IMHC

INTERNATIONAL MATERIALS HANDLING CONFERENCE

1989

In-Pit Crushing and Conveying
at Phalaborwa

G Fauquier
Manager, Mining
Palabora Mining Company

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ABSTRACT

Hauling rock from the Palabora open pit is the single greatest expense of the operation. During 1988 Palabora spent 28 percent of its total direct operating cost[1] on haul trucks this being equivalent to R71 million ($26.3 million). Conveying rock, however, has long been recognised as an attractive alternative to truck haulage and is generally used in the mining industry wherever practicable. It was for this reason that Palabora embarked upon the development of an extensive In-Pit Crushing (IPC) and Conveying project in 1985. The following paper describes the motivation for the project, its design and development and finally operation.

1 Direct operating costs includes all plant costs as well as the mine

Table 1: Summary of Operating Statistics

PALABORA OPEN PIT	June 1989	
Production		
Daily total tonnage	215000	
Daily ore tonnage	91800	
Stripping ratio	1.25	
Equipment	Fleet Size	Availability
150 tonne trucks	56	90
P&H 2800 shovels, 19 cubic metre	6	90
Gardner Denver 120 blasthole drills	6	97
CAT D9L track dozer	6	96
CAT 824C rubber tyre dozer	13	96
CAT 16G graders	6	96
Personnel	Supervision	Labour
Operations *	78	504
Maintenance	52	348
Productivity Factors		
Powder factor	2.90	
Metres drilled per tonne blasted	138	
Tonnes per manshift, total open pit	210	

* excludes Geology and Survey Sections

Haulage Optimisation

With 28 percent of direct operating costs attributable to truck haulage from the open pit there is a major incentive to reduce this component of cost. Over the years considerable savings have been realized through the implementation of innovative projects; some of the more notable achievements are described.

In 1981 trolley lines were installed on all permanent haulage ramps. As well haul trucks were equipped with pantographs and a sophisticated electrical control system which permitted the powering of the haul truck directly from the overhead trolley lines. Due to the favourable cost differential between diesel and electrical power the project now saves the operation approximately R15.5 per truck kilometre or the equivalent of 71 percent of the trucks energy cost per tonne of rock moved when compared to a truck powered by diesel only. Total savings attributable to trolley in 1988 amounted to R16.26 million.

Further major benefits from the trolley system are as follows:

- there was a 70 percent increase in speed of a fully laden haul truck travelling up an 8 percent gradient, thus increasing the overall productivity of the truck fleet, and

INTRODUCTION

Palabora Mining Company operates an open pit mine in the northeastern corner of South Africa approximately 550 kilometres northeast of Johannesburg. The property is a fully integrated mining and metallurgical operation, producing 140000 tonnes of refined copper per annum. In addition to copper, Palabora produces a host of other by-products such as zirconium oxide, uranium oxide, sulphuric acid, magnetite, nickel oxide and precious metal slimes. Gross revenues in 1988 were R921 million ($341 million).

The orebody, a carboniferrous intrusive pipe known as the Phalaborwa Complex, is low grade averaging 0.55 percent copper. The depth of the pipe at present is unknown, extending well beyond tehe final depth of the open pit.

The open pit measures 1.8 by 1.6 kilometres across the long and short axes respectively and is presently mined to a depth of 400 metres. Production rates peaked in 1980 at 348000 tonnes per day, however as the pit deepened the stripping ratio decreased resulting in an ex-pit production in 1989 of 208000 tonnes per day of which 94800 tonnes is ore.

The Palabora open pit has a planned life to the year 2000 at which point the bottom bench (number 58) will be 720 metres below the pit perimeter. Studies are currently in progress to determine the feasibility of developing an underground mine beneath the final open pit.

A brief summary of operating statistics as of June, 1989 (pre-IPC) is presented in Table 1.

- due to the decreased duty cycle of the engine its average operable life increased from 8000 to 15000 hours.

In 1985 Palabora implemented the computerised haul truck DISPATCH[2] Simulation System and immediately achieved an increase in truck utilisation of 7 percent; this resulted in a reduction of the truck fleet by 6 units with a consequent savings in operating and capital cost.

A concerted effort to improve truck maintenance by "paying attention to detail" resulted in an improvement in availability from 84 percent in 1985 to a consistent 90 percent in 1988/89, once again an equivalent savings of 3 trucks on the present fleet of 56 trucks.

Finally by 1986 it was recognised that the major gains had been achieved from truck haulage optimization and that further savings in rock haulage would only be possible by use of conveyors. However, due to major expansions of the open pit perimeter (started in 1980), the final pit walls and internal ramps in the lower sections of the open pit have only recently been exposed, and as a result the pit was considered suited for the development of In-Pit Crushing and Conveying.

