OPTIMIZATION OF PRIMARY GYRATORY CRUSHING AT
HIGHLAND VALLEY COPPER

BY

PERSIO P. ROSARIO

A thesis submitted in partial fulfilment of
the requirements for the degree of

Master of Applied Science in
The Faculty of Graduate Studies
Department of Mining Engineering

We accept this thesis as conforming
to the required standard

The University of British Columbia
Vancouver, B.C., Canada
October 2003
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Abstract

This thesis presents the work done in a collaborative research project between the University of British Columbia and Highland Valley Copper. The research was aimed at understanding gyratory crusher liner wear in the overall context of the crushing process. Wear measurements were taken for in-service crushers during the research period using a novel laser profile measurement device. Data from the wear measurements was correlated with crusher production information such as current draw and throughput. This work resulted in enhanced knowledge of crushing chamber characteristics and their impact on crushing performance. In addition, an innovative and powerful way to evaluate crusher liner profiles was developed and new mantle profiles were designed.
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Nomenclature

AG        Fully Autogenous Grinding
AMWP      Average Mantle Position for the Week
CDI       Conveyor Dynamics, Inc
CLP       Constant Liner Performance
CSS       Closed Side Setting
FMV       Feed Material Variables
GPS       Global Positioning Systems
HB        Brinell Hardness
HVC       Highland Valley Copper
LPD       Laser Profiler Device
MDV       Mechanical Design Variables
MOV       Machine Operating Variables
MP        Mantle Position
MTPH      Metric Tonnes Per Hour
OSS       Opened Side Setting
ROM       Run-of-mine
SAG       Semi Autogenous Grinding
TPH       Tonnes Per Hour
Acknowledgements

The author would like to express his gratitude to HVC for all their support for the work presented in this paper. In particular, Arnie Adams for his confidence in the author since the beginning of the project. The author would also acknowledge all the other participants of the HVC project team for their kindness.

The author is deeply thankful to his thesis supervisor Dr. Robert Hall and his co-supervisor Dr. Daan Maijer for all the guidance and support. In addition to their support, the author appreciates the bond that has been developed.

Of course, without my wife’s support and patience, and the love shown by her and my son, this thesis would not be completed. I am also thankful to my family, who always demonstrated trust in my capacity even from far away.

The author would also like to acknowledge the financial support from the Natural Sciences and Engineering Research Council of Canada.
1. Introduction

Canadian mining operations are facing competitive pressure from offshore high grade, low labour cost mines. To remain competitive, companies have introduced larger and more complex equipment. Complexity has been added through the addition of electronics and in some cases partial or full automation. Larger haul trucks, primary crushers and process plants are now commonly used in open pit mines. The increase in mine equipment size and operational improvements has resulted in higher production rates. For the successful integration of these operations, it is important that increased production levels from the mine still provide a consistent product in terms of grade and size to the mill. Many companies are looking for ways to better evaluate the relationships between blasting, primary crushing and milling efficiencies in order to develop new processing strategies to be applied in the optimization of the overall process.

At the present, it is known that the optimization of primary crushing provides a great opportunity to enhance the overall operational efficiency in high tonnage mines. Increasing throughput and product quality at the primary crushing phase improves the productivity throughout the rest of the comminution process (Burkhardt, 1982). In addition to reduced processing costs through the gains obtained in the crushing and milling stages, maintenance costs are reduced by better machine availability and by enhanced reliability of its components.
This thesis presents research aimed at understanding the influence of liner profiles and liner wear on gyratory crusher performance. Once a better understanding about these relationships is achieved, the mines may be equipped with better tools to optimize processing operations.
2. **Research Objectives**

The primary objective of this research is to improve the understanding of liner wear in primary gyratory crushers. In other words, how chamber geometries and their modification impact on crushing capacity and product quality. In pursuit of the primary objective the following secondary objectives are targeted:

- Determine an efficient methodology to monitor liner profile wear over time. This work involves the assessment of a prototype laser-based methodology recently introduced in the supporting mine.

- Establish an approach to accurately determine the dimension of crusher closed side setting (CSS) replacing the current “bucket test” test methodology. An online methodology is the ultimate goal.

- Develop a database of wear information linked to other monitored crushing parameters such as: current draw, product size distribution and production rate.

- Evaluate the liner profiles currently available at the mine and, if necessary, develop new profile designs. The concurrent enhancement of crushing performance and the extension of liner lives is desired.
3. Literature Review

3.1 Crusher Machines

3.1.1 History

Many centuries ago, weights were raised and dropped onto heavy rocks to crush them in order to enable minerals processing. The crushing process has evolved with the addition of different power sources to the process, beginning with the use of animal and waterpower. However, development of "modern" crushing machines only took place during the 19th century (Utley, 2002). The first crushing machine appeared in English mines in the early 1800's. During the Industrial Revolution, the "Cornish Rolls" device was developed, and though a very limited device, it started the process of minimizing handwork (Flavel, 1982).

Two kinds of crushing machines, not so different from the ones in operation nowadays, were invented in the second half of the 19th century. The Blake Jaw Crusher was the first in 1858 and the Gates' Gyratory Crusher was patented in 1881 (Figure 3-1 and Figure 3-2 respectively).
The industrial revolution in the early 1900’s promoted a growth in mining volumes. This growth resulted in an increase in crusher size as well as the invention of other types of crushers (the hammermill and the single sledger roll crusher). In 1919, the first 1.5 m (60 in.) gyratory crusher was manufactured by Traylor Engineering which remained the largest gyratory crusher for 40 years. In 1969 the same company introduced the largest machine to date at 1.8 m (72 in.) of feed size opening. Although the reasons are unclear, only one unit was made (Utley, 2002).

In addition, as mining complexity increased, the number of comminution phases grew which led to the development of other types of crushers. Cone crushers were first developed in the mid 1920’s by Edgar B. Symons to supply the demand for efficient fine crushing machines. Cone crushers are basically a small-scale gyratory crusher with chamber modifications and higher operation speed (O’Bryan & Lim, 2002).
3.1.2 Crushers types and operation principles

Jaw crushers and gyratory crushers are the most commonly used machines for primary crushing due to their capability and robustness to handle great volumes and high strength materials. Primary crusher feed, the run-of-mine (ROM) ore, may contain lumps as large as 1.5 m across (Utley, 2002). Usual reduction ratios for primary crushing are around 8:1 (Major, 2002).

Other types of crushers that can be employed in the primary comminution phase are: rotary breakers (MMD sizer), impact crushers, high-speed roll crushers, and hammer mills. These machines are usually used for ores with specific characteristics such as: low compression strength, low abrasion index, and/or high clay content.

In order to achieve the desired particle size necessary to process the ore by methods such as flotation or leaching, other stages of crushing and/or grinding are commonly applied after primary crushing and though several different processes exist for finer crushing, cone crushers are typically part of these processes. Thus, cone crushers can be used for secondary and tertiary crushing phases as well as for auxiliary phases in grinding mill applications.

In order to fulfil the requirements for such a variety of applications several types of cone crushers are available. They are: the standard cone crusher, the horizontal impact crusher, the high pressure grinding rolls, the waterflush cone crusher, the disk crusher
(Telsmith Gyrasphere), the short head cone crusher, the Metso Gyradisk and the vertical impact crusher.

Since this research is focused on optimization of the primary gyratory crusher operation, more attention is given to the description of this kind of machine. Since the jaw crusher was the basis for the development of the gyratory crusher and the cone crusher was derived from the gyratory crusher, they are also covered in this work.

**Jaw Crusher**

In the Jaw crusher, two planer surfaces alternatively crush the rocks imitating the animal jaw movement. One of the surfaces remains fixed and the other, the swing jaw, moves according to an eccentric drive - directly or indirectly, depending on the machine type. The way the swing jaw is pivoted and some other construction characteristics determine the different types of Jaw crusher. Details about the different types of Jaw crushers are shown in Figure 3-3. The dimensions of the rectangular receiving area are commonly used to describe them; for example, a 2.1 by 3.0 m (84 by 120 in.) jaw crusher has a 2.1 m width and 3.0 m gape. Jaw crushers are available in a wide range of sizes and capacities – from 50 to 1500 tonnes per hour (tph).
There are several types of jaw crushers, such as the Blake double toggle, the Single toggle, the Dodge, the Universal, and the Telsmith (Gaudin, 1939). The types most commonly found in mining operations are the Blake double toggle and the Single toggle. The Single toggle has limited application for high abrasives ores because in this type of jaw crusher the swing jaw moves elliptically resulting in greater liner wear (Utley, 2002).

The Blake double toggle is commonly used in primary crushing in both open-pit and underground operations. The double toggle drive mechanism and the positioning of the pivot point give a minimum displacement of the swing jaw at the inlet region as well as provide a strong breaking action for large ROM rocks.
In the Dodge crusher, as opposed to other jaw crushers, the location of minimum swing jaw movement occurs in the discharge region providing a more uniform product size. The Dodge type is the simplest jaw crusher but shows efficiency limitations in large-scale designs; hence, this type is restricted to laboratory-scale work (Wills, 1997).

Although the capacity is smaller than the capacity of a gyratory crusher, a jaw crusher has the advantages of low cost, simplicity of operation and maintenance, and low head clearance, therefore they are broadly used in underground primary crushing (Major, 2002).

**Gyratory Crusher**

In the gyratory crusher, rock flows through a chamber formed by two inverted conical surfaces assembled one inside the other. The inner surface, the mantle, is movable and sits on a shaft called the mainshaft; the outer surface, the concave (or concaves) is fixed on the main frame of the machine. The mainshaft is guided by a concentric sleeve at the top and an eccentric sleeve-assembly at the bottom; known as the eccentric. A motor-pinion-gear set propels the eccentric, which in turn drives the mantle in a gyratory movement. Crushing occurs by the circular approaching and receding movement between the surfaces.

Eccentric dimensions determine the displacement (or linear moving distance) of the mantle, also called the throw of the crusher. As a result of the assembly of the eccentric
at the bottom of the mainshaft, the maximum displacement of the mantle occurs at its bottommost region. Both mantle and concave are cast using abrasion resistant iron alloys and are designed to be replaced over time as they wear. They are also called mantle and concave liners.

Gyratory crusher capacities can range from 350 to 10,000 metric tonnes per hour (MTPH) (Utley, 2002). The radial receiving opening (in inches) is the characteristic generally used to determine the size of a gyratory crusher, i.e. a 48 gyratory crusher has a receiving opening measuring 1.22 m (48 in.). Some manufacturers add the largest diameter of the mantle to the size description, for example a 60-89 Superior (Metso) has an 1.52 m (60 in.) gap and the largest recommended diameter for the mantle is 2.26 m (89 in.).

Since its invention, several different configurations of gyratory crusher have been developed mainly related to the design and support mechanism of the mainshaft. The different types of gyratory crushers include: the long-shaft spider-suspended type, the fixed-shaft type, the short-shaft gearless type, the short-shaft spider-suspended, and the hydraulic supported short-shaft; the latter being the one most commonly manufactured today.

The hydraulic supported short-shaft gyratory crusher has the bottom extremity of the mainshaft supported by a hydraulic piston allowing a limited vertical displacement of the mantle. This vertical movement of the mantle, also found in spider-suspended types of
gyratory crushers, serves to compensate chamber wear. The hydraulic system gives an additional advantage as it also serves as a quick relief system that is used when the machine becomes blocked; this usually happens when the machine receives tramp material or is operated “too tight”, i.e. with a small CSS.

Another important aspect about the design of gyratory crushers is that the free movement of the mainshaft inside the eccentric minimizes attrition between the rocks and the surfaces; i.e. once the mainshaft, and therefore the mantle, is free to turn on its axis inside the eccentric, horizontal attrition over its surface is negligible and the main type of mechanical reduction is through compression; compression predominant crushers are most suitable for crushing extremely hard and abrasive rock (Utley, 2002).

Figure 3-4 shows a cross-section of a gyratory crusher and the main components mentioned before. Barry Wills (1997) refers to the cross-section representation of the gyratory crusher not only for a better illustration of its mechanism but also to correlate its operation principle with jaw crushers. He explains that at any cross-section, the gyratory crusher can be compared to two double toggle jaw crushers operating at opposite phases.

Similarly K. Gauldie (1954) used the jaw crusher operational principle to explain the gyratory crushing. He wrote: “The gyratory crusher may be regarded as a jaw crusher in which a large number of elementary, V-shaped jaws operate in succession. Each of these elementary jaws is advanced and retracted in each revolution and each contributes its share to the total output of the machine.”
The most advantageous characteristics of the gyratory crushers are as follows:
- the round shaped chamber provides higher capacities than jaws with the same gap,
- high capacities and the possibility of direct dump from haul trucks (Utley, 2002),
- low maintenance cost per ton processed and high availability (Utley, 2002),
- suitable for crushing hard ores up to 620 MPa (90,000 psi) compressive strength (Utley, 2002),
- tend to offer more flexibility with respect to moderating feed rates (Major, 2002),
- a more even power draw when compared to jaw crushers (Gaudin, 1939),
- low starting power peaks (Zandee, 1989),
- more uniform wear on liners than jaw crushers (Zandee, 1989), and,
- allow setting adjustment even when operating in automatic mode (Zandee, 1989).

