down (Graph 2). The result was a drop of almost 10 percent in the Mine Call Factor.

6.3.2. Gold allocation through go-belt sampling.

Go-belt samplers and weightometers were installed at all the Freegold shafts. It has been in operation on Western Holdings Mine since October 1995 and most of the problems were resolved. Analysis of the sampling

results has indicated that stability has been reached and that gold produced can be allocated to the individual shafts using this method.

This method is certainly different when compared with the gold produced allocation based on gold broken because the tonnage hoisted is not taken into account. In the periods of financial survival, it is a mining strategy to hoist ore broken from old areas in addition to that broken from current mining. The gold from the old areas is not **called for** but it does make a positive contribution to the gold produced. The allocation of the gold produced must therefore take into account the areas from which this additional effort stems. Gold produced allocation using the go-belt samplers and weightometers therefore is the correct method and will ensure that focus is placed on the deserving profit centres.

6.4. Conclusion.

There are no mystical connotations to gold loss as the gold loss does not take place in obscure ways. A consistent error has been introduced in the estimation of **gold called for** as the density of the different types of rock is not taken into account at some mines and results in **apparent gold loss**. The **real gold loss** primarily occurs because the ore blasted is not all removed from underground to the metallurgical plant.

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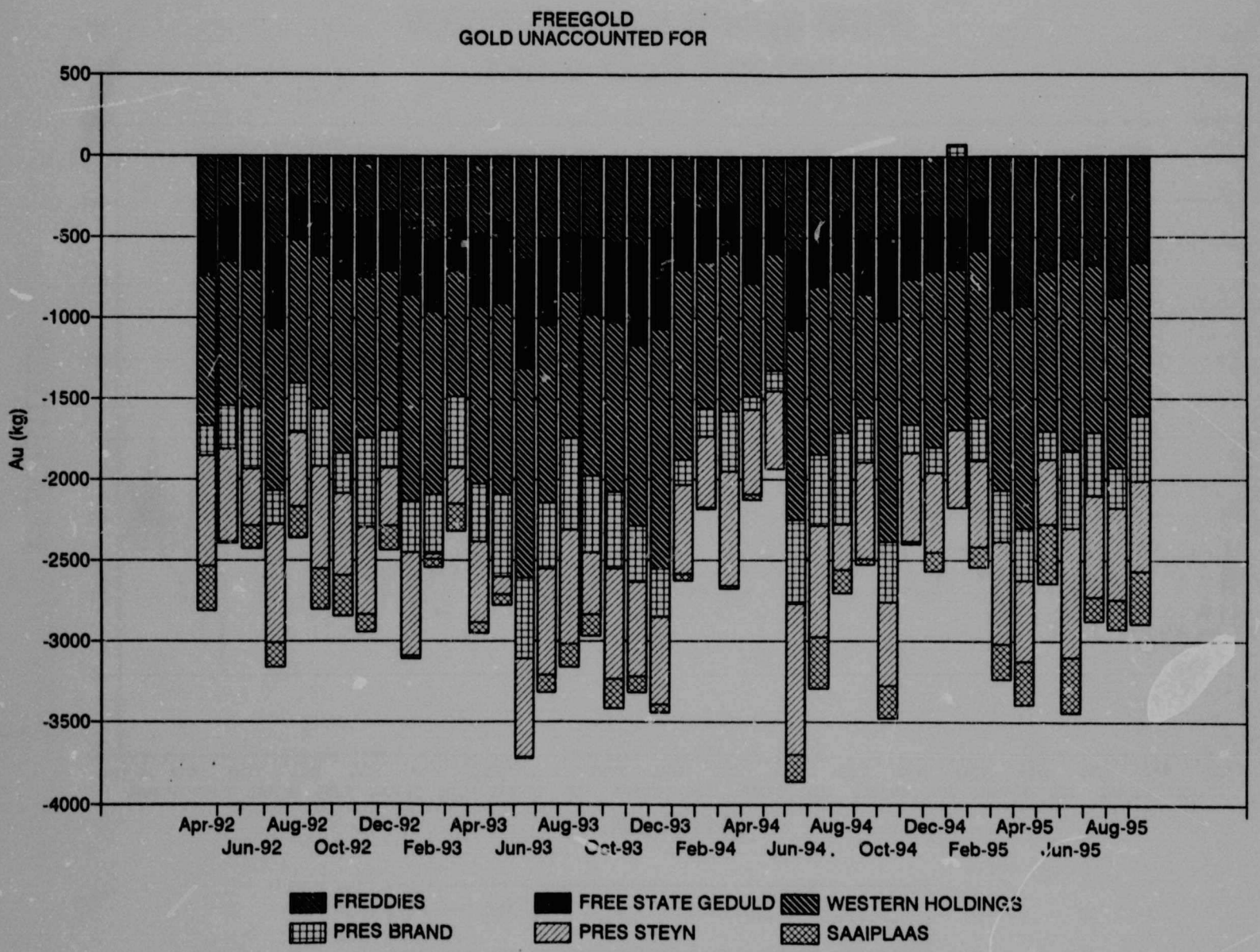
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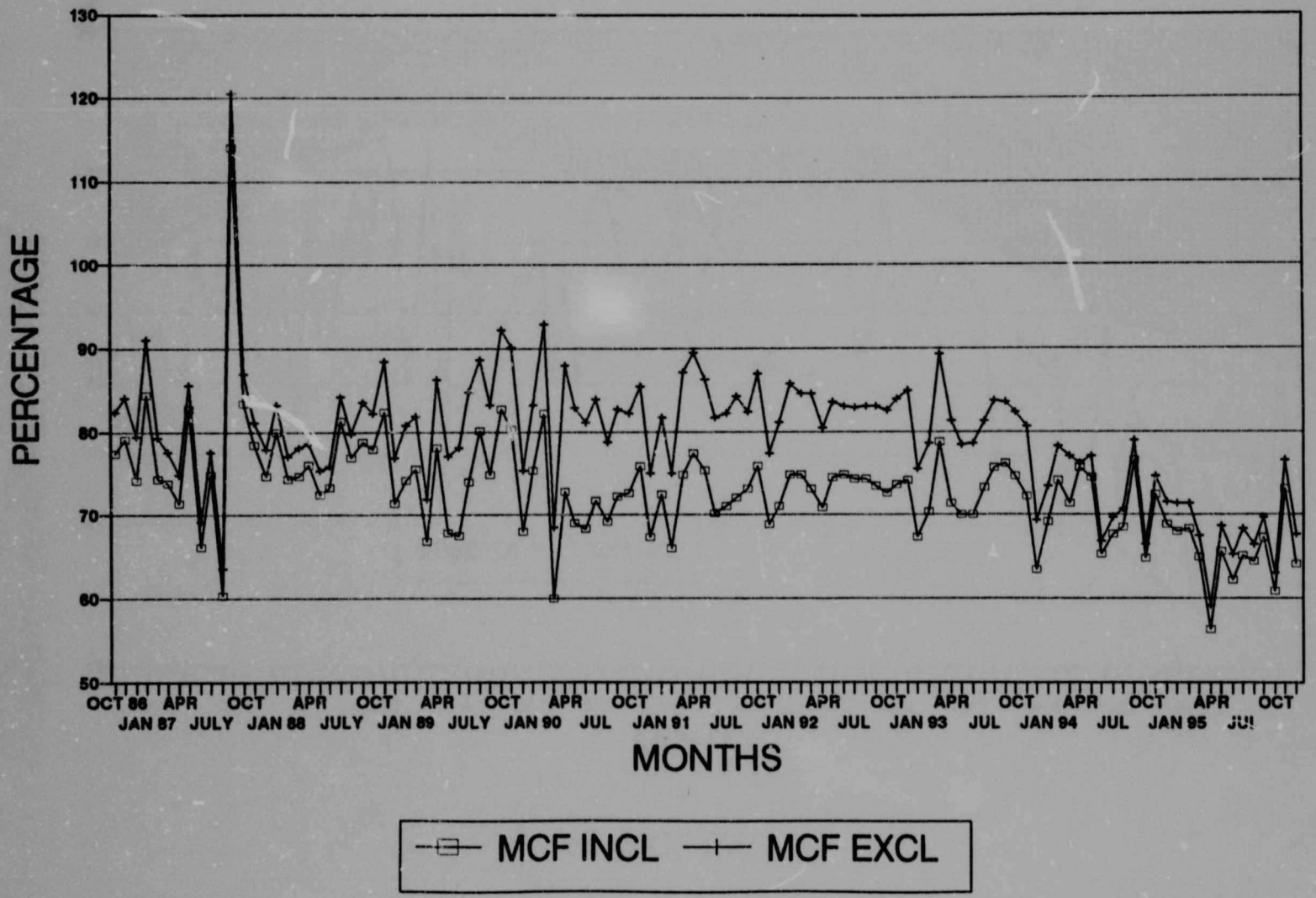
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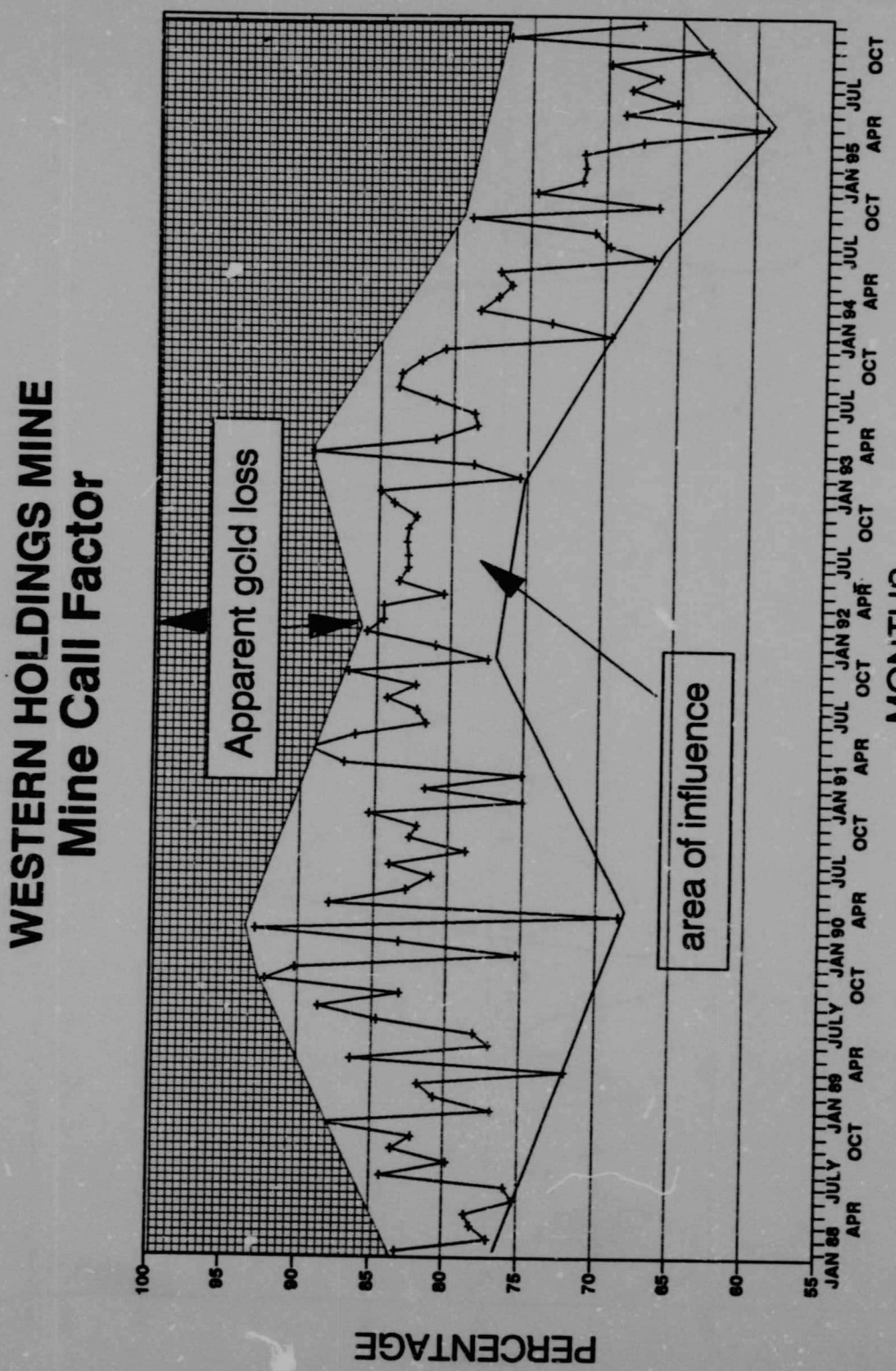