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[2] supplied by Modular Mining, Tucson, Arizona

PROJECT MOTIVATION

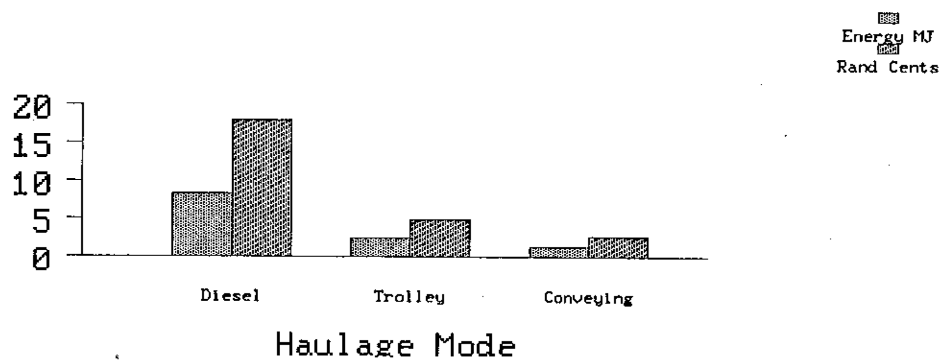
The motivation for conveyor transport versus truck haulage stems from the considerable savings in energy resulting from:

- the diesel engine only being 25 to 35 percent energy efficient,
- the requirement to haul a tare weight of 105 tonnes of truck with every 150 tonne payload of rock,
- energy inefficiencies associated with truck rolling resistance.

To illustrate the relative savings, the energy required to lift one tonne of rock through 100 vertical metres using diesel-electric haul trucks is 8.33MJ compared to 1.33MJ for the same tonne conveyed. On the other hand if the haul truck is powered continuously by trolley, as is the case at Palabora, then the energy requirement per tonne is 2.45MJ, (Figure 1). Thus, Palabora could anticipate a 46 percent savings in energy for ore conveyed rather than trucked under trolley.

Figure 1: Haulage Energy Efficiency

Haulage Mode	Comparable Energy Mega-Joules per	Comparable Energy Cost Rand Cents per
	--------tonne per 100 vertical metres--------	
Diesel	8.33	18.0
Trolley	2.45	4.9
Conveying	1.33	2.7



In addition to the favourable energy comparison, conveying reduces the vulnerability of mining costs to escalation and foreign exchange variations, this being of particular importance in the South African economy where almost all haul truck spares are imported. Palabora currently spends R43 million per annum on spares and tyres alone, representing approximately 28 percent of the total <u>mine</u> operating cost.

Conveyors translate this variable truck operating cost into a fixed depreciable cost minimally affected by monetary fluctuations.

Finally, conveyor haulage has deferred the replacement of older haul trucks in the fleet which inevitably would have been a foreign currency expenditure.

By placing a crusher in the pit on Bench 28, some 290 metres below the pit perimeter, and conveying the ore to the surface stockpiles, Palabora expected to reduce its fleet of haul trucks by 22 percent and its mine operating cost[3] by 10 percent.

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[3] Includes costs associated with the operation and maintenance of the open pit only.

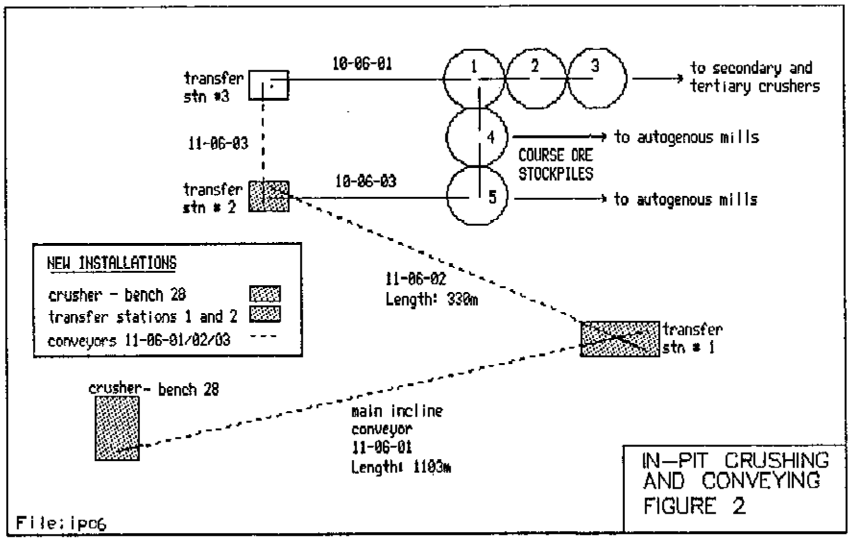
PROJECT DESIGN

General

The Project includes a gyratory crusher on bench 28 connected to the surface stockpiles by three conveyors in series, the first of which is installed in a one kilometre tunnel mined through the pit wall at an angle of 15.5 degrees (Figure 2). While the crusher installation is of conventional design engineering, the main incline conveyor belt, from bench 28 to surface, is "state of the art" technology.