Cone Crusher

The cone crusher, or reduction gyratory crusher, is basically a small scale gyratory crusher where the size of the mantle (head) is proportionally larger when compared to a gyratory crusher, and its outer surface flares out from top to bottom. This configuration results in a much flatter crushing angle than the one found in a gyratory crusher (Major, 2002) and provides an increased area of discharge to optimize throughput (Gaudin, 1939). The rotational speed is different too; cone crushers operate with speeds 2 to 3 times greater than the normal gyratory crushers.
The two most common types of cone crushers are the Symons and the Hydrocone, as shown in Figure 3-5. The Water-flush crusher is a design of cone crusher modified to allow the addition of water to the feed material.

In the majority of cone crushers, a mechanism to release tramp material is provided. In some cases the concaves are lifted, while in others the head is dropped momentarily - by means of coiled springs arrangements in the first case and hydraulic support in the latter.
Cone crushers provide consistent product size, which is an important advantage for their selection in the quarry industry. This aspect, and the reduced costs associated with this smaller piece of machinery, explains the greater number of studies found regarding cone crushers than for gyratory.

3.1.3 Gyratory crushers application

Gyratory crushers are the usual choice for primary crushing in high tonnage open pit operations. This trend is even stronger when the ore to be processed is hard and abrasive and the downstream phase requires relatively coarse material such as the feed for grinding circuits equipped with SAG mills (Major, 2002).

The gyratory crusher plays an important role in the link between the mine and the mill. The flow of material from mine to mill involves blasting, loading, hauling, crushing, milling and processing. Fragmentation and comminution occur during the blasting, crushing and milling steps of the process. In the overall context of the comminution process, the cost increases as the ore goes from blasting to crushing to milling (Wills, 1997).

Highland Valley Copper (HVC) has been closely analysing the parameters involved in the comminution process as a whole and has been very active in assessing the relationship between the mine and the mill operations. Experiments have been conducted and served to confirm that there is a direct correlation between the amount of fines in the
mill feed and its throughput. Figure 3-6 and Figure 3-7 show examples of the correlations that were observed at HVC in both SAG and AG mills during tests conducted by Dance. As shown in both graphs the mill production rate (Tonnage) is closely related with the amount of fines (Fines), (Dance, 2001).

Figure 3-6 Effect of Feed Size on AG Mill Tonnage (Dance, 2001)
Another test conducted by Dance at HVC served to confirm that the medium-size crushing product, also called critical size, plays an important role in mill productivity. With the use of an image analysis system for particle size measurements (discussed in detail in section 4.1), and tracking the flow of the material from the crusher until the semi autogenous grinding (SAG) mill, Dance confirmed the negative effect of the critical size in mill throughput. The graph in Figure 3-8 confirms that the amount of medium-size in the crusher product (●) and in the subsequent mill feed (□) are inversely correlated to the SAG mill throughput (Dance, 2001).
The effect of the critical size mill feed is so significant that doubts related to the design of previously accepted comminution flowsheets have arisen. Major (2002) underlines the fact that it had been common for operations to select and implement circuits containing gyratory crushers and SAG mills only; leaving out cone crushers in their comminution flowsheets. However, as he claims, the new trend seems to be the return of the use of cone crushers even in flowsheets containing SAG mills in order to crush "recirulating pebbles" (Major, 2002).

Major and Dance's work appears to suggest that the best way to solve the problems involving primary crushing product (grinding demands) would be the addition of more secondary crushers into the flowsheet.
However, analysing different facts reported by Dance and other authors who have worked on improving various crushing processes, there seems to be room for the alternative approach of optimizing primary crushing performance. This has been the focus of several authors (Flavel et al, 1988), (Svensson and Steer, 1990), (Burkhardt, 1982), (Dance, 2001). Flavel (1988) listed examples of successes obtained by several research programs and operations that achieved gains in grinding efficiency through improving crushing product quality. Some of these examples are listed below:

- Edmiston and Keller (1975) from Sierrita mine, Arizona, reported that the performance optimization of the crushing process, resulting from detailed analyses of the crushing parameters, increased the capacity of the concentrators from 65,336 to 78,040 tonnes per day.

- Excell and Fitzpatrick (1978) from Broken Hill Proprietary Co. mine at Whyalla, Australia, reported a 20% increase in grinding mill throughput attained by changing the cone crushing settings which also enabled a 15% improvement in crushing and screening plant throughput.

- “In 1961 and 1962, Bergstrom, et al hypothesized that, based upon research findings by Boliden Allis (previously Allis-Chalmers), efficient crushing processes could be used to significantly reduce overall comminution energy usage.”

Svensson and Steer pointed out that in mining operations, many times, inefficient crushing is easily “masked”, i.e. coarser crushing product imperceptibly flows directly to...
the grinding mills. In addition, they claim that on average crushing plants in the mining industry are less developed technologically than the ones in the aggregate industry (Svensson and Steer, 1990).

### 3.1.4 Gyratory crushers performance

Taggart (1927) discussed the capacity of gyratory and jaw primary crushers and concluded: "capacity depends primarily upon character of ore, size of feed and discharge setting. Throw, speed, angularity of jaws, and character of crushing surfaces have a material effect" (Taggart, 1927)

Detailed information about the factors influencing crushing performance is the object of Bearman and Briggs’ work. They stated: “crushers operate within a performance envelope encompassing throughput, product size and shape, and power consumed”. Though their work is based on cone crushers, the similarities between these types of crushers and gyratory crushers suggests that the results may be transferable to an analysis using gyratory crushers. In the analyses of crushing performance, Bearman and Briggs group the variables into three different categories: "Mechanical Design Variables (MDV) which do not change with time, Machine Operating Variables (MOV) which can be changed with time by the user, and Feed Material Variables (FMV) which may change significantly over short periods of time and are very difficult to control" (Bearman and Briggs, 1998).
In the same work, Bearman and Briggs underline the effects of liner material wear, wear profile, feed size distribution, feed type, and feed rate as time dependent factors that may be included in MOV or FMV and that significantly impact crusher performance over time.

Accordingly, following the above-mentioned categories some examples of gyratory crusher performance variables are:

- MDV: eccentric throw, fulcrum point position, speed (gyrations per time unit), and original chamber design\(^1\).
- MOV: feed rate and closed side setting (CSS).
- FMV: mechanical properties of the ore, feed size distribution, and choke feed level.

It should be noted that crusher feed is an important parameter to be assessed. Fortunately, different approaches are available in a number of works (Tunstall and Bearman, 1997), (Burkhardt, 1982), (Dance, 2000).

“Well-fragmented material passes easily through the primary crusher, maximizing crusher productivity and minimizing power consumption, liner wear and mechanical breakdowns” (Tunstall and Bearman, 1997). The amount of fines generated by blasting has a definite impact in crushing performance (Burkhardt, 1982). Table 3-1 shows results

\(^1\) If considering wear profile, this parameter might be categorized as MOV (this variable is detailed later in this section).
from tests done by Burkhardt that indicate that as the ratio of fines increases in the feed, crusher throughput rises. Furthermore, it can be concluded that primary crushing performance is intimately linked to the degree of fragmentation achieved by blasting (Tunstall and Bearman, 1997). In fact significant efforts have gone into optimizing blast design for crusher feed quality control (Tunstall and Bearman, 1997) (Dance, 2000) (Valery et al, 2001).

Table 3-1 Increase in crusher throughput by changing feed characteristics (Burkhardt, 1982)

<table>
<thead>
<tr>
<th>Test No.</th>
<th>Feed (mm)</th>
<th>TPH</th>
<th>Percent Change</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>-533 +125 (No fines)</td>
<td>655</td>
<td></td>
</tr>
<tr>
<td>2</td>
<td>-508 + 0 (20% -125)</td>
<td>763</td>
<td>+16</td>
</tr>
<tr>
<td>3</td>
<td>-483 + 0 (40% -125)</td>
<td>887</td>
<td>+35</td>
</tr>
</tbody>
</table>

Since blasting costs are smaller than crushing and milling costs, crushing performance can be improved by increasing the quality of the feed (Wills, 1997), (Burkhardt, 1982), i.e. increasing blasting effectiveness is a good way to optimize the overall process.

Following this suggestion, in 1998 HVC conducted a field trial to evaluate different blast patterns. An area was divided in two halves and subjected to different blast patterns, one following the standard design and the other following a design with a higher powder factor. The material was tracked and sent to the same milling line (C line). The results were contrary to what was expected. The finer blasted material did not generate higher tonnage at the mill, instead the inverse occurred (Dance, 2000).
By analysing what happened during the blast pattern trial it was realized that the finer
blasted product rapidly slipped through the crusher producing a courser product than the
one achieved with the coarser starting material from the standard design. Following the
trend in the mining industry, HVC had been operated the primary crushers to always get
maximum tonnage and avoiding a bottleneck condition, but this test served to bring more
attention to the reality of the trade off between tonnage and product quality.

After thorough analysis, HVC realized the importance of the gyratory crusher in the
overall process: “unless we can maintain the finest crush possible, any gains to be made
in blasting finer could be lost before it reaches the mill” (Dance, 2000). The blasting trial
served to exemplify product quality degeneration from loose operation of the crusher, i.e.
attention was given to one variable (feed size distribution) but others were forgotten (feed
rate and choke feed level).

Obviously one way to achieve choke fed crushing is by control of the feed rate. This is
done with apron feeders ahead of the crushers. Using this crushing configuration, the
feed rate can be considered an operating variable and its optimization is possible.
Taggart described the optimum feed rate as being one that brings the machine as near as
possible to its capacity (Taggart, 1927).

Burkhardt accepts Sheppard and Witherow’s recommendation that constant choke
feeding increases the amount of finer product and adds that the maximization of gyratory
crusher product quality can be achieved by operating the machine at small CSS and choke fed condition (Burkhardt, 1982), (Sheppard and Witherow, 1938). Choke fed condition is also recommended for cone crushers, and “means that the volume above the crushing chamber is always full of material which flows into the crushing chamber at a pace decided by the crusher” (Svensson et al, 1996).

The accepted reasons supporting the choke fed condition for cone crushers are listed below (Bearman and Briggs, 1998).

- “develops a uniform wear profile;
- gives a more consistent product sizing;
- maximises throughput;
- extends the liner life;
- improves the cubicity of the product”

The gap or CSS is obviously an important variable, and is categorized as an MOV. Being part of the control of the machine CSS adjustment should be done carefully to optimize crusher performance.

Work on the control aspect of cone crushers for the aggregate industry has been done in an attempt to predict and control the output product size (Moshgbar et al, 1995), (Bearman and Briggs, 1998). This work has relied on a mathematical model developed by Whiten to predict the rock breakage during crushing (Whiten, 1972). Using Whiten’s
model Moshgbar developed a differential equation to describe the crusher product output as a function of wear and therefore gap variation.

Figure 3-9 shows the results using this model to simulate crushing with no control of the gap, manual control of the gap and automatic control of the gap. In this figure DS is the wear effect on the gap, %CS is the percentage volume of product with the required optimum size and %OS is the percentage volume oversized.

The simulation results show that with no adjustment, the gap gradually increases, as expected, due to crusher wear, and the product quality degrades with time. These results demonstrate that manual adjustment of the gap setting at predetermined intervals
improves the product quality, but at the cost of downtime. Finally, automatic control provides the best product control and the most consistent gap setting (section 3.2 contains more details about wear effect over gap adjustment procedures).

Tests conducted at HVC demonstrate that by operating the crusher with a fixed setting the variation in feed size distribution results in similar fluctuation in product size distribution. Figure 3-10 shows the effect of feed size on product sizing for the crusher operating with a constant setting (Dance, 2001).

![Figure 3-10 Crusher Feed & Product % Course - Constant Setting (Dance, 2001)](image)

Chamber design, or cavity design, has been considered a key parameter in crushing performance by both the manufacturers and users of cone and gyratory crushers (Gaudin, 1939), (Burkhardt, 1982). In addition, some of them also claim that with the right design
of the liners, wear may be minimized and made more uniform along the profile (Bearman and Briggs, 1998), (Westerfeld, 1985), (Svensson and Steer, 1990).

Westerfeld discussed the advantages (including the ones mentioned above) of the design of curved concaves also called non-choking concaves\(^2\). He demonstrated that this design of the concaves offsets the choke point to a higher position compared to the position achieved with straight concaves. Thus it minimizes excessive level of stress at the bottom of the chamber, which is responsible for localized and rapid liner wear in this region (Westerfeld, 1985).

The drawings in Figure 3-11 were used by Westerfeld to explain the differences in material flow between the two configurations of crushing chambers. It is assumed that the two crushers have the same eccentric throw and the same discharge setting. The difference between the chambers is provided by the different profile of their concaves: the crusher on the left side (1) has a straight concave and the crusher on the right side (2) has a curved profile. Crusher 1 represents a straight chamber and crusher 2 represents a non-choking chamber.

To analyse the crushing action in these two chambers, it is first assumed that both crushers are completely filled with a friable material and second, that the crushing action occurs in steady steps as numbered. Thus, at each step (or complete gyration of the

\(^2\) "Even though the design is called non-choking it does not afford absolute insurance against choking" (or blockage) "inasmuch as a choke point exists in the crushing chamber" (Westerfeld, 1985).
mantle) a volume of material is compressed in one region and then moves to a lower region. As can be seen from the figures, in crusher 1 the volumes successively decrease from region 0 to the bottommost region 19. So, region 19 has the highest probability of packing the material and therefore the choke point in a straight chamber is at its discharge level.

