PIT OPTIMIZATION & DESIGN OF THE NOLANS GOLD PROJECT

Mark Moddejongen[1.] & Peter Bourke[2.]

ABSTRACT

The Nolans development, a joint project between Carpentaria Gold Pty Ltd (CGPL) and Haoma Mining NL, is situated at Ravenswood in Northern Queensland. The Interim Phase Stage 1 pit which was completed in March 1995, will be followed by Phase 2 which comprises the Stage 2 and 3 pit designs. These subsequent stages were designed with the aid of Whittle 4D and Microlynx.

The Whittle 4D package was used to optimize the LOM pit. The operating profit and ore tonnages were determined for each pit shell. The sensitivity of the pit to changing operating costs and gold prices was examined. The effect of varying the cutoff grade was also analysed. A pit shell was then selected upon which the detailed pit design was based. The detail design of the Stages 2 and 3 pits was undertaken using Microlynx.

INTRODUCTION

• GENERAL

The Nolans Gold Project is undertaking the mining of oxide and primary ore on the Nolans and Sarsfield Leases which are near Ravenswood in North Queensland (Figure 1).

The Interim Phase pit involved the mining and heap leaching of the shallow oxide ore which extended to approximately 20m below the surface. There was also some primary ore mined below this to provide feed for the existing processing plant.

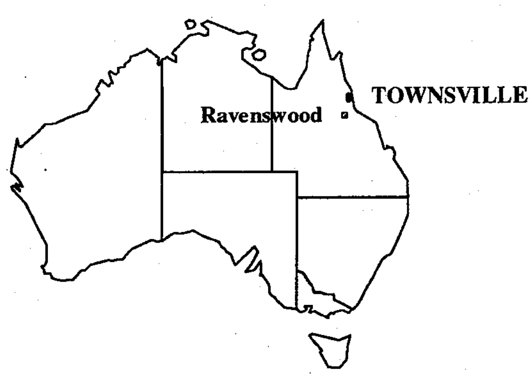

Figure 1: **Locality plan**

1. MARK MODDEJONGEN
Qualifications: BEng (Mining), University of NSW, Australia.
Membership: AusIMM.
Experience: Capricorn Coal Management, German Creek Mine; Ulan Coal Mine; White Mining Ltd; Exxon Coal & Minerals Australia Ltd; Poseidon Gold, Mount Leyshon Gold Mine (Australia).
Currently: Senior Mining Engineer, Dames & Moore Mining Services, Queensland, Australia.

2. PETER BOURKE
Qualifications: BEng (Hons) Mining, University of Queensland, Australia. Queensland First Class Mine Manager's Certificate (Metalliferous).
Membership: AusIMM.
Experience: Mount Isa Mines Ltd and MIM Holdings (Australia).
Currently: Mine Superintendent at Ravenswood, Carpentaria Gold Pty Ltd, Australia.

This paper examines the pit optimization and design of Phase 2 which started in March 1995 and extends to the end of the mine's life in 2001. Prestripping, tailings dam and stockpile construction, and some ore mining will be undertaken between the start up of Phase 2 and the commissioning of a new 2 Mtpa processing plant in October 1995. From September 1995 the mine is designed for an annual ore production rate of 2 Mt.

• OBJECTIVES

The objective of the pit optimization and design work was to :

- Optimize the pit & establish the final pit limits
- Examine the sensitivity of the pit to changing input parameters
- Design the LOM pit
- Devise a staged development plan for Phase 2
- Create a LOM production schedule
- Extraction of 12.5 Mt of primary ore

GEOLOGY

The host rock at Nolans is a Tonalite. This is a granular rock with black and white mineral grains of between 1 and 5 mm diameter. Its speckled appearance lead to it being called "spotted dog" by the local miners of 100 years ago. It contains the minerals Plagioclase, Quartz, Biotite and Hornblend. The Tonalite is extensively and weakly altered with localised areas of intense alteration. Alteration minerals include sericite, chlorite, silica, muscovite and epidote.

There is little outcrop, but the intrusives have been mapped in cores, in the test pit and with the aid of geophysics. The gold mineralisation is deep epithermal or high level mesothermal in nature, the predominant structural controls on mineralisation are sets of south dipping veins; there are two sets:

- The first strikes 80° (grid) and dips on average 20° to 30° south.
- The second strikes 110° (grid) and dips on average 20° to 30° south.