Graph 1. Freegold, gold unaccounted for. April 1992 to September 1995.

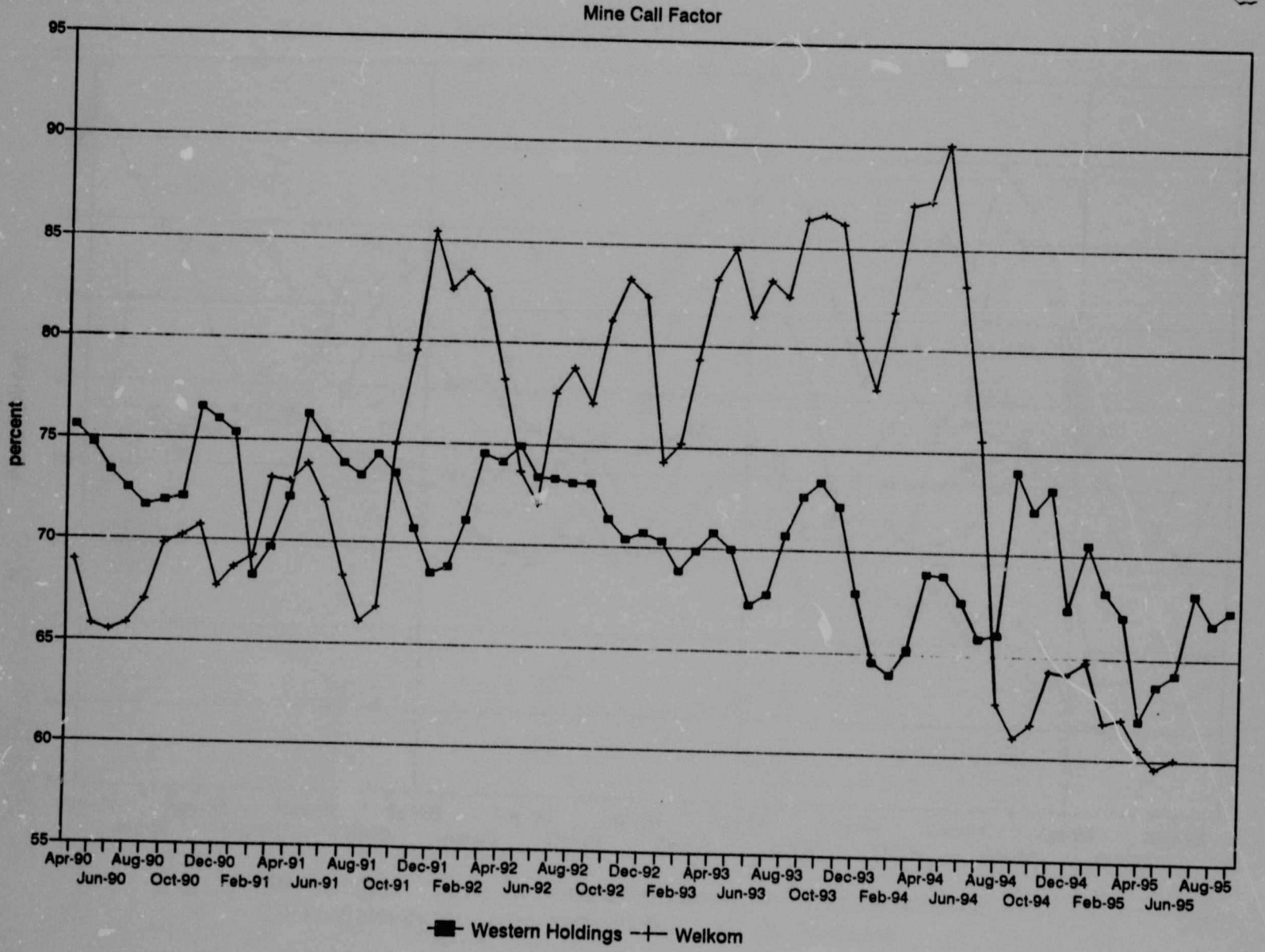
WESTERN HOLDINGS MINE