On the other hand, for crusher 2 the volumes successively decrease only from region 0 to region 14. Thus, this region has the highest probability of packing the material and as shown in the figure the choke point of a non-choking chamber is located above the discharge level.

Figure 3-11: Straight versus non-choking concaves (Westerfeld, 1985)
Following Westerfeld's work, Svensson and Steer described the mechanisms of the crushing process inside the chamber with the aid of slices and areas to introduce the constant liner performance (CLP) crushing chamber concept that has been used in cone crushers. They suggest that the advantage of this enhanced chamber design is that it "controls" the liner profile wear in such a way that "the feed opening and the capacity is maintained almost constant throughout the life of the liners" (Svensson and Steer, 1990). Figure 3-12 shows this type of chamber when new and at the end of its life.

![Figure 3-12: CLP liners (Svensson and Steer, 1990)](image)
3.2 Liner wear

As described in the previous section liner wear directly impacts the gap dimension as well as the chamber profile (Bearman and Briggs, 1998), (Svensson and Steer, 1990), (Westerfeld, 1985). Hence, liner wear is an important variable in the overall crushing operation as it is intimately related to product quality (size consistency and throughput) and cost. The rate of wear and its distribution among the different regions of the chamber results in profile modifications affecting liner life. Moreover, not only the replacement cost of the liners but also the costs associated with the variations in crusher performance can be attributed to liner wear.

In addition, lack of an effective real time measurement system for the wear during operation complicates the application of a systematic adjustment of the gap over time. This lack of accurate gap information may result in either running the crusher too tight, causing a reduction in throughput, or running the crusher with a gap that is too wide, resulting in poor product quality. As a result, unintentionally operating the machine too tight may accelerate the deterioration of the crusher’s drive components and once more impact costs.
3.2.1 Mechanisms of wear

The mechanisms of crushing wear, the variables involved in it, as well as efforts in modelling its behaviour have been the focus of other works in the application of cone crushers (mainly in the quarry industry); (Delalande, 1986a), (Moshgbar et al, 1994), (Moshgbar et al, 1995), (Bearman and Briggs, 1998). Since system kinematics for cone and gyratory crushers are quite similar, the results from these studies are of great value for an investigation of wear mechanisms in gyratory crushers.

During crushing, the rock particles are in rolling, impact and sliding contact with the liners of the machine (Moshgbar et al, 1994). Hence the wear of the liners is inevitable and is caused by gouging/ploughing (Bearman and Briggs, 1998). Following a more comprehensive tribological study, it has been determined that the system has a dominant open three-body abrasion wear mechanism, i.e. the type of wear “associated with the abrasion of one or two surfaces of moderate separation by abrasive particles which can move relative to each other as well as rotating and sliding over the abraded surfaces”(Moshgbar et al, 1994).

In addition, as the three-body abrasion can be divided into gouging, high-stress and low-stress regimes, Moshgbar concluded that in cone crusher systems there are basically two wear regimes: low-stress at the top of the crushing chamber changing to high-stress as it gets closer to the bottom region where the main crushing zone is located. This theory is
supported by the commonly observed larger loss of material in the main crushing zone (Moshgbar et al, 1994).

There is general agreement between several researchers about the variables that affects wear rate and the reasons for uneven-wear profiles (Parks and Kjos, 1991), (Bearman and Briggs, 1998), (Moshgbar et al, 1995). Of main interest are those listed below:

- material properties of the liners,
- properties of the feed (i.e. chemical composition, strength and moisture content),
- operational parameters e.g. CSS, power, feed rate and crusher chamber.

### 3.2.2 Liner materials

Cone crusher liners are usually made of an austenitic 12% Manganese steel namely Hadfield steel (Moshgbar et al, 1995), the same material had been exclusively used for both concaves and mantles of large gyratory crushers since their invention until the late 1960’s when the first upper row concaves made of martensitic cast steel were introduced. Nowadays concaves and mantles can be cast from several different ferrous materials. The alloys most used are: martensitic steels, martensitic Cr-Mo steels, Ni-Cr (Ni-Hard) white irons and austenitic manganese steels (Parks and Kjos, 1991).

Chrome white irons are frequently preferred for gyratory crusher concaves because of their high abrasion resistance, their high yield strength that minimizes plastic deformation in service ("growth"), and their cost effectiveness (Esco, 2003), (Parks and Kjos, 1991).
Austenitic manganese or Mn-Cr steels and the martensitic Cr-Mo steels are the alloys most recommended for gyratory crusher mantles (Esco, 2003).

During the design and/or selection of alloys to be applicable in sacrificial wear components in the mining industry, the relationship between abrasion resistance and toughness must be considered. Generally, materials showing high abrasion resistance are hard and brittle. An exception is the austenitic manganese steel which combines both requirements, originally austenitic manganese is relatively soft with 200 HB (Brinell hardness) but under certain work conditions its hardness can increase to more than 500 HB which improves its abrasion resistance while retaining its desirable toughness (Parks and Kjos, 1991); (Diesburg and Borik, 1974).

Suitable work hardening conditions as mentioned before seem not to be consistently achievable for the concaves under normal operation conditions in gyratory crushers. This limits the application of austenitic manganese steels for concave applications. The martensitic steels and white irons commonly exhibit high hardness values (typically from 500 to 600 HB) even at normal work conditions and have been commonly used for concave liners. Additionally, martensitic alloys have longer wear life than austenitic manganese steels, though their impact toughness is much lower than the one exhibited in austenitic manganese steels (Parks and Kjos, 1991).

For material assessment, several laboratory wear tests have been developed (pin-on-drum, dry-sand rubber wheel, jaw crusher, and impeller drum) to assess abrasive wear
The US Department of Energy has compared results for different materials from laboratory wear tests to field test data developed using a “Planar array field test”. Basically, a plate is designed with various steel samples and placed in the conveyance section of a mineral processing crushing circuit where it will be exposed to impact and sliding material at different angles and velocities (Tylczal et al, 1999). Although interesting from a material evaluation point of view, this paper does not provide a wear rate model that can be applied to gyratory crushers. It does suggest that laboratory tests are beneficial in evaluating material wear, but care must be taken to ensure that laboratory test mimics the wear process occurring in the actual application. For example the crusher test, which is a gouge test, does not correlate well with the “Planar array test” primarily due to the field test being an abrasive wear test.

3.2.3 Wear measurements

As previously discussed, crusher liner wear influences product quality and is closely linked to the overall effectiveness of the process, therefore impacting crushing total cost. Moreover, the lack of accurate ways to measure the wear during operation, limits the efficacy of automated adjustment of the CSS resulting in undesirable variability of product quality over the lifetime of the liners.

Gyratory crushers are large-scale types of equipment, which contain robust moving parts that gyrate eccentrically. In addition, the crushing feed usually contains large pieces of
abrasive tough rocks that are directly dumped at the top region of the machine. Currently it is impossible to assess liner deterioration in real time for gyratory crushers. The intrinsic characteristics of these types of machines and of the feed they process can provide an idea of the limitations in achieving a desirable online wear measurement.

In the following sections descriptions of some examples of liner wear measurement techniques are given. These procedures have been applied by research labs and/or by industry and they serve to better describe the difficulties that are involved in this task.

**Drilling Holes**

Parks and Kjos describe the periodic use of profile measurements on worn liners as a way to monitor wear behaviour and assist in the selection of mantle profiles to be used over time (Parks and Kjos, 1991). They describe the technique of drilling holes of small diameter at the seams in between the concave parts (through the soft backing-material until the drill reaches the shell) to perform posterior measurements of the holes with the aid of pieces of wire. The limitations of this procedure are quite straight forward: the evaluation is only possible at discrete points, observation of scrap concave parts reveals different thicknesses close to the edges, mantles are mounted in a maximum of three pieces resulting in a low number of radial seams, large inaccuracy caused by the nature of the procedure, and time consuming.
Mines use a similar technique with the main difference being that holes are drilled directly into the liners (Adams, 2003). Although this technique can be better applied to mantles and may avoid the error incurred when measuring thicknesses close to the edges, it suffers from most of the disadvantages of the original procedure. In addition, it may create stress concentrators and lead to the deterioration of liner life.

**Ultrasonic Thickness Gauge**

There have also been attempts to measure the thickness of worn liners with the use of ultrasonic gauges (Parks and Kjos, 1991), (Adams, 2003). Difficulties and limitations mentioned by Parks and Kjos include the necessity of selecting sections with known parallel wear surfaces where the measurement is made to avoid loss of back reflection. Adams (2003) reported that HVC performed some tests in the past but the results were inaccurate and most of the problems were related to reflections and/or originated from the backing materials.

**Experiments to Measure Cone Crusher Chambers**

Delalande first showed interest in determining the optimum operational period for cone crusher liners following an observation of the degeneration in product quality over time and an apparent opportunity for cost reduction when shortening the life of the liners (Delalande, 1986a). In a later paper, Delalande reported that to continue the investigation it was necessary to get an understanding of the dynamics of wear across liner lives, he
then described three different methodologies to obtain the chamber profile of the crusher at the closest plane of use (Delalande, 1986b). Since the methods were designed for laboratory scale tests there are obvious limitations for easy adaptation of the methods to large primary crushers. However, the functional principles may serve to provide the foundations for other novel approaches for large-scale gyratory crushers. The three methods are briefly described in the following:

1- One method to obtain the chamber profile utilises a profiler device, this apparatus contains a horizontal bar (fixed at the top of the crusher) that support a second articulated bar which lies inside the chamber, a measure device similar to a tape measurement which slides over the second bar, and an electrical system that reads and records the inclination of the bar and the distance measurements over time. Basically the functioning of this apparatus is in reading the length and the inclination of the articulated set of measuring bars as its extremity travels touching the surfaces of the two opposite walls (concave and mantle).

2- Another method is called profile by casting. In this procedure a cylindrical latex bag is inserted inside the chamber and kept in a fixed position with the aid of a metallic tripod fixed at the top of the crusher and a rod linked to the tripod and kept inside the bag. The bag is filled with a pre-mixed two-component-resin (approximately 50% in volume) and the mixture is let to expand and solidify. The cast is cut in a predetermined manner, removed from the crusher and then the profile can be analysed.
3- The third method is called profiler by direct sketch. In this method a wood-board is patterned in the approximate shape of the crushing chamber and placed in the crusher. Once the board is in place, a compass with a fixed opening is drawn along the profile of the concave or mantle and its shape is traced onto the board.

**Sacrificial Sensors**

An online methodology to monitor the wear in cone crusher liners for the quarry industry has been the object of research and development for several years. The methodology would complement the implementation of a full condition monitoring system in order to achieve several operational benefits such as the optimum utilisation of the liners, product quality enhancement with the use of an automatic adjustment of the gap and feed rate, as well as the reduction of maintenance downtime (Moshgbar et al, 1995); (Yaxley and Knight, 1999).

In this wear measurement methodology sacrificial sensors of approximately 0.5 mm diameter are embedded in different regions over the liners. The sensors wear away at the same rate as the metallic liners. Each sensor sends a signal that corresponds to its current length which enables the system to give an accurate representation of the wear at real time. Different prototype sensors have been developed using capacitive, resistive and conductive principles, and for each configuration laboratory and field tests have been performed. It was concluded that the most promising configuration is the one that applies multiple surface mount resistive sensors. Although the field tests showed problems such as short-circuiting of the sensors and signal spikes it is expected that with further
development the reliability and accuracy of the multiple surface mount resistive sensor may be improved and this measurement methodology may be used in industrial applications.
4. Highland Valley Copper Project

4.1 Operations Background

HVC is a Teck Cominco and BHP open-pit mine operation located near the town of Logan Lake in the southern interior British Colombia, Canada. HVC is one of the largest copper mining operations in the world. In 2002, HVC mill achieved 50 million tonnes of total throughput (on average 137,000 tonnes per day), the largest throughput in 20 years of operation (Teck Cominco, 2003).

At the beginning of 2000, ore reserves totalled 387 million tonnes at a grade of 0.417% copper and 0.009% molybdenum - gold and silver are present in small quantities that become noteworthy in the copper concentrate. On average, the strip ratio is 1:1 resulting in 270,000 tonnes mined per day in two pits simultaneously. Valley, the main pit, is located 3 km northwest of the mill plant and contains approximately 74% of the reserves; the remaining ore comes from the Lornex pit, located 1 km southwest of the mill plant (Richards, 2000).

Mineralization is bornite and chalcopyrite for both pits. Valley pit shows a higher ratio of bornite to chalcopyrite and the reverse occurs in Lornex; Valley pit has a lower molybdenum sulphide mineralization than the Lornex pit (MacPhail, 1992).
Being a high tonnage and low grade mine, HVC's existence relies heavily on economies of scale. HVC has always applied innovative technology to increase productivity and to reduce operating costs. Some examples of such advanced technologies in the history of HVC are the application of a computer based truck dispatch system, movable in-pit crushers and conveyors for ore transportation in the Valley pit, Global Positioning Systems (GPS) location technology on drills and shovels, shovel weighing system and fragmentation image analysis system.

The overall milling process can be visualized in Figure 4-1. The ROM ore from the Lornex pit is trucked and directly dumped into a fixed 1.52 by 2.26 m (60 by 89 in.) Metso Superior Gyratory crusher; the crusher is driven by a 520 kW (697 hp) motor and is equipped with a heavy duty hydraulic hammer at its top to deal with over size material and to help clear blockages. The Lornex crusher is designated Crusher No. 1.
The ROM ore from Valley pit is trucked and dumped into two 1.52 by 2.26 m (60 by 89 in.) Metso Superior Gyratory crushers, No. 4 and No. 5. These crushers are semi-mobile and located deep in the Valley pit. Differently from No. 1, the in-pit crushing layout includes a dump hopper and a 2.44 m variable speed inclined apron feeder which avoids truck direct feed and enables feed rate control.