The maximum design capacity of the system is 6500 tonnes per hour, which is approximately 65 percent greater than the hourly ore requirement of the mill. The reasons for this additional capacity are to account for the inevitable surges that are necessary to fill the coarse ore stockpiles, and to allow for mechanical availability and operational utilisation factors which are typical of such systems. Of further consideration is the vast spread in material densities due to the variable magnetite content in the ore.

A Project Facts sheet follows, (Table 2).



IN-PIT CRUSHING AND CONVEYING
FIGURE 2

Crusher

The gyratory crusher, a 60x89 Fuller Traylor, is permanently installed in a rock excavation on bench 28 (Figure 3). The 154 tonne haul trucks tip run of mine material into the dump pocket from two opposing sides. The Palabora ore is generally hard[4] and blocky by comparison to typical porphyry copper deposits, and thus an impact hammer is extensively used to clear bridges. As a back-up the operation uses a hook attached to the auxiliary overhead crane in addition to a CAT 245 backhoe.

The crushed rock passes into a rock box having a live capacity of 635 tonnes. The level of ore in the box is controlled by a sonar level device (to the belt feeder) with "high" and "low" cut-out protection provided by nuclear level sensors; the latter instrumentation automatically prevents tipping of the haul trucks on "high" or "low" indication.

From the rock box the ore is fed onto the main incline belt by a 2800mm wide belt feeder. The feeder is powered by a direct coupled variable speed hydraulic motor which drives the feeder between 0 and .33 metres per second. The hydraulic power pack comprises three identical hydraulic motor/pump combinations rated at 75kW each; under normal operating conditions only two units are required. The feeder is designed so that the "endless" ST2000 steelcord belt can be fitted and withdrawn from the pulleys without disassembly of the feeder.

The feeder speed is regulated by four measured inputs, the level in the rock box, the power draw from the main incline conveyor, the main inclined conveyor weightometer and finally the loadcells monitoring torque on the conveyor drive units.

When the rock box level drops below 55 percent of "full", the feeder will reduce speed to a minimum equivalent of 2000 tonnes per hour at which point the feeder will stop; below this tonnage there is the danger of roll back on the incline belt resulting from insufficient volumetric loading.

Both the "power draw" and the "weightometer" will cause the feeder to reduce speed or stop if either measurements are above certain pre-set limits.

As a final safety feature the load cells beneath the shaft mounted drive units on the main incline conveyor are designed to stop the belt feeder in the extreme condition of over-loading, (refer to "Drive System" in the following Section).

The objective of the feeder is to maximise both volumetric and mass loading of the system within the load limitations of the following conveyor belts, in particular the main incline belt.

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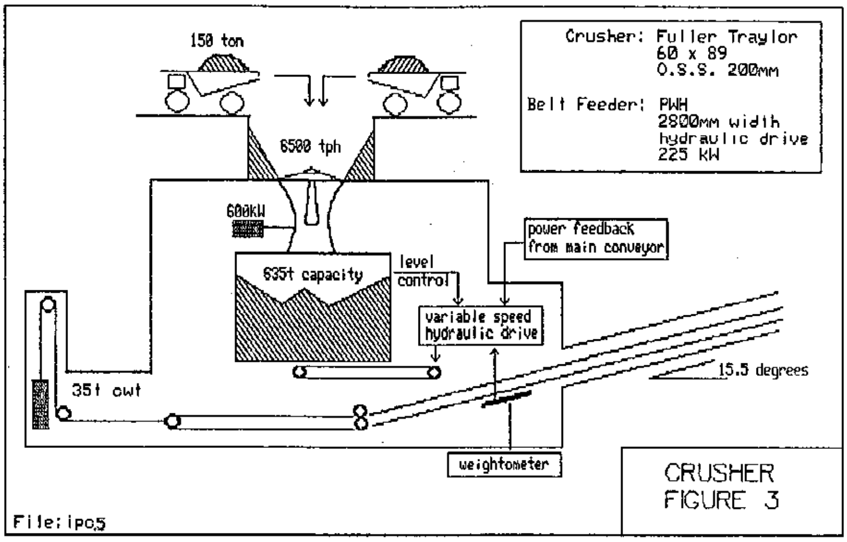
[4] Compressive strengths varying from 100 to 125 MPa



CRUSHER
FIGURE 3

Main Incline Conveyor

Drive System

The main incline conveyor (11-06-01) is the second highest tension belt of a similar capacity in the world and as a result, it is very much the focal point of the Project.