Although one or the other set is frequently better developed in local areas of the deposit, both are normally present. The south dipping veins are narrow, ranging from a few centimeters up to 1.5 metres wide. They generally have mineralised selvages, which are usually only a few centimeters wide, although they may be up to 10-15 metres wide in some places. The gold grade, which may be quite high in the veins (from 15 g/t to over 100 g/t), is generally less than 1 g/t in such marginal alterations. Most of the gold is contained in the south dipping veins and the system can be considered as an open stockwork.

The upper part of the deposit (typically to 12 metres) is completely oxidised, with an average density of 2.1. At a depth of approximately 12 to 20 metres lies a partially oxidised zone, which consists of oxidised veins in unoxidised host rock, and has a density of 2.60 t/bcm. The primary fresh ore lies below 20 metres and has an average density of 2.78 t/bcm.

PIT OPTIMIZATION

• MODEL PREPARATION

A grided block model of the deposit was supplied by CGPL. It was loaded into Microlynx then exported into a Whittle 4D model. The optimum pit in the initial 4D analysis indicated that ore could be extracted up to the western limit of the model at 14100E. In order to facilitate the development of this end wall the model was extended by adding extra blocks to the west. These additional blocks were considered to be waste.

The revised model had the following parameters:

♦ Eastings	14025 to 15000
♦ Northings	13400 to 13750
♦ Levels	80 to 315
♦ Block sizes	12.5 x 10 x 5 m
♦ Number of blocks	78 x 35 x 27

The model was adjusted by CGPL to ensure that all the blocks above the Stage 1 pit were air blocks.

The following variables were imported and stored in the model:

- Grade
- Density
- Block tonnage

• INPUT PARAMETERS

The following CGPL supplied parameters were used in the Whittle 4D setup:

♦ Metallurgical recovery	95%
♦ Mining recovery	95%
♦ Overall recovery	90.3%
♦ Cutoff grade	0.70 g/tonne
♦ Gold price	A$500/oz

The processing and contract mining costs were supplied by CGPL.

The CGPL supplied draft pit design was used as a guide to devise the Whittle 4D wall structure file. The pit was divided into four sub-regions in order to approximate the position of the ramp. This facilitated the fitting of the pit design to the Whittle perimeters. The regions were assigned slopes of 42, 48, 54, and 54 degrees respectively.

• OPTIMIZATION RESULTS

A total of three pit optimizations were run. In each run a total of 52 pit shells were examined. The maximum profit pit shell (#34) in Run 3, had approximately 12.7 Mt of ore at a grade of 1.73 g/t. A plot of the operating profit and the head grade versus the ore tonnage is given in Figure 2.