Previously, all three gyratory crushers reduced the ore to app. 250 mm (~ 10 in.). However, for the past few years, the crushers have been operating to reduce the ore size
to app. 150 mm (~ 6 in.) with the use of CSSs ranging from 127 to 140 mm (5 to 5.5 in.). As commonly described at the mine, the crushers were operating “loose” in the past but now they have been operating “tight”.

A network comprised of several kilometres of conveyor belts, feeders and surge piles is used to deliver the crushed product to three stockpiles located just beside the mill plant. Although there are limited crossovers between the different crushed products in the conveying phase, some blending of the grinding feed is possible if desired. Three variable speed apron feeders and two hydra stroke feeders are used to transfer the material from the three stockpiles (1, 2 and 3) to five grinding lines (A, B, C, D and E).

The grinding lines were built at different times and consequently, there are substantial differences between them. A and B lines are similar, each of them is comprised of one primary SAG and two ball mills. C grinding line is similarly equipped with one SAG mill and two ball mills. The mills in C are larger than the mills in A/B lines and line C utilises a cluster of ten cyclones instead of seven for the A/B lines. Each of the other two grinding lines, D and E, consists of one fully autogenous mill (AG) and one ball mill. Each AG mill is equipped with one discharge grate and one vibratory double deck screen for the removal of critical size rock particles. A 2.1 m (7 ft) Symonds short head cone crusher is used in closed circuit to crush the oversize material from the screen in each line, and the ball mill operates in close circuit with a cluster of ten cyclones.
After comminution, flotation cells are used to produce a concentrate that contains both copper and molybdenum. Regrind circuits, comprised of ball mills and cyclones, are additional components of the flotation cells. In the final processes, molybdenum is separated from the bulk copper-molybdenum concentrate by flotation and leaching and the final products are arranged for shipment.

As mentioned before, several advanced system are applied at HVC. The systems relevant to this research are discussed next.

HVC utilises an image analysis system to monitor the size distribution of crusher feed, crusher product and the feed of the grinding lines. The system consists of several video cameras mounted in strategic locations and a PC-based fragmentation analysis system developed by WipWare Inc. called WipFrag.

The software captures and digitises images of the material and isolates individual fragment boundaries as shown in Figure 4-2. The results of this fragmentation recognition are used to calculate particle areas, volumes, masses and the size distribution by weight.
The system has some limitations such as the inability to recognize fine particles (smaller than 15mm) and the need for controlled lighting conditions (in some location this is solved by the addition of halogen lamps). However, these restrictions do not compromise the objective of its application at HVC, as WipFrag outputs are mainly used as control signals. Even though the output accuracy is not the same as from lab screening analyses, WipFrag output has proven to be repeatable and reliable for its designed application (Simkus and Dance, 1998). The use of WipFrag output as a control signal is possible because, in this case, relative changes in the distribution are more important than the comparison of the signal to a “standard” sieve analysis (Simkus and Dance, 1998).

HVC relies on state-of-the-art systems to monitor ore properties. At HVC, all the drills are equipped with GPS-based navigation and blasthole guidance systems as well as material recognition system from Aquila Mining Systems. The Aquila material
recognition system provides rock characteristics by the analyses of drill parameters such as rate of penetration and vibrations.

Since 1997, HVC has been utilizing the Citect process control system from Citect Pty Ltd, Australia. Using Citect, operational data is gathered automatically from equipment instrumentation and then processed and recorded at small time intervals. Citect facilitates the search for real time and historical information as well as enables the overview of the entire operation from several workstations at the mill. The system comes with a detailed graphic user interface and enables the creation of tailor made reports.

Using Aquila, Citect, Dispatch, WipFrag and other technology systems HVC has achieved the capacity to track ore properties throughout the crushing and grinding processes. More details of a study conducted at the mine with regards to the systems mentioned can be found in (Simkus and Dance, 1998).
4.2 Crusher Operations at HVC

As described before, every day a massive amount of material is milled at HVC (on average 137,000 tonnes). However, only three crushers are responsible for all the ore processed from the two pits. This condition by itself underlines how crucial the availability, maintenance and proper operation of the crushers is. Maintenance problems like unscheduled shutdowns may affect the profitability of the entire operation, while improper operation conditions at the crushers decrease the final throughput of the mill.

The dynamic nature of the crushing process is fundamental to a desired flow of material from mine to mill. Though fragmentation and comminution occurs during blasting, crushing and milling steps of the process, there is a better opportunity with regards to costs in optimizing the first two processes since the cost increases as the ore goes from blasting to crushing to milling (Wills, 1997).

HVC has been very active in enhancing the product quality resulting from both blasting and crushing, aiming for an overall improvement in performance of the comminution process. As discussed in section 3.1.4, in 1998 HVC initiated this optimization development by the investigation of the effects in mill productive by application of enhanced blasting designs. This work resulted in the recognition of the key importance of the primary crushing process in the overall comminution process at the mine.
In 1999, drawing on advances in cone crusher technology, HVC initiated the development of an automatic control system for the crushers working in the Valley pit. Flavel demonstrated that the use of automatic CSS controls in a cone crusher substantially improved both the capacity and the quantities of finer sized product (Flavel, 1982). Figure 4-3 shows the difference between two similar sized cone crushers operating with and without automatic CSS regulations. In this case, the crushers are used in the secondary crushing process and they are equipped with screens to sieve the feed.

As shown in Figure 4-3, the machine equipped with automatic CSS control achieves a net production of 775 tph and has 48.9% of the discharge product passing -13 mm (-0.5 in.). On the other hand, the machine with fixed CSS achieves a net production of 383 tph and...
has 35.3% of the discharge product passing -13 mm (-0.5 in.). Flavel explains that when using a fixed CSS the average operating power drawn is usually restrict to app. 50 percent “of that connected to guard against crusher stalling and minimize mechanical damage” and the application of the automatic control allows the machine to normally operate at much higher average power rate. As shown in Figure 4-3, there is a considerable difference between the operating power drawn for the two machines.

The control system of the crushers at HVC aims to maintain a constant choke fed operating condition and to keep the product size distribution within a predetermined quality range. The design and layout of the semi-mobile crushers, containing the dump hopper and the variable speed apron feed, enables the application of such an automatic control system.

A fuzzy logic-based control algorithm is the basis of the system. Based on a group of operational parameters, the algorithm adjusts the apron feeder speed and the vertical mantle position at 30 seconds intervals. The operational parameters that serve as inputs for the system are as follows:

- dump hopper level,
- crusher pocket level,
- crusher motor power,
- product size distribution, and,
- tonnage.
Figure 4-4 shows the system graphical interface available for the operator and gives an example of real measurements and outputs. In this example, it is possible to observe that the dump level was at 46%, the pocket level at 36%, the tonnage was 4248 tph, the motor power drawn was 77.0 amps and the product size distribution showed 30%-37%-30% moreover, the system outputs were: 31.1% for the apron feeder speed and 109 mm (4.3 in.) for the mantle position.

![Figure 4-4 Automatic Crusher Control Graphic (Dance, 2001)](image)

When a higher tonnage is requested in detriment of quality product, for example, when one of the crushers is down and/or the level of the stockpiles are low, a special system mode can be activated where the high tonnage is given priority and crushing is loose.
On the other hand, when the two crushers are in full operation and the frequency of the trucks is low, the control favours improved product quality and the crusher operates with a "tighter gap" (small CSS). Motor power and oil temperatures frequently serve as "health" parameters of the machine and, when extreme conditions are perceived, the system lowers the mantle and decreases the speed of the feeder until normal conditions are restored.

Being a crucial determinant of crusher operational conditions, the vertical mantle position adjustment varies only within predetermined limits when the crusher is set for automatic control. To determine the mantle position limits, frequent direct measurement of the gap is necessary. In practice, what must be verified, as frequently as possible, is the relationship between a set of mantle positions (for example 50, 100, 150, and 200 mm) and their corresponding CSS dimension. This assessment is valuable because it serves as a tool to forecast which range of mantle vertical positions would provide the acceptable range of CSSs.

As mentioned in the literature review, the wear that liners are subjected to modifies the chamber shape and the gap over time (refer to section 3.1.4), therefore accurate forecasts of CSS versus mantle position are virtually impossible unless online measurements are available.
HVC has been assessing the relationship between CSS and mantle position every week using a measurement procedure known as the “bucket test”. In this procedure, several metallic buckets (filled with sand) are thrown into the crusher while it is operating empty. The process is repeated for two or three predetermined mantle positions. By evaluating the bucket size before and after, the gap variation as a function of mantle position can be estimated. Typical results of this procedure are shown in Figure 4-5. Although effective, the method requires the crusher to be down, lacks accuracy, and does not provide chamber profile information details.

![Crusher 4 Mantle Position vs CSS](image)

**Figure 4-5** Gap measurement results and analysis performed at HVC

3 The units presented in graphs and in HVC’s reports shown in this work follow the U.S. customary system because this is the system commonly used in the mine.
In order to create a better alternative to the “bucket method” for measuring the gap, and to develop an accurate method to collect comprehensive wear data, HVC purchased a prototype laser profiler device developed by Conveyer Dynamics, Inc. The equipment, its application, and measurements results are discussed in the next section (5.1).
5. **Experimental Approach**

5.1 **Equipment**

The equipment used to measure crusher wear and the chamber shape was a laser profiler device (LPD). HVC purchased a prototype device of this type from Conveyer Dynamics, Inc. (CDI). It should be noted that the LPD is the first of its kind used to measure a crusher chamber profile. Figure 5-1 shows the major components of the LPD and Figure 5-2 shows a schematic of its installation inside the crusher. Basically the LPD is comprised of:

- a support structure to mount it to the crusher,
- a track for the laser to run on,
- an additional structure containing five calibration bars,
- an actuator motor to drive the laser up and down the track,
- a time of flight laser with a mirror for reversing the target direction, and,
- software and a computer to collect and process the measurement data.
Figure 5-1 Major components of the LPD

Figure 5-2 LPD Installation schematic
The laser is a DME 2000 Distance Measuring Device from SICK Optic-Electronic Inc. The device optically measures the distance to a target (opaque) by transmitting a modulated red laser beam and measuring the time of flight of the beam. The laser is used in its proximity mode with a maximum range of 2 m (6.7 ft). The output signal is 4-20 mA with a resolution of 1 mm, a repeatability of 0.8 mm and an absolute accuracy of +/- 5 mm. CDI assures that the absolute accuracy is improved to 1 mm by the use of the calibration bars to yield a correction factor. The output signal of the laser is sent to an analog input of the actuator motor for transmission to the PC (Nims, 2001).

The actuator motor is a SilverMax® “E” from Quicksilver Control Inc. This actuator motor communicates as a slave to a PC. The PC polls the actuator motor for the contents of the position/distance data. On average, a pair of coordinates containing the laser position on the track and the distance to the target is obtained approximately every 3 mm along the track (this can vary depending on the velocity used).

The user interface, CDI Laser Scanner Program, is written in Visual Basic. This program executes all the necessary functions to run the tests, to acquire data and to provide final profiles.

The laser device provides an excel spreadsheet for each test performed. The excel file contains data representing several points from the surface of the liner given as pairs of
coordinates. Table 5-1 shows an excerpt of the spreadsheet (first 5 points from a concave measurement).

### Table 5-1 Example of a partial table result for a concave profile measurement generated by the LPD

<table>
<thead>
<tr>
<th>Actuator Position ($Y_L$) (mm)</th>
<th>Laser Distance ($X_L$) (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>3.78</td>
<td>1291.48</td>
</tr>
<tr>
<td>7.14</td>
<td>1287.92</td>
</tr>
<tr>
<td>11.34</td>
<td>1287.17</td>
</tr>
<tr>
<td>14.7</td>
<td>1280.42</td>
</tr>
<tr>
<td>19.32</td>
<td>1279.74</td>
</tr>
</tbody>
</table>

As shown in the example, each point has its actuator position on the track ($Y_L$) in the first column and the laser distance to the target ($X_L$) in the second column. These coordinates are not Cartesian X, Y pairs of coordinates. Figure 5-3 illustrates the measurement process.
During the crusher profiling procedure four types of measurements are taken:

1\(\text{st}\) - shooting the calibration bars with the laser beam perpendicular to the track;

2\(\text{nd}\) - shooting the bars after levelling the mirror;

3\(\text{rd}\) - shooting the concaves of the crusher; and,

4\(\text{th}\) - shooting the mantle after rotating the mirror 90 degrees.

The first two measurements are performed to calibrate the LPD prior to acquisition of the liner measurements. The CDI software uses these two calibration tests to calculate the angular dimension formed by the actuator and the vertical centre line of the crusher, or inclination angle of the track (\(\alpha\)). This angle is used to map the laser coordinate system \((X_L, Y_L)\) to a Cartesian system. The calculation is made possible by comparing the results from the first two types of measurements when the laser shoots the same bar; at first
having the laser mirror base perpendicular to the track and second having the laser mirror base levelled horizontally. Figure 5-4 illustrates the procedure.