The belt is installed in a tunnel mined at an angle of 15.5 degrees over a distance of 1043 metres. The vertical rise of the belt is 294 metres. The belt is 1800mm wide and runs at a speed of 4 metres per second.

The basis of the design is a study undertaken by Conveyor Dynamics Incorporated, in which the starting and stopping sequences of the belt were dynamically analysed. The objective of this analysis was to find the optimum design parameters which minimized the potentially damaging transient forces (travelling waves) resulting from a high tension belt being accelerated or decelerated.

The belt is driven by three identical units, each comprising:

- a Brown Boveri 2350kW wound rotor motor,
- a Flender right angle triple reduction gearbox,
- three Voith brake calipers hydraulically activated
- a 1100mm diameter by 40mm thick flywheel
- a Bikon low speed coupling
- a Flender high speed coupling, and
- a Marland holdback

These components are mounted on a swingbase, the gearbox being "shaft mounted" from the primary and secondary drive pulleys respectively. A load cell beneath the swingbase provides static support for the drive unit and is used to monitor driving torque. The load cell is positioned so that the reaction to the driving torque at 75 percent full power equalises the static loading forces on the coupling. Being shaft mounted the entire assembly is free to gyrate about the axis of the pulley shaft. (Figures 4 and 5)

The total mass of the complete drive unit is 60 tonnes.

The brake calipers are necessary to bring the empty or partially empty belt to a full stop within nine seconds, this being the prescribed optimum stopping time. The control of the brake comes from the latest power draw of the main drive motors monitored prior to the shutdown sequence being initiated; this power feedback determines the rate of proportional braking.

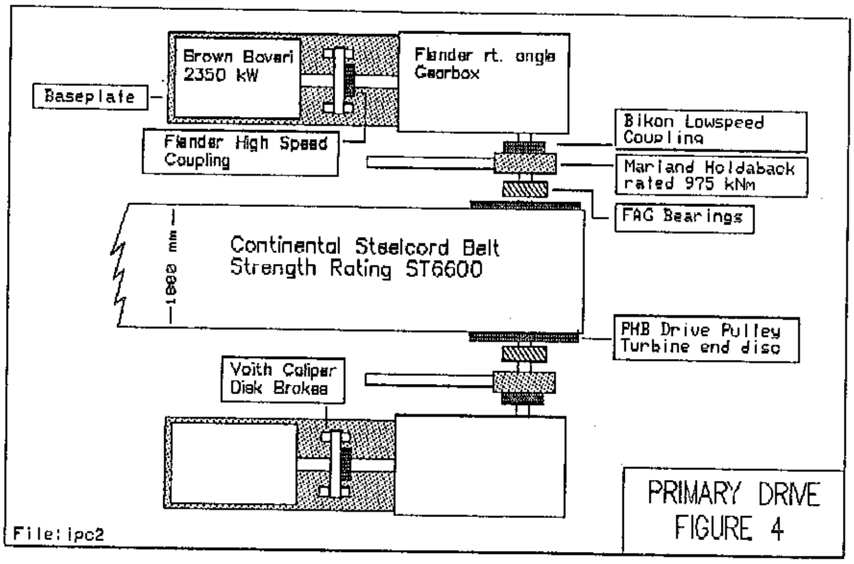
The holdbacks, one for each drive, are designed to share the load through the use of a "Belville" spring support arrangement; load sharing between the two primary pulleys is within 10 percent and between the primary and secondary pulleys within 25 percent of each other.

The conveyor has a simple gravity take-up of 31 tonnes, the configuration and mass having been established through dynamic analysis of various starting and stopping conditions.

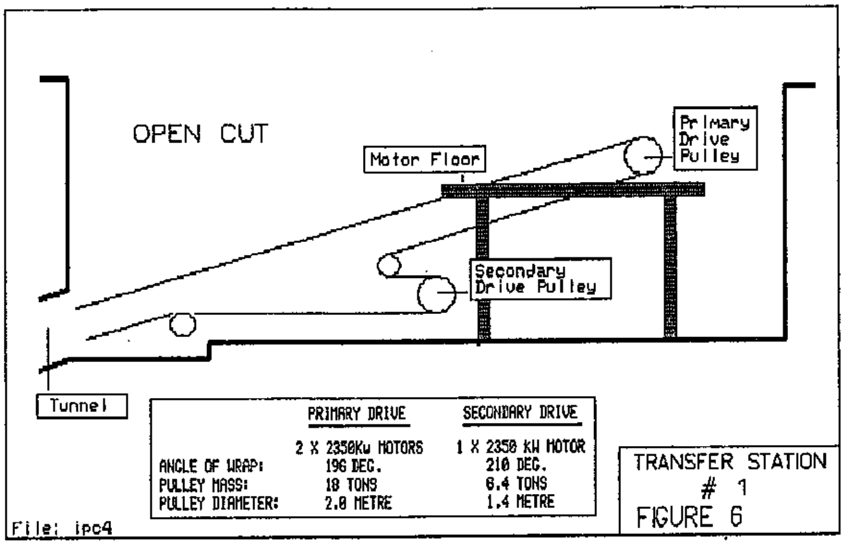
Starting of the conveyor is controlled through 24 stepping resistors (8 per motor phase) which are timer sequenced by the PLC to be brought in and out of the motor rotor circuit on feedback of power draw from the drive motor. On starting the main incline belt the PLC will configure a total of 63 resistor steps, 21 per drive unit. At start-up peak motor torque is limited to 130 percent of nameplate rating. The slip between the secondary and primary pulleys is balanced by slip resistors permanently in the rotor circuit of the secondary drive.