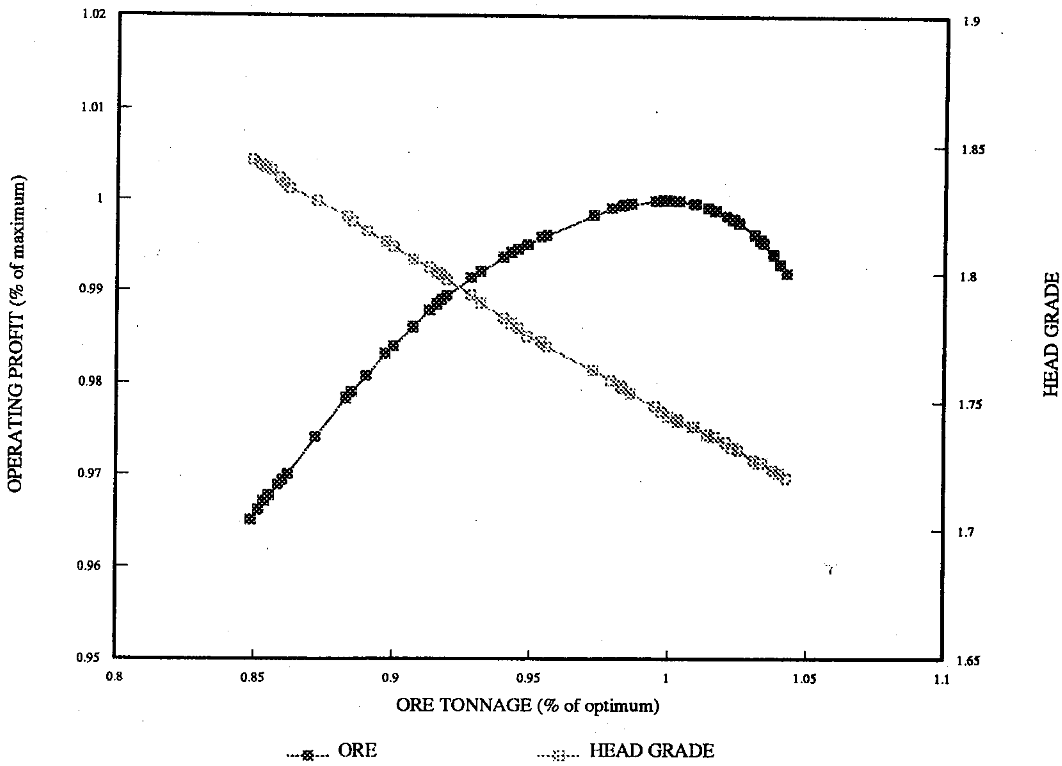

Figure 2 Optimum Pit Shells

As some oxide ore was included in the optimum pit tonnage, a slightly larger than optimum pit shell was chosen so that the objective of approximately 12.5 Mt of primary ore could be achieved. This pit shell (#44) had approximately 13.0 Mt at a grade of 1.73 g/t

A 5 m contour plan of the Whittle 4D bench plan was created in Microlynx. This was then used to develop the LOM pit design using the CGPL design parameters.

- **CUTOFF GRADE**

A CGPL nominated cutoff grade of 0.70 g/t was used in the pit optimization. In order to examine how this could vary another Whittle 4D optimization run was carried out. It was considered important to have a detailed cutoff grade strategy in place before mining started to ensure proper management of the low grade stockpile.

With a non-restrained cutoff grade the optimum pit had an extra 762 kt of ore and 15,743 oz of contained gold compared with the restrained pit shell. The average grade of the additional ore was 0.61 g/t.

The Whittle 4D derived cutoff grade was 0.49 g/t. This is very close to the CGPL nominated subgrade cutoff of 0.50 g/t.

- **SUBGRADE**

The subgrade ore (i.e. less than cutoff grade of 0.70 g/t) was included in the waste tonnage. It was however calculated for both the Stage 2 and 3 pits. There was 1.423 Mt of subgrade at a grade of 0.532 g/t

• GOLD PRICE SENSITIVITY

The sensitivity of the optimum pit shell to varying gold price was examined using Whittle 4D. A price range between $500 to $675, in $25 increments, was examined. A plot of the ore tonnage and head grade for the various gold prices is shown in Figure 3.

A 35% increase in the gold price from $500 to $675 per oz, would raise the operating profit by 68% and the ore tonnage by 7%, and result in a 3% lower head grade.

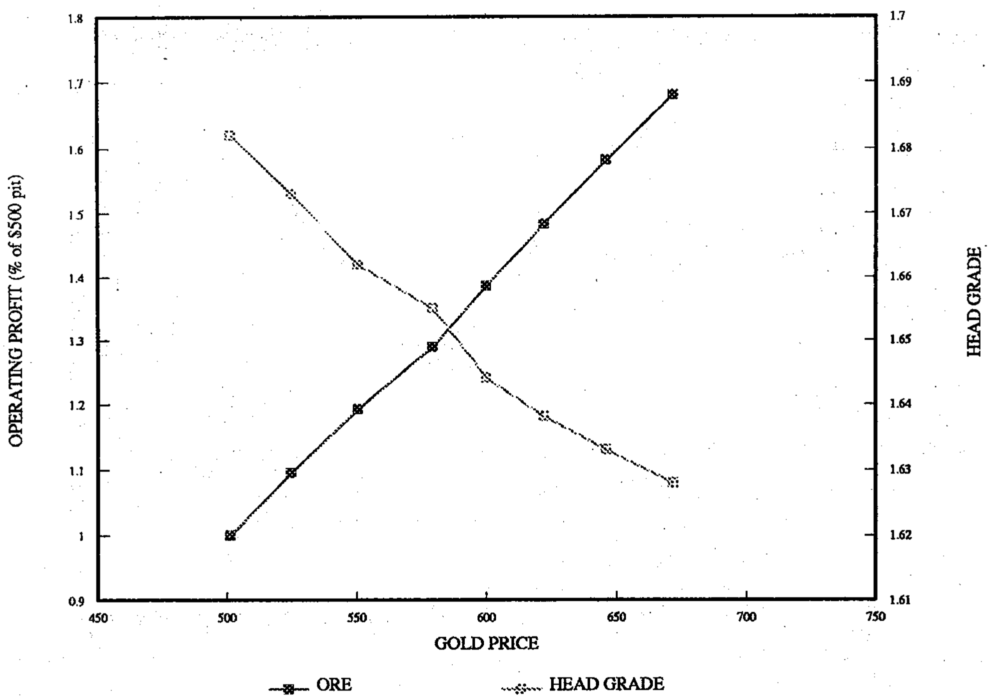

Figure 3 Gold Price Sensitivity

PIT DESIGN

• DESIGN PARAMETERS

The following CGPL supplied design parameters were used:

- Interberm Slopes
  - 50 degrees above RL 295
  - 60 degrees between RL 285 and RL 295
  - 70 degrees below RL 285
- Berm width of 6 m.
- Berms at RL's 295, 285, 265, 245 and 225.
- Haul road width of 20 m, and a grade of 1:9.

• LOM PIT DESIGN

The LOM pit was designed using the Whittle perimeters and the draft CGPL plan as guides. It blends in with the existing Stage 1 design which is shown in Figure 4. The pit has a clockwise rotating haul road that starts in the southern wall, and finishes at RL 200 in the northwestern corner.

The pit has approximately 12.7 Mt of ore at a grade of 1.72 g/t. This differs from the Whittle figures because the detail design has to smooth the pit walls, and to take into account the haul roads and berms more precisely than the Whittle wall structure file could accommodate. The pit was also shortened in the west to leave sufficient room between the existing pit on the adjacent lease.

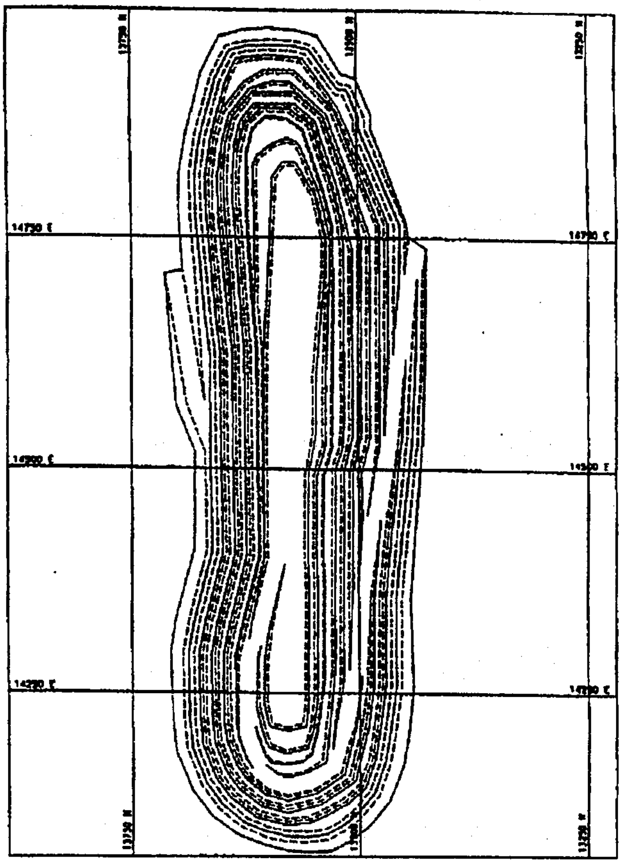

Figure 4 LOM Pit

• STAGE 2 PIT DESIGN

After the first schedule was completed it was found that an interim pit design, fitting between the Stage 1 pit and the LOM pit was required. One of the advantages of the interim pit was that it allowed mining of the low grade ore in the cutback at the same time as the eastern dropcut was being developed, thus maintaining the head grade. It also provided immediate access to ore after the eastern dropcut was finished.

Another advantage of the interim pit was that it provides multiple access to both the eastern dropcut and the cutback in the northwest, and delays establishment of the final northern and western walls.

The eastern drop cut extends down to RL 225. This was necessary in order to gain access to the high grade ore on the eastern side of the deposit. It is a logical development from the Stage 1 pit. The northern access ramp is in the cutback and goes down to RL 265, and connects up with the ramp in the eastern drop cut. A plan of the pit is shown in Figure 5. It has approximately 5.3 Mt of ore at a grade of 1.78 g/t.