![Figure 5-4 Calibration procedure schematic](image)

Once the inclination angle of the track, \( \alpha \), is calculated the software generates the profiles of the mantle and the concaves as polylines in an AutoCAD format. Figure 5-5 and Figure 5-6 show an example of a concave and a mantle profile, respectively.
The profiles are transferred to a section-view drawing of the crusher and aligned with the original liner profiles in order to compare the wear. Figure 5-7 shows an example of the
As expected, during the initial use of a prototype device, some issues needed to be addressed. One major problem with the LPD was the time necessary to perform the
measurements, in particular the set up time. It was identified that the excessive set up time was related to the removal of the crusher spider cap in order to install the LPD support. The set up time was significantly reduced by the redesign and construction of a new support structure. Figure 5-8 shows a picture of the original support on the right and a picture of the new support on the left. The actual drawings for the new support are included in Appendix A.

The new support structure was designed such that:

- it is installed around the mainshaft and there is no need to remove the spider cap (necessary with the original structure),
- it is lighter than the original, dismissing the use of the crane for its transportation,
- it provides a new mechanism of connecting the track to the support, minimizing the time spent during set-up,
- there was low costs involved in its construction and its design allowed an in-house construction. Thus, one could be built for each crusher.

Another difficulty was related to the generation of two separated profile drawings, one for the mantle and one for the concave in a horizontal position. To get the two profiles together following their real inclination as well as to solve some issues with the original software it was decided to develop a new program.
5.2 Data Collection

5.2.1 Crusher operational data

Crusher operational data from the Citect system was collected during periodic field visits. The data was collected for crushers 4 and 5 from the 1st of January 2001 to the 30th of September 2002 and included the following items:

- Crusher Bowl Level
- Mantle Position
- Motor Current Draw (Amps)
- Production (mtph. at the conveyor)
- Feed Size Classes (course, medium and fine)
- Product Size Classes (course, medium and fine)

Data points were obtained for one-hour periods. The Citect system processed the records for each 30-second interval to produce an hourly average of the 120 records. In addition, the maximum and minimum values within the hour were also recorded and collected.

5.2.2 Liner information

In order to assess the impact of liner life on maintenance costs, the costs involved in liner replacements were collected. The costs reported by the maintenance department for Feb/2002 are listed in Table 5-2 and will serve as a basis for liner management analyses.
Table 5-2 Liner rebuild/installation: labour and costs (Wolff, 2002)

<table>
<thead>
<tr>
<th>Mantles- Rebuilt</th>
<th>Concaves – Installation</th>
</tr>
</thead>
<tbody>
<tr>
<td>Labour/Parts</td>
<td>Cost (Cn)</td>
</tr>
<tr>
<td>Liners 4 Row</td>
<td>$30,550</td>
</tr>
<tr>
<td>Supplies</td>
<td>$4,000</td>
</tr>
<tr>
<td>Labour (144 man hrs)</td>
<td>$5,500</td>
</tr>
<tr>
<td>Installation (48 man hrs)</td>
<td>$1,850</td>
</tr>
<tr>
<td>Total* (12 hrs down time)</td>
<td>$41,900</td>
</tr>
</tbody>
</table>

For more than three years, HVC has been using the same type of concave liners for Crushers 4 and 5. The concaves are supplied as a four-row set made of high chrome white iron by Penticton Foundry Ltd.

Information and drawings for several different mantle types are available at the mine. As shown in Table 5-3, basically three sizes are available: under-size, standard-size, and over-size. In addition, as different designs, diameters and configurations (two or three pieces) exist, a total of ten types are listed. Although these ten types of mantle have been used at HVC, only the eight types shown in bold were used during the period analysed.

Table 5-3 Mantle types

<table>
<thead>
<tr>
<th>Mantles</th>
<th>Maximum Diameters</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Under-size</td>
</tr>
<tr>
<td>Vendor/Type</td>
<td>mm (in.)</td>
</tr>
<tr>
<td>Esco/2pcs</td>
<td>2184 (86)</td>
</tr>
<tr>
<td>Esco/2pcs (Type II)</td>
<td>---</td>
</tr>
<tr>
<td>Esco/3pcs</td>
<td>---</td>
</tr>
<tr>
<td>Frog Switch (Transwest)/2pc</td>
<td>---</td>
</tr>
<tr>
<td>Columbia/2pcs(with ribs)</td>
<td>---</td>
</tr>
</tbody>
</table>
Mantle and concave information, such as: period of use, type, alloy and number of reused parts was collected for the period of this analysis. The first source for this information was reports provided by HVC maintenance department (Figure 5-9 shows an example of these reports). In addition, information available from the Technical Development Department was cross-referenced (Figure 5-10 shows one example of these reports). Moreover, direct consultation with HVC personnel was initiated when any inconsistencies or lack of information occurred.

<table>
<thead>
<tr>
<th>#5 CRUSHER MANTLES</th>
</tr>
</thead>
<tbody>
<tr>
<td>Date Installed</td>
</tr>
<tr>
<td>08/03/01</td>
</tr>
<tr>
<td>09/01/01</td>
</tr>
<tr>
<td>11/23/01</td>
</tr>
<tr>
<td>12/21/01</td>
</tr>
<tr>
<td>01/04/02</td>
</tr>
<tr>
<td>02/01/02</td>
</tr>
<tr>
<td>03/15/02</td>
</tr>
<tr>
<td>03/21/02</td>
</tr>
<tr>
<td>03/29/02</td>
</tr>
<tr>
<td>04/12/02</td>
</tr>
<tr>
<td>04/27/02</td>
</tr>
<tr>
<td>05/10/02</td>
</tr>
</tbody>
</table>

Please Note: The mill Tonnages may not reflect an accurate figure – may be missing some Met. Tonnage data.

Figure 5-9 Partial example of crusher-mantles report
Combining all the information available comprehensive tables were prepared for each crusher. The results for Crushers 4 and 5 are shown in Table 5-4 and in Table 5-5, respectively.
### Table 5-4 Crusher 4 liners detailed information

<table>
<thead>
<tr>
<th>Mantle Code</th>
<th>Install. Date</th>
<th>Removal Date</th>
<th>Hours in use</th>
<th>Cum. Tonnage</th>
<th>Vendor</th>
<th>Type</th>
<th>Material</th>
</tr>
</thead>
<tbody>
<tr>
<td>M302u</td>
<td>6/29/01</td>
<td>7/26/01</td>
<td>610</td>
<td>1,598,086</td>
<td>F.S.</td>
<td>2-pce Std 87.56&quot;</td>
<td>Mang.</td>
</tr>
<tr>
<td>M503u</td>
<td>7/26/01</td>
<td>9/7/01</td>
<td>965</td>
<td>2,562,607</td>
<td>Esco</td>
<td>3-pce Std 87.25&quot;</td>
<td>CZ 18</td>
</tr>
<tr>
<td>M104n</td>
<td>9/8/01</td>
<td>10/18/01</td>
<td>844</td>
<td>1,978,516</td>
<td>Esco</td>
<td>3-pce Std 87.25&quot;</td>
<td>CZ 18</td>
</tr>
<tr>
<td>M505n</td>
<td>10/18/01</td>
<td>11/15/01</td>
<td>618</td>
<td>1,317,435</td>
<td>Esco</td>
<td>3-pce Std 87.25&quot;</td>
<td>CZ 18</td>
</tr>
<tr>
<td>M106u</td>
<td>11/17/01</td>
<td>11/29/01</td>
<td>262</td>
<td>568,610</td>
<td>Col.</td>
<td>2-pce Ribbed Std</td>
<td>Mang.</td>
</tr>
<tr>
<td>M507n</td>
<td>11/29/01</td>
<td>12/27/01</td>
<td>614</td>
<td>1,115,941</td>
<td>Esco</td>
<td>2-pce Std 88&quot;</td>
<td>Mang.</td>
</tr>
<tr>
<td>M408n</td>
<td>12/28/01</td>
<td>2/7/02</td>
<td>907</td>
<td>2,027,699</td>
<td>Esco</td>
<td>2-pce O/S 90&quot;</td>
<td>Mang.</td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td></td>
<td>4820</td>
<td>11,168,894</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>M509u</td>
<td>2/9/02</td>
<td>2/21/02</td>
<td>257</td>
<td>517,944</td>
<td>Esco</td>
<td>3-pce Std 87.25&quot;</td>
<td>CZ 18</td>
</tr>
<tr>
<td>M410n</td>
<td>2/22/02</td>
<td>5/2/02</td>
<td>1573</td>
<td>3,531,753</td>
<td>Esco</td>
<td>3-pce Std 87.25&quot;</td>
<td>CZ 18</td>
</tr>
<tr>
<td>M311n</td>
<td>5/2/02</td>
<td>6/21/02</td>
<td>1144</td>
<td>2,733,731</td>
<td>Esco</td>
<td>3-pce Std 87.25&quot;</td>
<td>CZ 18</td>
</tr>
<tr>
<td>M112n</td>
<td>6/24/02</td>
<td>7/25/02</td>
<td>714</td>
<td>1,349,298</td>
<td>Esco</td>
<td>2-pce Std 88&quot; T. II</td>
<td>Mang.</td>
</tr>
<tr>
<td>M213n</td>
<td>7/27/02</td>
<td>9/3/02</td>
<td>855</td>
<td>1,797,474</td>
<td>Esco</td>
<td>2-pce O/S 90&quot; T. II</td>
<td>Mang.</td>
</tr>
<tr>
<td>M414n</td>
<td>9/4/02</td>
<td>9/25/02</td>
<td>408</td>
<td>862,988</td>
<td>Esco</td>
<td>2-pce O/S 90&quot; T. II</td>
<td>Mang.</td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td></td>
<td>4951</td>
<td>10,793,188</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

### Concaves

<table>
<thead>
<tr>
<th>Code</th>
<th>Install. Date</th>
<th>Removal Date</th>
<th>Hours in use</th>
<th>Cum. Tonnage</th>
<th>Vendor</th>
<th>Type</th>
<th>Material</th>
</tr>
</thead>
<tbody>
<tr>
<td>C002</td>
<td>6/29/01</td>
<td>2/7/02</td>
<td>4820</td>
<td>11,168,894</td>
<td>Penticton</td>
<td>Standard</td>
<td>W.I.</td>
</tr>
<tr>
<td>C004</td>
<td>2/9/02</td>
<td>9/25/02</td>
<td>4951</td>
<td>10,793,188</td>
<td>Penticton</td>
<td>Standard</td>
<td>W.I.</td>
</tr>
</tbody>
</table>
# Table 5-5 Crusher 5 Liners Detailed Information

<table>
<thead>
<tr>
<th>Mantle Code</th>
<th>Install. Date</th>
<th>Removal Date</th>
<th>Hours in use</th>
<th>Cum. Tonnage</th>
<th>Vendor</th>
<th>Type</th>
<th>Material</th>
</tr>
</thead>
<tbody>
<tr>
<td>M281n</td>
<td>1/5/01</td>
<td>2/24/01</td>
<td>948</td>
<td>2,222,258</td>
<td>Esco</td>
<td>2-pce U/S 84&quot;</td>
<td>Mang.</td>
</tr>
<tr>
<td>M531u</td>
<td>2/25/01</td>
<td>3/1/01</td>
<td>92</td>
<td>188,265</td>
<td>Esco</td>
<td>2-pce Std 88.73&quot;</td>
<td>Mang.</td>
</tr>
<tr>
<td>M182n</td>
<td>3/2/01</td>
<td>5/25/01</td>
<td>1,715</td>
<td>2,970,699</td>
<td>Esco</td>
<td>3-pce Std 87.25&quot;</td>
<td>CZ 18</td>
</tr>
<tr>
<td>M283n</td>
<td>5/26/01</td>
<td>6/21/01</td>
<td>566</td>
<td>1,071,485</td>
<td>Col.</td>
<td>2-pce Ribbed Std</td>
<td>Mang.</td>
</tr>
<tr>
<td>M551n</td>
<td>6/22/01</td>
<td>7/5/01</td>
<td>315</td>
<td>686,248</td>
<td>Esco</td>
<td>3-pce Std 87.25&quot;</td>
<td>CZ 18</td>
</tr>
<tr>
<td>M101u</td>
<td>7/6/01</td>
<td>8/2/01</td>
<td>621</td>
<td>1,443,101</td>
<td>Esco</td>
<td>2-pce O/S 90&quot;</td>
<td>CZ 18</td>
</tr>
<tr>
<td>M351n</td>
<td>8/3/01</td>
<td>8/30/01</td>
<td>614</td>
<td>1,339,742</td>
<td>Esco</td>
<td>2-pce O/S 90&quot;</td>
<td>CZ 18</td>
</tr>
<tr>
<td>M452n</td>
<td>9/1/01</td>
<td>9/27/01</td>
<td>549</td>
<td>1,312,926</td>
<td>F.S.</td>
<td>2-pce O/S 90.6&quot;</td>
<td>Mang.</td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td></td>
<td>5,420</td>
<td>11,234,723</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Mantles</th>
</tr>
</thead>
</table>