The drive pulleys, designed and manufactured by PHB Weserhutte, West Germany, are of the "turbine" end disk design with hot vulcanized lagging. The primary and secondary pulleys weigh 18 tonnes each and are 2 metres in diameter.

A hydraulically driven variable speed motor is fitted externally to the gearbox of the main drive unit to enable the conveyor belt to be "creep" driven for maintenance activities. This facility has proven to be invaluable.

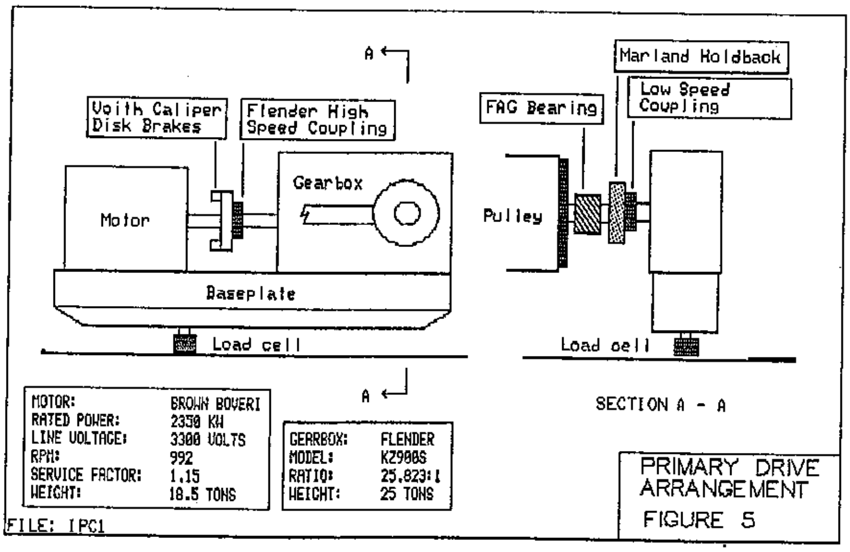


PRIMARY DRIVE
FIGURE 4



	PRIMARY DRIVE	SECONDARY DRIVE
	2 X 2350Kw MOTORS	1 X 2350 kW MOTOR
ANGLE OF WRAP:	196 DEG.	210 DEG.
PULLEY MASS:	18 TONS	6.4 TONS
PULLEY DIAMETER:	2.0 METRE	1.4 METRE

TRANSFER STATION # 1
FIGURE 6



MOTOR:	BROWN BOVERI
RATED POWER:	2350 KW
LINE VOLTAGE:	3300 VOLTS
RPM:	992
SERVICE FACTOR:	1.15
WEIGHT:	18.5 TONS

GEARBOX:	FLENDER
MODEL:	KZ900S
RATIO:	25.823:1
WEIGHT:	25 TONS

PRIMARY DRIVE ARRANGEMENT
FIGURE 5

Conveyor Belt

The belt, supplied by Continental, West Germany, is of "steelcord" construction having a strength rating of 6600 Newtons per mm of belt width (ST6600). The safety factor of the belt is 6.5:1, while the splice is tested to fail at 48 percent of the belt strength rating or the equivalent of 3168 Newtons per mm of belt width.

As part of the vendor qualification process, the splice design underwent a rigorous set of fatigue tests at the Hanover University in West Germany. The tests included cycling a 15 metre length of belt at a speed of 6 metres per second for 200000 cycles. During each cycle the pulley tensions the belt to within a prescribed fractional value of the strength rating. The minimum acceptable "fractional value" before failure of the splice was specified as 36.9 percent of strength rating.

Although a 1500 mm width belt would have been adequate to handle the volumetric capacity, the tension rating finally dictated the width to be 1800mm. There are 92 steel cords of 12.4mm diameter at a pitch distance of 19mm. The belt when at full capacity of 6500 tonnes per hour is only 60 percent volumetricly loaded.