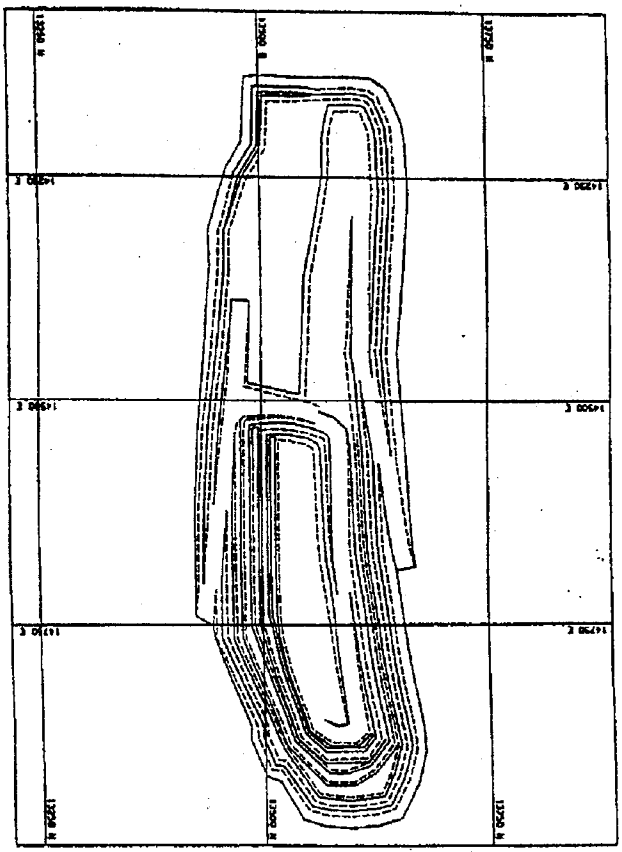

Figure 5 Stage 2 Pit Design

- **STAGE 3 PIT DESIGN**

A final cutback was designed between the interim pit and the LOM pit and is referred to as Stage 3. It has approximately 7.5 Mt of ore at a grade of 1.68 g/t.

A ramp was introduced in the northern wall to provide access from the surface (RL310) to RL 295. This assists in the mining of the upper benches of the cutback.

PRODUCTION SCHEDULE

- **AIM**

The aim of the production schedule was to devise a logical mining sequence that could be used to develop the mine and meet operational targets.

- **TARGETS**

The first 12 periods of the schedule had to meet the ore and waste requirements listed in Table 1. The primary concerns were to ensure enough waste was available to construct the tailings dam, and that sufficient ore was mined to provide feed to the plant.

TABLE 1 SCHEDULING TARGETS

Year	Period	Waste (bcm)	Ore (bcm)
94/95	10	120,000	0
	11	150,000	0
	12	150,000	0
	13	150,000	0
95/96	1	150,000	0
	2	140,000	10,000
	3	110,000	40,000
	4	90,000	60,000
	5	90,000	60,000
	6	90,000	60,000
	7	90,000	60,000
	8	90,000	60,000

After period 8 the ore mining had to continue at a minimum rate of 60,000 bcm per period.. Waste needed to be removed at a minimum rate of 35,000 bcm per period to enable the tailings dam to be constructed.

• BLOCK CREATION

Block layout plans were prepared for each 5 m bench. They are for the mid bench level (i.e. ±2.5 m). The plans show the interim pit and LOM pit limits. Some of the blocks were further sub-divided to assist in the scheduling.

A scheduling database was then devised using blocks from each 5 m bench.

Blocks were sub-divided into ore and waste in order to meet the scheduling criteria. It was beyond the scope of this study to create smaller discrete ore/waste blocks using the grade overlay as a guide. A global method of using percentages for each block was used in the scheduling.

• RESULTS

The scheduling targets were converted to tonnages using a density of 2.78 t/bcm. The quarterly targets were 542,100 t of ore and a minimum 316,225 t of waste.

It was found necessary to keep the waste removal rate at 813,150 t per quarter (i.e. 90,000 bcm per period), between Q1 96/97 and Q4 97/98. This allowed the western cutback to be developed at a rate that makes sufficient ore available as the eastern drop cut is mined out. It also permits some evening out of the head grade.

The waste removal rate was lowered to the minimum requirement of 316,100 t per quarter between Q1 98/99 and Q4 99/2000. After this period it decreases to approximately 120,000 t per quarter.

The ore mining is maintained at a rate of 166,788 t per period, and 542,100 t per quarter. The yearly quantities are given in Table 2.

TABLE 2 YEARLY QUANTITIES

Year	Ore (kt)	Waste (kt)	Grade (g/t)
94/95	0	1584	0
95	1808	3613	1.44
96	2169	3252	1.93
97	2169	3252	1.74
98	2171	1264	1.62
99	2166	1264	2.03
2000	2168	641	1.55
2001	35	21	1.39