<table>
<thead>
<tr>
<th>Mantle Code</th>
<th>Install. Date</th>
<th>Removal Date</th>
<th>Hours in use</th>
<th>Cum. Tonnage</th>
<th>Vendor</th>
<th>Type</th>
<th>Material</th>
</tr>
</thead>
<tbody>
<tr>
<td>M452u</td>
<td>9/30/01</td>
<td>11/22/01</td>
<td>1195</td>
<td>2,501,162</td>
<td>F.S.</td>
<td>2-pce O/S 90.6&quot;</td>
<td>Mang.</td>
</tr>
<tr>
<td>M253u</td>
<td>11/23/01</td>
<td>12/19/01</td>
<td>605</td>
<td>1,257,902</td>
<td>Esco</td>
<td>3-pce Std 87.25&quot;</td>
<td>CZ 18</td>
</tr>
<tr>
<td>M106u</td>
<td>12/19/01</td>
<td>1/3/02</td>
<td>346</td>
<td>955,560</td>
<td>Col.</td>
<td>2-pce Ribbed Std</td>
<td>Mang.</td>
</tr>
<tr>
<td>M507u</td>
<td>1/3/02</td>
<td>1/31/02</td>
<td>619</td>
<td>1,443,466</td>
<td>Esco</td>
<td>2-pce Std 88&quot;</td>
<td>Mang.</td>
</tr>
<tr>
<td>M254n</td>
<td>2/1/02</td>
<td>3/14/02</td>
<td>922</td>
<td>2,335,759</td>
<td>Esco</td>
<td>2-pce O/S 90&quot;</td>
<td>CZ 18</td>
</tr>
<tr>
<td>M155n</td>
<td>3/15/02</td>
<td>3/20/02</td>
<td>34</td>
<td>22,911</td>
<td>Esco</td>
<td>2-pce O/S 90&quot;</td>
<td>CZ 18</td>
</tr>
<tr>
<td>M254u</td>
<td>3/20/02</td>
<td>3/28/02</td>
<td>196</td>
<td>423,797</td>
<td>Esco</td>
<td>2-pce O/S 90&quot;</td>
<td>CZ 18</td>
</tr>
<tr>
<td>M556n</td>
<td>3/29/02</td>
<td>4/11/02</td>
<td>309</td>
<td>777,223</td>
<td>Esco</td>
<td>3-pce Std 87.25&quot;</td>
<td>CZ 18</td>
</tr>
<tr>
<td>M155u</td>
<td>4/11/02</td>
<td>4/26/02</td>
<td>337</td>
<td>640,472</td>
<td>Esco</td>
<td>2-pce O/S 90&quot;</td>
<td>CZ 18</td>
</tr>
<tr>
<td>M158n</td>
<td>5/10/02</td>
<td>6/7/02</td>
<td>618</td>
<td>1,218,575</td>
<td>Esco</td>
<td>2-pce O/S T. II 90&quot;</td>
<td>Mang.</td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td></td>
<td>5,463</td>
<td>12,172,495</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Concaves</th>
</tr>
</thead>
</table>

<table>
<thead>
<tr>
<th>Mantle Code</th>
<th>Install. Date</th>
<th>Removal Date</th>
<th>Hours in use</th>
<th>Cum. Tonnage</th>
<th>Vendor</th>
<th>Type</th>
<th>Material</th>
</tr>
</thead>
<tbody>
<tr>
<td>C003</td>
<td>1/5/01</td>
<td>9/27/01</td>
<td>5420</td>
<td>11,234,723</td>
<td>Penticton</td>
<td>Standard</td>
<td>W.I.</td>
</tr>
<tr>
<td>C005</td>
<td>9/30/01</td>
<td>6/7/02</td>
<td>5463</td>
<td>12,172,495</td>
<td>Penticton</td>
<td>Standard</td>
<td>W.I.</td>
</tr>
</tbody>
</table>
As can be seen in the tables only two materials have been used for the mantles, austenitic manganese steel ("manganese") and martensitic chrome moly steel ("CZ 18 alloy").

In addition to the liner information previously mentioned, dimensional details about the original liner profiles and some parts of the crushers were collected. For all types of liners available, drawings containing their profile were supplied by the vendors. Although original dimensions and part details were not supplied by the manufacturer of the crusher (they refused to provide it), historical measurements taken by the maintenance department were used to generate a section-view drawing of the crusher. More details about this drawing and its use in the wear determination procedure will be covered in the next section.

5.2.3 Chamber profile data

Chamber profile data collection was initiated on June 14 2001 using the LPD (described in section 5.1). The original plan was to obtain one set of measurements for each crusher every other week, following a predetermined positioning arrangement. For every other measurement, the LPD was positioned at the 4 o’clock and 10 o’clock regions (relative to the control cabin), Figure 5-11 shows a positioning diagram. The two locations used for measurements were chosen based on practical observations which suggested that these locations present different concave profile wear, i.e. distinct high-localized wear regions.
Due to changes in the crusher maintenance shutdown schedule by the mine, the data collection period increased from every second week to every third week. However, due to operational issues, situations arose where measurements could not be taken. This resulted in an actual frequency of measurements per crusher of three measurements every two months on average.
5.3 Data Analysis

5.3.1 Crusher Operational Data

For each crusher the complete set of data from Citect along with basic liner information and measurement dates were all grouped into a single spreadsheet. Figure 5-12 shows examples of records as they are listed in this comprehensive data file.

Figure 5-12 Example of the complete data file of Crusher #5 (records 5075 to 5714 are hidden to facilitate visualization).

<table>
<thead>
<tr>
<th>Record</th>
<th>Date &amp; Time</th>
<th>Liner Info</th>
<th>Throughput</th>
<th>Max. Conc.</th>
<th>Motor Current</th>
<th>Percent Level</th>
<th>Feed Size</th>
<th>Max.</th>
<th>Product Size</th>
</tr>
</thead>
<tbody>
<tr>
<td>5066</td>
<td>6/30/01 00:00:30</td>
<td>0.0 0.0 0.0 0.0 0.0 0.0 0.0 0.0 0.0</td>
<td>9582.069</td>
<td>57.9 57.9 57.9 57.9</td>
<td>57.9 57.9 57.9 57.9</td>
<td>21.0 21.0 21.0 21.0</td>
<td>38.8 38.8 38.8 38.8</td>
<td>32.3 32.3 32.3 32.3</td>
<td></td>
</tr>
<tr>
<td>5067</td>
<td>6/30/01 00:00:30</td>
<td>0.0 0.0 0.0 0.0 0.0 0.0 0.0 0.0 0.0</td>
<td>9582.069</td>
<td>57.9 57.9 57.9 57.9</td>
<td>57.9 57.9 57.9 57.9</td>
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In the table, each row contains a complete set of information for each hour of operation as extracted from the Citect system. In addition, relevant data obtained from other sources
are as follows: liner installation/removal dates, liner identifications and measurement dates.

The throughput column contains average-hourly values of throughput in metric tonnes per hour that were calculated and recorded by the Citect system from values gathered by a weightometer installed at the discharge conveyor belt. This average value represents the total crusher throughput for the one hour period and is used in the calculation of the mantle and the concave cumulative tonnage.

The current information available in the Citect system is used as an indirect measure of the power draw assuming the voltage to be constant. Both the average per hour and maximum value per hour are listed in the table.

The remaining columns list the bowl level, the size distribution of the feed as percentages of course, fine and medium categories, the mantle vertical position as well as the size distribution of the product in a similar manner as described for the feed size.

A graphical representation of the production data was chosen to investigate correlations between the variables. The period for this analysis was the total time comprising two concave lives for each crusher. Thus, four comprehensive graphs have been used to present this information (the four graphs are given in section 6.3).
Ideally, the analyses should include all the parameters listed in the complete spreadsheet, however some of them have been omitted. Feed size distribution was left out of the graph due to knowledge of high likelihood of distortions in its values. Experience shows that the WipFrag system used for feed size distribution classification may give erroneous information during the day because of interference from sun light in the image acquisition system (Reddick, 2002).

Bowl level was also excluded since its values vary significantly within a one-hour period and the average value does not represent a meaningful parameter. In addition, it was assumed that because of the automatic control employed on the crusher, this variable has already been optimized, i.e. dump level permitting the feed rate is adjusted in order to maintain the crusher as choked as possible.

In the graphical analyses, production rate is investigated by plotting its maximum value per hour only. The rationale behind this decision is that the average value per hour does not properly indicate if the machine is operating “loose”, i.e. high throughput has been the target instead of adjusting CSS below the maximum limit.

Of primary interest in the analyses is the determination of extreme conditions of product quality such as high quality and poor quality production occasions. Crushing product quality is usually related to the amount of course material and sometimes also related to the amount of fines therefore these two percentage values are incorporated in the analyses leaving the percentage of medium size out (Dance, 2001).
To better investigate using product quality, the fine and coarse size distributions are plotted underneath each other in the same graph. In addition, the Y-axis direction of coarse distribution is inverted so that when both the plot lines move upward or downward they indicate good and poor quality product trends respectively. An example is shown in Figure 5-13.

![Graph with product quality plots](image)

**Figure 5-13 - Example of a graph with product quality plots**

In order to accommodate all the different parameters plotted together in a comprehensive graph for long periods (approximately 6000 hours per graph) SigmaPlot technical graphing software from SSPS® Inc., US was used. Each parameter is plotted using an individual Y-axis and a single-common X-axis corresponding to the time.
Figure 5-14 shows part of the graph for crusher #5-concave C005 and serves to exemplify how liner information is included in the graphs. Each period corresponding to one mantle life is represented as rectangle on the graph and contains its sequential number at the top. The cumulative tonnage achieved by each mantle is plotted on the bottom of the graph. The measurement dates correspond to long vertical lines.
Figure 5-14 – Selected liner information for Crusher 5 from October 2001 to January 2002

The vertical mantle position, or shaft position, can vary between the 0 (bottommost) and 10 inches (topmost). An example plot of the historical mantle position data is shown in Figure 5-15 (dotted line). In order to mitigate the effect of short term upsets on the
analyses (i.e. mantle position variation during idle operation) a weekly average value plot was produced using the same Y-axis and the result can be seen in Figure 5-15 (solid line).

The average values are calculated per each week by weighting the mantle position values to the hourly production for each mantle. The formula is given below:

\[
\bar{M}_w = \frac{\sum_{h'=1}^{h'=168} P_{h'} \times M_{h'}}{\sum_{h'=1}^{h'=168} P_{h'}}
\]

where \( \bar{M}_w \) is the weekly average mantle position, \( P_{h'} \) is the hourly average production rate, \( M_{h'} \) is the hourly average mantle position, and, \( h' \) is equal to 1 for the first hour with \( P_{h'} > 0 \) after mantle installation or start of a new week.
Weekly weighted averages are similarly produced for some of the other parameters. In addition, the calculation of the weighted average product quality value is enhanced with Excel conditional functions to cancel the average calculation during periods of high percentage of noisy data within the week period. WipFrag noisy data can be identified from situations containing several consecutive repeated values during normal crushing (i.e. consecutive meaningful positive hourly average production rates, greater than 1500 tph). Thus, periods containing problems with the data gathering system are shown in the graph as a missing part of the average line, as can be seen in three different week periods on the graph for Crusher 4 graph, shown in Figure 5-16. This was done to avoid misinterpretations during the analyses.

Figure 5-16 – Example of the representation of data problems in product quality (circles indicate areas of noisy data where the weighted average of product quality was not calculated).
For current draw (amps) not only the hourly and weekly averages are plotted but also the maximum values per hour are included in the graph. As shown in Figure 5-17, average amps are plotted in a line format and maximum amps are plotted in a scatter format. The maximum amps scatter plot indicates the occurrence of high amplitude "spikes" which may confirm overload conditions at the machine if the average value is also high.

Occasionally a bias in the motor current reading is observed. This is generally due to a pending motor failure. This phenomenon was recorded for two occasions within the total period analysed, one occasion for Crusher 4 (10/7/2002-1/25/2002) and the other for
Crusher 5 (1/5/2001-4/13/2001). When this occurred, the raw current draw values were adjusted to be in accordance to the rest of the period facilitating visual analyses.

In order to adjust the values, a comparison between values observed for idle operation of the crusher is performed. The comparison is made between a known period containing normal motor operating characteristics and a period when the motor shows distortions. A calculation of the difference between the average idle values for these two periods is performed and the result is used to adjust the values to be plotted. The value used for the adjustment in the graph for Crusher 4 was -30 amps and for Crusher 5 the adjustment was -60 amps for the mentioned periods. These occurrences are shown on the graphs in lighter tones and one example is given in Figure 5-18.

![Figure 5-18 - Current draw adjusted plot](image)

Once the complete graph containing all the parameters for the period of a concave life is plotted, a broad picture of the crushing characteristics over time is achieved. Thus,
periods containing significant results may be easily identified. The periods of main interest in this analysis are the ones showing overload conditions at the machine, identified by the current draw plots or conditions showing high quality product and mantle’s high cumulative tonnage together.

Moreover, after identifying these periods, the analysis may be enhanced by gathering information from measurements performed within the chosen periods. As the dates for the measurements are also included in the graphs they are easily identified. Cross-referencing the two sets of information serves to provide an understanding of the influence of the crusher chamber profile on operational parameters and vice versa.

The graphs are also useful when the analysis is performed in the opposite direction, that is, when interesting results obtained from wear measurements (for example when atypical wear of the mantle is observed) determine regions of interest on the graphs.