Above and below the steel cords is a nylon breaker ply which is intended to eject tramp steel should it pierce the belt. Every 30 metres an electronic loop antenna is fitted in the upper cover of the belt which if broken, is detected by electronic sensors at the loading zone and the head end of the conveyor. Should any of these loops become defective the sensor may be programmed to ignore that particular loop until it is repaired.

The total belt length of 2320 metres was installed in 9 segments, all of which were spliced together prior to feeding into the tunnel. Due to the extreme mass, 40 tonnes on the 15.5 degree incline, a carefully engineered rail mounted trolley carriage fitted with hydraulic clamps, was used to lower the belt into the tunnel. Belt installation was the subject of intense pre-planning and anxious moments.

Each splice is 6 metres long with a four step cycle of rope overlay. X-rays were taken of all splices, which in one case revealed a short cable and resulted in the splice being re-done.

Conveyor Structure

From a survey of the local market it was apparent that there were no idlers available which could handle the duty, therefore both idlers and support structures were custom designed. The idler is a five roll design, the frame being tubular. Due to the high running tensions of the belt and the need to minimize transient vibrations, the alignment tolerances were very strict and difficult to achieve.

Fire Protection

The entire length of the belt is protected by an electronic fire detection system supplied by Anglo American Research Laboratories. A desk top computer continuously monitors 50 nuclear activated smoke detectors positioned in pairs throughout the length of the belt. A signal from any one of the detectors, if not acknowledged within 30 seconds, will automatically set off the fire spray system within the zone of the specific detector.

Remaining Conveyors

The remaining conveyors in the system, numbering eight, allow the selection of five different stockpile destinations . Two of the eight conveyors are new and were part of the Project while the remainder are existing plant. (Figure 2)

One of the new conveyors (11-06-02) is an ST3000 steelcord belt having a vertical rise of 43 metres. This conveyor has one drive unit, similar to that of the main incline belt, except the installed power is 1250kW.

Of particular note is the design of the transfer chutes at the feed and discharge end of the second conveyor both of which are difficult configurations through which to drop the rock.

At the feed end the rock must be turned through 170 degrees to be fed onto the second conveyor. Full use is made of rock boxes. All liners plates are rubber backed for sound adsorption. The chute is held under negative pressure by a dust extraction system.

At the discharge end of 11-06-02, the rock feeds into a "moveable chute" which is designed to allow the redirection of the discharged rock between one of two conveyors without interrupting the material flow.

All the conveyors in the existing plant were doubled in speed from 2 to 4 metres per second.

Control System

Programmable Logic Controllers (PLC)

Control of the entire system is derived from distributed and interconnected Siemens PLCs. Each of the four PLCs serve to monitor and control operations within their respective areas of influence. The master PLC, a Siemens SU-150, is located in the crusher control room and not only controls the crusher and associated equipment but as well provides overall control to the system, for example sequenced start-up of all conveyors. The remaining three PLCs, Siemens SU-115s, are geographically distributed throughout the project.

All PLCs are interconnected by a dual optical Ethernet system using the standard Siemens H1 highway, adapted for optical fibre transmission. Optical transmission has the advantage of being immune to electro-magnetic and radio frequency induction, this being of particular importance in the one kilometre long conveyor tunnel. A software fault detection procedure was developed which switches the highway over to the redundant fibre optic cable within 5 ms of detecting a fault.

The control software resides in the PLCs and controls five specific areas of operation by confirming the physical inputs and the operator controls to produce the required outputs.

The software was designed to allow three basic modes of operation:

- the "manual mode", where an individual piece of equipment may be operated independent of other system interlocks; this mode is restricted to maintenance activities only.

- the "local mode", which allows operation of all equipment within the influence of the specific PLC, but ignores interlocks from other PLCs or the master PLC; this mode may be used in the event of a communication failure between PLCs.

- the "automatic mode" wherein the entire system is run under the control of the master PLC in which resides the Mine Control System (MCS).

The software was written in the Siemens statement list language known as Step Five.

The entire system comprises 1395 digital and 307 analog inputs/outputs.

Operator Interface

Interface with the control system is provided by the Siemens COROS VDU graphics system. The objective of the interface is to:

- provide operational control
- diagnose fault conditions

The screens are designed with a hierarchal tree structure such that the highest level is the composite screen of the entire material flow system and the lowest level is the individual motor control centre. When using COROS for diagnostics, the operator has the facility for paging "down" to the specific equipment screen showing the fault within seconds. Every input and output in the PLCs has been accounted for in the graphic displays.

Mine Information System (MIS)

The MIS is a stand-alone software package to which the PLCs download selected information. The purpose of MIS is to provide operational and diagnostic trending. The package used is Multiview run on a standard IBM compatable AT computer.