Mine Call Factor



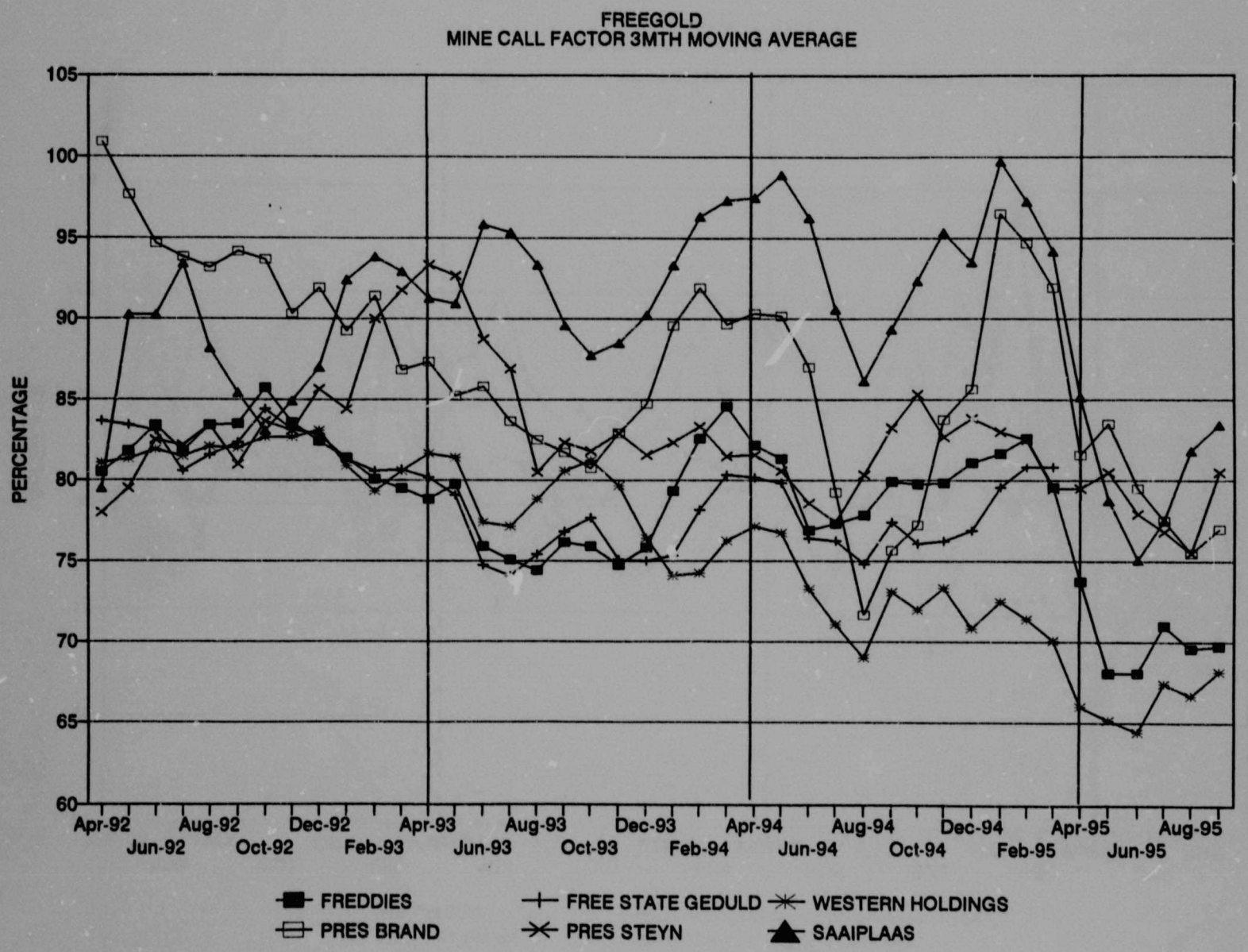
Graph 2. Western Holdings Mine. Mine Call Factor ,1986 to 1995.