5.3.2 Chamber Profile Data

As mentioned in section 5.1, issues related to the prototype LPD were previously addressed. Of major concern were apparent problems in the measurement device, in both the hardware and the software components of the LPD which were affecting the accuracy of the results. Thus, an investigation was conducted by performing several laboratory tests with the equipment as well as carrying out field checks in order to compare LPD results and actual dimensions of the crusher.
The investigation resulted in the detection of some key issues that were responsible for a degradation in the accuracy of the results, as listed below:

- a misalignment between the mirror base and mirror vernier causing an error in the calculation of the inclination angle of the track;
- the occasional appearance of noisy data in measurement results;
- a bug in the software resulting in a systematic increase of the length of the profile drawings;
- inconsistencies between the positioning of the profile drawings (mantle and concave profiles) and the actual original positioning of the liners in relation to each other and to the other parts of the crusher; and,
- systematic deviations between distances measured using the laser and real distances, as shown in the graph in Figure 5-19.
Figure 5-19 Deviations in laser measurements

More details concerning the investigations of the LPD accuracy as well as information regarding the tests conducted can be found in reports by the author (Rosario, 2001), (Rosario, 2002a) and (Rosario, 2002b).

Following this investigation, it was decided to write a new program to replace the major functions of the original software provided with the equipment. This new program utilizes the raw data generated by the equipment, i.e. the resulting spreadsheets from the two calibration tests, the mantle test, and the concave test. This new program was also used to correct past measurements. A description of program functions as well as the rationale applied in the calculations is available in Appendix B.
The new program satisfies several objectives such as improving accuracy and providing more comprehensive results. The new program provides several new features, such as the calculation of CSS by mantle position (more details about the enhancement of the measurement procedure is given in section 6.1).

In addition, as this program incorporates the data from the original liners (drawing information previously transformed in a numeric format), wear calculations such as wear areas and wear rates for selected regions are made available. Furthermore, a tool to simulate the replacement of different liner profiles is also available (more details about simulations capabilities of the program are given in Appendix B).

As discussed in the previous section, the achievement of meaningful, or “standardized”, information from the measurements was of great interest in this analysis to complement the graphical analyses of crushing operational parameters. This new program utilises a rationale of “dividing” the crushing chamber in small slices to perform calculations resulting in new and enhanced information. This enhanced set of results facilitates the correlation between the measurements and operational data.

The slicing technique that is applied in the program is shown in Figure 5-20. In this figure, two potential crushing chambers are given to illustrate a possible difference in the bottom region of the crusher.
Figure 5-20 - Section view of two chambers and their slices
Although the entire profile is obtained by the measurement process\textsuperscript{4}, only the bottom region of the concave (corresponding to the two bottommost concave rows) is analysed by dividing this region into eighty 25.4 mm-high slices (1.0 in.). The slices are used in the calculation of:

- Chamber volumes by slices (in litres):
  a. with the mantle position (MP) equals to 0 mm (at the bottom); and,
  b. with MP equals to the average position for the week (AMPW).

- Radial distances between mantle and concave per slice (in millimetres):
  a. with MP equals to 0 and the mantle at 0 degrees of throw (in the middle);
  b. with MP equals to 0 and the mantle at its maximum displacement of throw;
  c. with MP equals to 0 and the mantle at its minimum displacement of throw.
  d. with MP equals to AMPW and the mantle at 0 degrees of throw (in the middle);
  e. with MP equals to AMPW and the mantle at its maximum displacement of throw; and,
  f. with MP equals to AMPW and the mantle at its minimum displacement of throw.

- Minimum distance among the 80 slices for result “e”, i.e. CSS.

\textsuperscript{4} Due to the lack of reference targets inside the crusher chamber, the top region of the profiles (where minimum wear occurs) is used to orientate the positioning of the measured profiles in reference to previous measurements and to the original drawings of the parts.
Minimum distance among the 80 slices for result "f", i.e. opened side setting (OSS).

(All the results above are also generated using the profile of a new mantle).

- CSS results corresponding to the entire range of MPs, from 0 to 254 mm (10 in.) by 12.7 mm (0.5 in.) steps.
- Concave-wear by slice, i.e. the radial difference in mm between the new profile and the measured profile of the concave.
- Mantle radial-wear by slice (mm), i.e. the radial difference in mm between the new profile and the measured profile of the mantle.
- Concave-wear rate by slice (mm per million tonnes of throughput).
- Mantle-wear rate by slice (mm per million tonnes of throughput).

In addition, the program generates three graphs, listed as follows:

- chamber volume by slices,
- CSS results versus MP ranging from 0 to 254 mm (10 in.), and
- liner-wear rate by slice.

Among the information available from the measurements and that provided by the new program, the first two graphs are of great value in complementing the graphical analyses of crushing operation parameters. These two graphs summarize the chamber profile information and describe the impact of MP adjustment on crusher chamber dimensions and the choking condition of the chamber. More detail about these graphical results is given next.
"Choking" condition of the chamber

Several authors correlate crushing performance to chamber profile and more specifically to the choking condition of the chamber (more details in section 3.1.4). In order to assess this relationship, the variation of chamber volume with height was included in the program as a graphical result. Similar to the approach discussed in the Literature Review, this type of graph helps in checking the chamber choking condition and in determining a choking point or region.

Figure 5-21 gives an example of the graph generated by the program with the information for two hypothetical chambers plotted on it. This graph gives a visualization of the main geometric characteristics of the chambers. The characteristics of the chambers contained in this example and the differences between them can be extracted from the graph and are listed below:

- in chamber "A" the slice-volumes decrease following two different patterns (a linear and rapidly decrease rate from slice # 80 to slice # 35 and a slowly decrease rate from slice # 35 to slice #13);
- in chamber "B" there is a single linear pattern for slice-volume decreasing from slice #80 to slice #13;
- chamber "A" has a defined choke region at slice #35;
- in chamber "B" the choke point is at the very bottom of the concave;
- in chamber "A" the volume drops from 40 to 28 litres from slice #35 to slice #13;
- in chamber “B” the volume drops from 64 to 28 litres from slice #35 to slice #13;
- chamber “A” is a non-choking chamber; and,
- chamber “B” is a “straight” chamber.

![Figure 5-21 Chamber volume by height](image)

**CSS information of the chamber**

The CSS and OSS data from a given chamber are calculated by the program based on the measurements. To calculate these numbers the program requires the input of a mantle position value (MP). Thus, to determine CSS and OSS at the “moment” of the measurement the weekly averaged MP for the date of the measurement is input. In
addition, a ratio of the CSS and OSS values for each wear measurement is calculated and used in the analyses.

The graph available from the program describes how the CSS changes with vertical position of the mantle. On this graph, the relationship between CSS and a series of simulated mantle positions is given (from 0 to 254 mm (10 in.) with 12.7 mm (0.5 in.) increments). To perform the calculations the program uses the measured profiles for the mantle and the concave. Figure 5-22 shows an example of this type of graph containing the information for two hypothetical chambers (C and D).

![Figure 5-22 - CSS versus vertical mantle position](image)
As shown in Figure 5-22, chamber information may be compared by their range of CSS and the relationship between CSS and mantle position. From this graph it is possible to see that:

- chamber “C” initial CSS is 140 mm (5.5 in.), result given MP equal to zero;
- chamber “C” CSS-range is within 83 and 140 mm (3.25 and 5.5 in.);
- chamber “C” CSS and MP relationship is linear;
- chamber “D” initial CSS is 203 mm (8.0 in.);
- chamber “D” CSS-range is within 127 and 203 mm (5.0 and 8.0 in.),
- chamber “D” CSS and MP relationship is non-linear;
6. Results and Discussion

6.1 Improvement in Measurement Process

**Measurement Time**

As mentioned in section 5.1, the redesigned support structure of the LPD resulted in a significant reduction in set up time. This gain combined with other developments, such as the reorganizations of the electrical boxes and cables as well as improved efficiency developed through experience by the maintenance personnel, reduced the original measurement time approximately 45% (from app. 2 h and 35 min to app. 1 h and 25 min). Figure 6-1 compares the measurement time for eight tests; four before the use of the new support and four after.

![Figure 6-1 Time spent in measurement tests (tests 5 to 8 were performed using the new support).](image-url)
Measurement Results

With the new program and the measurement raw data collected during the period of the analysis, 40 complete sets of measurement results were produced. Table 6-1 and Table 6-2 give a summary of the measurements for Crusher 4 and Crusher 5, respectively. In addition, these tables show other information about the crusher at the dates of attempted measurements: the status of the measurement attempt, the average mantle position during the week of the measurement, the cumulative tonnage of the concaves and mantles as well as the mantle code; indicating which specific mantle was in place at the time of the measurement.
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For each measurement used in the analysis, the complete set of results provided by the program were grouped in an Excel file (details in section 5.3.2) and the generated mantle and concave profile drawings were plotted together in an AutoCAD drawing containing their original profiles. These drawings were used for a visual evaluation of the wear (Appendix C shows an example of one measurement drawing result).
In addition to these drawings, three graphs were generated for each measurement. The first graph gives the chamber volumes by slices for two conditions: the chamber formed with mantle and concave as measured and the chamber formed if a new mantle was installed (Figure 6-2 shows the result for measurement 7). The second graph plots the relationship between CSS and a series of simulated mantle positions (Figure 6-3 shows the result for measurement 7). The third graph plots mantle wear rate versus slice numbers and concave wear rate versus slice numbers together (Figure 6-4 shows the result for measurement 7).

Figure 6-2 Chamber Volumes graph (using MP equals to AMPW) for measurement 7.
Figure 6-3 CSS versus MP graph for measurement 7.

Figure 6-4 Concave and mantle wear rate by slices for measurement 7.
6.2 Wear Determination

**Concave-wear**

As described in section 5.3.2, the new program enabled the inclusion of new features related to wear. Moreover, the analysis of the concave-wear rate by slices resulted in the determination of an average wear rate per slice for the bottom of the concave for the two different measured regions, i.e. the wear occurring at the 4 o’clock and the 10 o’clock position. Figure 6-5 shows the average concave-wear for the bottom of the concave.

To achieve the results plotted in Figure 6-5, some measurement information was excluded in the calculation due to issues with the data collected. Measurement results containing noisy data and measurement results for short concave-lives were excluded to enable a more accurate wear rate result. This was necessary due to the fact that the measurement accuracy of +/- 3 mm limits the wear rate calculation accuracy for the initial period of concave life. For example, consider the two cases that follow:

- in measurement 3 in the early concave life (737,689 of concave tonnage), the radii difference in slice 40 of 5.85 mm +/- 3mm results in a wear rate between 3.9 and 11.9 mm/Mton (approximate accuracy +/- 4)
- in measurement 62 in the end of concave life (9,412,297 of concave tonnage), the radii difference in slice 40 of 39.68 mm +/- 3mm results in a wear rate between 3.9 and 4.5 mm/Mton (approximate accuracy +/- 0.3).
The criteria used to reject measurements for the wear rate calculation was to not use measurements that give accuracy ranges greater than 0.75 mm/Mton.

Figure 6-5 Average concave-wear rate
Following the determination of the average wear-rates, it was possible to estimate the amount of wear occurring at the bottom of the concave for any given tonnage. Figure 6-6 gives the result for a simulation using 8 mega-tonnes of throughput for the two measured spots.

This result correlates well with practical observations at HVC. Commonly, the highest wear rate is observed close to the seam between the two bottommost rows of the concave parts at the 4 o'clock position. In addition, on several occasions when concaves were
used for long periods, this highly worn region resulted in broken edges on some concave parts.

**Mantle-wear**

Although drawing results served to better visualize the effects of mantle-wear on chamber profile, a similar wear-rate average calculation for the mantles did not provide a meaningful result. In contrast to the concave results, the wear-rate of any specific slice on the mantle shows greater variation for different measurements. This variation may be explained by the following facts:

- different mantle profile types were used during the concave life;
- two different mantle materials were used during the analysis period;
- the mantle movement in relation to the concave position;
- the impact of crushing operational conditions on mantle-wear is greater due to a lower wear resistance of the mantle when compared to the concave.
6.3 Correlation of Operational Data and Liner Characteristics

As discussed in section 5.3.1, four graphs were used to analyse crushing parameters for both crusher #4 and #5. Each of these graphs corresponds to a period of one concave life and are shown in Figure 6-7 to Figure 6-10, being:

- Figure 6-7 Graph #1 – Concave C002 at crusher #4 (29 June 2001-7 Feb. 2002);
- Figure 6-8 Graph #2 – Concave C004 at crusher #4 (9 Feb. 2002-25 Sep, 2002);
- Figure 6-9 Graph #3 – Concave C003 at crusher #5 (5 Jan. 2001-27 Sep, 2001);
- Figure 6-10 Graph #4 – Concave C005 at crusher #5 (30 Sep. 2001-7 June 2002).
Figure 6.7 Graph #1 - Concave C002 at Crusher #4 (29 June 2001-7 Feb. 2002)

[Cohesive measurements and cumulative temperature at the mantle are shown in the graph.]

CRUSHER #4 - CONCAVE C002 (6/29/01 - 2/17/02)
The graphical analysis allowed the observation of a series of interesting relationships between the parameters plotted and resulted in the identification of key periods for cross reference with the information provided by the wear measurements. The most relevant observations and significant relationships are discussed next.

6.3.1 Issues during the end of concave life

As shown in Figure 6-7 to Figure 6-10, on the majority of the occasions that overload conditions appeared, i.e. unstable current draw as well as high amplitude maximum current values (power spikes), the concaves had been running for more than 7 megatonnes on average. The only exception can be visualized in Figure 6-9 for concave C005 during the first 4 megatonnes of operation. However, in this instance the overload condition indicated by the current draw was related to problems with the motor, as explained in section 5.3.1.