Table 2: Project Facts

Design Capacity	tonnes	6500
Nominal Capacity	tonnes	5100
Material Bulk Density	tonnes/m^3	2.19
Operating Days per Year	days	308
Crusher		Fuller Traylor type TC 60 x 89
Open Side Setting	mm	200
Installed Power	kW	600
Belt Feeder		PWH hydraulic powered
Belt Width	mm	2800
Belt Strength	N/mm	ST2000
Installed Power	kW	225
Main Incline Conveyor		
Inclination	degrees	15.5
Length	metres	1103
Vertical Rise	metres	295
Speed	mps	4.06
Belt		Continental, Stalchord
Width	mm	1800
Strength	N/mm	ST6600
Mass	kg/m	127
Splices		9 x 6.0 metres long
Steel Cords		92
Diameter	mm	12.5
Rip Detection		Montan Forchung, electronic loops
Conveyor Drive		PHB design, shaft mounted
Installed Power	kW	2350 x 3
Total Mass	tonnes	60
Primary Pulley		PHB design, turbine end disc
Diameter	mm	2000
Mass	tonnes	18

MINING AND CONSTRUCTION

General

Fluor Engineers were appointed as the Management Contractor in May, 1986 and were responsible for the engineering design, materials procurement and construction supervision. All engineering was undertaken in Johannesburg.

Conveyor Dynamics Incorporated (CDI) provided the basic design for the two high tension incline conveyors (11-06-01/02) as well as designing the control system and respective software.

A significant contribution toward the design and manufacture of the main conveyor drive units and the belt feeder was made by PHB Weserhutte.

The major items sourced overseas were the crusher (USA), the main conveyor (11-06-01/02) drive units (Germany) and the belt plus vulcanizing equipment and clamps (Germany).

Mining

Mining of the tunnel commenced in September, 1986 with the simultaneous advance from both ends. The tunnel has a width and height of 5.0 x 3.4 metres with an arched back and a total length of 1043 metres.

The tunnel gradient of 15.5 degrees was particularly difficult for mining as it was too steep for normal LHD operation and yet not steep enough for the free flow of blasted material when mining on the incline.

In the decline a winch assisted LHD was used, side-tipping into a 10 tonne rail mounted car which was then pulled out of the tunnel by a sinking hoist.

In the incline blasted material was removed with two scrapers in series, approximately 175 metres of tunnel per scraper. The advance in the decline was faster than the rock removal resulting in a large build-up of blasted material in the tunnel.

All tunnel headings were drilled and blasted using conventional air operated jacklegs and prilled ammonium nitrate explosives respectively.

The crusher excavation was 13900 cubic metres. After collaring through five metres of blasted subdrill from the bench 28 elevation, a raise borehole was drilled through to bench 30 (25 metres in length) into which was blasted successive two metre benches to finally "hole" through into the underground development on bench 30. The excavation was completed in 11 months.

The mining activities extended the planned project schedule by four months due to a lack of adequate experience and organization on the part of the mining contractor.

Construction

Construction commenced in June 1987 with the civil works in the main transfer station at the top of the tunnel and was complete in November 1988, for a total of 18 months.

The construction effort was complicated by the necessity to tie the Project into an existing plant as well as an operating open pit.

Palabora's blasting consultant specified a range of "peak particle velocities" resulting from open pit production blasts which could be tolerated by varying ages of poured concrete. Civil construction activity in the open pit were therefore regulated by this criteria.

The following three activities are of particular interest in that each posed some unusual problems:

Concrete Tunnel Floor

Due to the steep gradient of the tunnel normal concrete placement techniques could not be applied. Batched concrete was hoisted down the tunnel in rail mounted cars and shovelled into moving forms. The control on "slump" was particularly important to avoid movement of the concrete down-grade during curing. The placement of 1043 linear metres of floor, equivalent to 998 cubic metres, took 5 months to complete.

Installation of the Main Incline Conveyor Modules

The installation of the conveyor modules in the tunnel was a logistics problem due to their size and weight. The sinking hoist was used to move the sets down the tunnel on a custom designed rail car. Allowable line tolerance between any three idler frames was 1.0mm.

Installation of the Main Incline Conveyor Belt

The installation of the main conveyor belt was a major exercise. All nine rolls were spliced together above the collar of the tunnel on a specially prepared asphalt roadway. The belt was first lowered down the top side of the conveyor using a custom-made designed trolley equipped with Wagener clamps; the trolley was moved up and down the asphalt roadway with a D9L dozer. Once the top side of the belt was in place and securely clamped, the bottom side was fed into the tunnel over the primary and through the secondary pulleys. The two long sections of belt were then spliced at the tail and head ends of the conveyor.