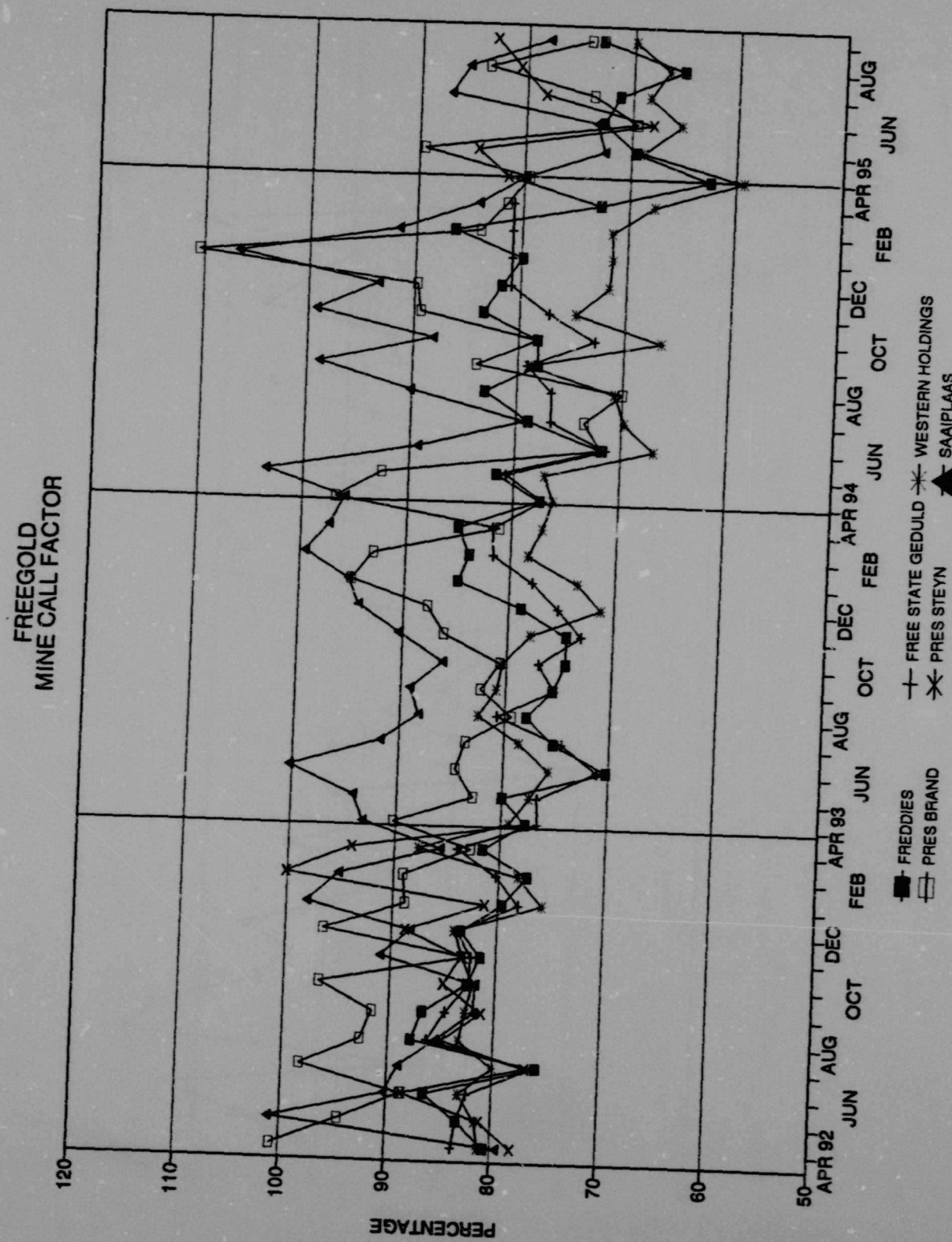
Graph 3. Western Holdings Mine. Mine Call Factor , 1988 to 1995.



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Graph 5. Freegold. Mine. Mine Call Factor (3mth ave), 1992 to 1995.



Graph 6. Freegold. Mine. Mine Call Factor, 1992 to 1995.

Author De Jager E J
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