July 25, 1989

Table 3: Mining Facts

Tunnel	
length	1043 metres
gradient	15.5 degrees
cross-sectional area	17.25 square metres
Average Tunnel Advance	2.5 metres per day per heading
Crusher Chamber rock volume	13900 cubic metres

Table 4: Construction Facts

Concrete Poured, cubic metres	
crusher	7190
tunnel	998
main transfer station	3700
miscellaneous	33
Total	11921
Steelwork, tonnes	1089
Construction Manhours	
management	119600
mining	917700
civil	917000
mechanical	507300
electrical and control systems	175100

OPERATIONS

Availability and Throughput

In order to achieve the daily ore target of 94800 tonnes per day the IPC system must have a utilisation of at least 75 percent and maintain its nominal capacity of 5100 tonnes per utilised hour.[5] Experience shows that to achieve this utilisation the mechanical availability must be in the order of 90 percent.

To place this throughput into perspective, the crusher must tip 633 truck loads per day or one truck every 1.7 minutes. To date the crusher has achieved a maximum sustained digestion rate of 6900 tonnes per hour or the equivalent of 1.24 minutes per truck.

Since operational commissioning commenced in January of this year, the IPC has demonstrated its ability to achieve in excess of its nominal rating. Generally the integrity of the design has been proven.

Table 5 illustrates the monthly availabilities and tonnage throughput (as a percent of total ore) for the first six months of the year. As of June 30 the availability and throughput was 62 and 47 percent respectively. The downturn in availability during May and June is related to a few major difficulties resulting in several days of system downtime.

Some of the more specific "difficulties" are noted:

The crusher liners were cemented into place with a commercially available chemical epoxy resin.[6] After a month the resin failed and the liners had to be replaced using zinc backing.

A loose retaining nut on the mainshaft necessitated the change-out of the mantle.

Approximately 10 metres of the main incline conveyor (11-06-01) belt ripped, requiring extensive repairs supervised by the supplier.

The hydraulic impact hammer, used in the crusher dump pocket, is generally underdesigned for the duty. This has resulted in numerous failures of major components.

The Siemens controller card in the master PLC and a VDU controller card in the COROS has repeatedly failed for reasons as yet unexplained.

Corrupt software eluded detection for a number of days as did a fault on the communications fibre optic "highway".

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5 Nominal capacity is 78 percent of "design" capacity, 6500 tph.
6 Palabora's standard is zinc metal backing of crusher liners

Although the low mechanical availability continues to be an immediate concern, there is no doubt that the system will be capable of continuously sustaining an availability of at least 90 percent.

Interface With Pit Operations

As a result of the IPC, there has been a major change in the emphasis placed on primary crushing. Where previously the three surface crushers were consistently underutilised at 35 percent, (due to their overcapacity), and thus did not represent a constraint to the operation, now primary crushing is an acute activity in the material flow.

The effective control of ore trucks to the crusher is critical if the best utilisation is to be realized from the IPC and the truck fleet. This control is maintained through the DISPATCH system. Dependent upon the status of the IPC and the potential waiting time for individual trucks at the crusher, DISPATCH will direct trucks to the in-pit crusher or the standby surface crushers. However, in the latter case the truck's cycle time is 30 minutes more than had it tipped at the IPC. The net effect of sending trucks to surface is to diminish the productivity of the operable fleet.

On the other hand if trucks are allowed to queue at the IPC while waiting for the crusher to be operational, then the truck cycling becomes imbalanced and shovels are left without trucks. The net result is a reduction in truck and shovel utilisation.

The interface of the loading and hauling operation with the in-pit crusher is continuing to be optimised. Both systems are particularily sensitive to one another and require a continuous effort on the part of the DISPATCH operator to maintain an even flow of ore. The DISPATCH operator can and does require to over-ride the automatic dispatching of haul trucks and therefore his decisions can have a marked effect on the productivity and utilisation of the crusher and the truck fleet.

July 25, 1989

Table 5: Operational Facts

1989	Mechanical Availability Percent	Tonnage Throughput Percent of Total Ore
January	47	7
February	47	31
March	76	67
April	83	81
May	80	65
June	40	33
Overall	62	47

CONCLUSION

Given the correct operating environment, conveying is the preferred mode of rock transport in the mining industry today. In most open pit mines the application of in-pit crushing and conveying is only limited by the physical configuration of the mine, wherein the changing geographic limits of the mine prohibit a relatively permanent structure. There are many examples of operations which have overcome this physical restriction by installing portable or mobile crushers, however the placement of the ex-pit conveyor generally requires premanent siting, particularly if it is a high tension belt.

Palabora has successfully designed and installed a crushing and conveying system which is on the forefront of conveying technology. Apart from the economic benefits derived, Palabora believes that the Project represents a notable contribution toward the development of mine materials transport systems. The combination of tension, speed, capacity and belt inclination makes this application unique and of interest to the continuing design optimisation of mine conveying systems throughout the mining industry.