Although several different types of mantles, containing different profiles, were used during the periods of concave life after 7 megatonnes, none of them achieved a substantial cumulative production. Actually, most of the lowest tonnage per mantle results appeared in these periods.

The characteristics related to these issues in the final period of concave life were investigated and the results are discussed next.
**Non-choking condition of the chambers**

Assessing the chamber volume characteristics using the graphical method described in section 5.3.2 provided a comparison between the chamber geometry characteristics achieved during different periods. These periods are: the periods when the issues occurred ("bad operation") and other selected periods when not only the current draw remained stable but also product quality and mantle tonnage were the best ("good operation"). From the assessment, it was clear that the chambers that provided the best results were closer to a non-choking type and the ones providing bad results were closer to a straight type (as suggested in section 3.1.4).

Figure 6-11 shows a chamber volume graph with the results of three measurements corresponding to periods of "good operation". In addition, Figure 6-12 shows a chamber volume graph with the results of three measurements corresponding to periods of "bad operation" (more detail about the determination of the "non-choking" condition of chambers is given in section 5.3.2).
Figure 6-11 Example of 3 measured chambers that resulted in "good operation".

Figure 6-12 Example of 3 measured chambers that resulted in "bad operation".
Increased volume of the chambers

It is a common procedure in the mines to increase the size of the mantles to compensate for worn concaves. The impetus for this is to try to maintain the original CSS. This practice of trying to maintain the CSS dimension by the successive installation of larger diameter mantles affects chamber volumes. The volume modifications may also contribute to rising power requirements.

Figure 6-13 serves to illustrate this procedure. In the figure, two hypothetical chambers with the same CSS (152 mm - 6 in.) are shown in section view. Although the mantle on the right is 254 mm (10 in.) larger in radius (idealistic extreme case), both chambers show an equal chamber in cross-section. However, with the use of a three-dimensional representation of these chambers, as shown in Figure 6-14, the difference between the chamber volumes becomes apparent.

![Figure 6-13 Cross-section view of two similar but not identical chambers](image)
The example chambers in Figure 6-13 and Figure 6-14 do not exactly match the dimensions of the liners used at HVC. However, a rough estimation of two chamber volumes normally utilised at HVC providing similar CSS confirms the difference. One chamber consisting of: a new standard size mantle and a new set of concaves has a volume of approx. 19.57 m$^3$ (1,194,200 in.$^3$), and a chamber comprised of worn concaves and a new over size mantle has a volume of approx. 20.35 m$^3$ (1,242,100 in.$^3$); an increase of 4%.

Since the lifetime of a concave is much longer than that of a mantle, the practice of using several mantles with one concave is worthwhile. However, as the impact of the chamber volume change is often overlooked, situations can arise where uneconomical mantle concave combinations are used.
**Increased CSS area**

Similar to the increase in chamber volume, the discharge area increases by approximately 10% during the final stage of the concave life. Thus, although the CSS can be maintained at a desired value, the amount of material being discharged increases and the size distribution becomes courser as the area increases. Figure 6-15 shows the difference between the CSS area achieved with the same gap of 122 mm (4.8 in.) for two extremely different, but feasible, situations. The first area, A1, was calculated for a new concave and a new standard-size Esco 3-piece mantle, while the second area A2 was calculated for a concave as it was measured after approximately 87% of its useful life (measurement # 62 on 22 August 2002 at crusher # 4) and a new over-size 90” Esco-2 piece mantle.

Using automatic control on the crusher, product size distribution (percentage of product course) and crusher power (amps value) are part of the input parameters used to adjust the mantle position and the feed rate. Thus, an increased discharge area may be an additional factor in explaining the occurrences of current “spikes” and intermittent overload conditions of the machine during the final life of the concave. In other words, as the product becomes coarser, the control algorithm tends to raise the mantle closer to its maximum limit. Alternatively, as the current draw increases, the control tends to lower the mantle. These contrary trends may result in a greater instability when using the automatic control during the final life of the concave.
6.3.2 Mantle and crushing performance

The analysis of the mantle position adjustment (described in section 5.3.2) revealed the correlation between mantles which lasted for short periods of time and their limited room for adjustment, i.e. when a mantle needs to be set at a high position just after its installation, obviously its life will be short. An example of this problem occurred when the mantle M257n was installed in Crusher 5 with concave C005 and needed to be immediately set at the 203-mm (8-in.) position in order to achieve the desired CSS (refer
to the 10th mantle in Figure 6-10). Only 595,668 tonnes were produced with this mantle before it had to be replaced.

Similar cases occurred with all four concaves analysed and served to identify two facts. First, poor knowledge of concave wear at the time of selection and installation of the mantle generates short mantle lives. Second, although several different types of mantles were used (8 types) during the period of the analyses, frequently the mantles that were installed could not be used for their full range of mantle positioning, i.e. an initial adjustment close to 0-mm position and a final adjustment close to the 254-mm (10-in.) position. These short life mantle occurrences were mostly observed during two specific phases of the concave life: the initial life period of the concave (0 to app. 2 mega-tonnes) and the second half of its useful life (more than 5.5 mega-tonnes). Based on the fact that all the available under-size and over-sized profiles were used during these two specific periods of concave life, this observation suggests that those profiles are inadequate to provide long mantle life.

6.3.3 Liners that provided optimum performance

In general, product quality varied greatly during the period of the analysis. Only a few occasions were observed where reasonable product quality, smooth operation, and normal product rate occurred simultaneously for considerable time periods. These rare events happened for Crusher 4 concave C004 with mantles M410n and M311n (refer to the 2nd and 3rd mantle in Figure 6-8), and Crusher 5 concave C003 with mantle M182u (refer to
the 3rd mantle in Figure 6-9). Each of these mantles were 3-piece standard 2216 mm (87.25 in.) diameter Esco CZ 18 alloy, and the periods of these occurrences were approximately within 2.0 and 5.5 mega-tonnes of concave life.

In addition to performing well, the three mantles each achieved a considerable cumulative tonnage of app. 3.1 megatonnes on average. From this, two things were observed. First, the best matches between mantle and concave occurred between 2.0 and 5.5 mega-tonnes of concave life. Second, the Esco 3-piece standard type mantle has the most suitable profile among the mantles used during this period of concave life. These facts supported an investigation of the characteristics of these chambers and to designate them as "good chamber" characteristics to be targeted during the entire concave life. The results from this investigation are discussed in the following section.
6.4 Liner Management

An interesting result of this analysis was the observation of a large variation in the number of mantles utilized on each concave as well as the large variation in the tonnage achieved by these mantles as summarized in Figure 6-16.

![Mantle Production Cr 5 & Cr 4 graph]

Figure 6-16 Number of mantles used per concave and their total tonnage
These variations in mantle use plus the issues related to the final life period of the concaves, discussed in section 6.3.1, suggested that the liner replacement policies at HVC should be changed in order to improve crushing performance.

The first modification to the usual replacement schedule is the reduction of concave life to app. 7.5 megatonnes. Second, as it is apparent from the analyses, only one of the current mantle designs fulfills the requirements necessary for effective crushing. Therefore, the 3-pieces standard mantle design is the only design considered useable for a new liner replacement policy. In addition, two new mantle types, under-size and over size profiles, should be designed for use during start-up and final concave operational conditions.

In the design process, the non-choking characteristic of the chamber and the room for adjustment of the mantle position are two essential design characteristics that must be optimized. A “good” non-choking condition and a CSS-MP relationship that provides a long life for the mantle are the design targets.

Considering that under optimal circumstances mantle lives exceeded 3 million tonnes, and following maintenance department recommendations to maintain the replacement schedule following the current three-week pattern, the expected useful lives for the mantles are designed as follows:

- 1st mantle – 9 weeks (app. 2,900,000 tons)
- 2nd mantle – 9 weeks (app. 2,900,000 tons)
- 3rd mantle – 6 weeks (app. 1,950,000 tons)

Note: Only three mantles are used per concave and the maximum concave tonnage is close to the 7.5 megatonnes targeted.

In the design of the new over-size mantle profile, the concave wear rate by slice (described in section 6.2) has been used to estimate the wear of the concave after 5.8 megatonnes. A new concave profile was used in the design process of the new under-size mantle. After the design of the new profiles, their chamber conditions and CSS-MP relationships were assessed using the graphical analyses discussed in section 5.3.2.

Figure 6-17 compares the expected chamber condition of a new under-size mantle and the best option of mantle profile available (Esco 2 pieces 2184 mm – 86 in.). As well, the CSS-MP relationships for these two mantles are compared in Figure 6-18.
Figure 6-17 Comparison of chamber condition for under size mantles

Figure 6-18 Comparison of CSS-MP relationship for under size mantles
Figure 6-19 and Figure 6-20 show a comparison of the chamber conditions and CSS-MP relationship for two over-size mantles, the new design and the best option available (recent Esco 2 pieces 2286 mm – 90 in.).

Figure 6-19 Comparison of chamber condition for over size mantles.
The new designs follow the 3 pieces arrangement to facilitate the reuse of the top piece and the middle piece. Two new bottom pieces (one under-size and one over-size) and one new medium piece (over-size) were designed (drawings of the new parts are shown in Appendix D).

With the introduction of two new mantle designs and the reduction of the concave useful life, the use of three mantles per concave is recommended in a proposed schedule for liner replacement. Figure 6-21 shows how the pieces are used over the life of the concave.
The application of the new designed mantles and the suggested management policy for the replacement of the liners will not only result in better operational conditions and increased average product quality but also in reduced costs.

A cost analysis was performed to compare the current liner costs and the projected costs for the suggested procedure. First, based on the liner information (section 5.2.2) and the historical data for the last six concaves used in Crushers 4 and 5 (summarized in Table 6-3) the current average cost of liner replacement per tonne (in Canadian dollars) was calculated as shown in Table 6-4. In addition, the average downtime was calculated, based on the same period, as shown in Table 6-5.
Table 6-3 Summary of liners information for 6 recent concave life periods

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<td>9/25/02</td>
<td>5462</td>
<td>10,794,188</td>
<td>10,814,188</td>
<td>6</td>
</tr>
<tr>
<td>C007</td>
<td>6/8/02</td>
<td>1/22/03</td>
<td>5462</td>
<td>9,730,099</td>
<td>9,740,099</td>
<td>9</td>
</tr>
<tr>
<td>C006</td>
<td>9/26/02</td>
<td>3/26/03</td>
<td>4334</td>
<td>8,300,496</td>
<td>8,310,496</td>
<td>6</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td></td>
<td></td>
<td>32969</td>
<td>63,402,897</td>
<td>63,422,897</td>
<td>47</td>
</tr>
<tr>
<td><strong>Average</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>1,923</td>
</tr>
</tbody>
</table>

Table 6-4 Liner costs (parts, rebuilt and installation) and current total cost per ton

<table>
<thead>
<tr>
<th>Liner costs</th>
<th>Quant.</th>
<th>Unit Cost</th>
<th>Total Cost</th>
</tr>
</thead>
<tbody>
<tr>
<td>Used mantle</td>
<td>17</td>
<td>$1,850</td>
<td>$31,450</td>
</tr>
<tr>
<td>New mantle</td>
<td>30</td>
<td>$41,900</td>
<td>$1,257,000</td>
</tr>
<tr>
<td>Concave</td>
<td>6</td>
<td>$102,930</td>
<td>$617,580</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td></td>
<td></td>
<td>$1,906,030</td>
</tr>
<tr>
<td><strong>Total tonnage</strong></td>
<td></td>
<td></td>
<td>63,402,897</td>
</tr>
<tr>
<td><strong>Cost per ton</strong></td>
<td></td>
<td></td>
<td>$0.0301</td>
</tr>
</tbody>
</table>

5 Occasionally in the mine, a mantle that was used in one crusher is utilized in another. In such cases, the mantle-mainshaft assembly is just reinstalled. Thus, part costs and labour/supplies-assemblage costs are not incurred in these procedures.
Table 6-5 Specific downtime per liner and current total liner downtime

<table>
<thead>
<tr>
<th>Liner</th>
<th>Quant.</th>
<th>Unit</th>
<th>Downtime (h)</th>
<th>Total Downtime (h)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mantle</td>
<td>41</td>
<td>12</td>
<td>492</td>
<td></td>
</tr>
<tr>
<td>Mantle and concave</td>
<td>6</td>
<td>72</td>
<td>432</td>
<td></td>
</tr>
<tr>
<td>Total downtime</td>
<td></td>
<td></td>
<td>924</td>
<td></td>
</tr>
<tr>
<td>Total tonnage</td>
<td></td>
<td></td>
<td>63,402,897</td>
<td></td>
</tr>
<tr>
<td>Average downtime</td>
<td></td>
<td></td>
<td>14.57</td>
<td>(hour/Mton)</td>
</tr>
</tbody>
</table>

Second, assuming that the cost of the new designed mantle parts would be similar to the correspondent standard-size parts, and following the same rationale used for the calculation of the current cost and downtime, the projected cost and downtime were then calculated as shown in Table 6-6 and Table 6-7, respectively.

Table 6-6 Liner costs (parts, rebuilt and installation) for one concave life and projected cost per ton

<table>
<thead>
<tr>
<th>Liners details</th>
<th>Quant.</th>
<th>Unit Cost</th>
<th>Total Cost</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mantle</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Top piece liner</td>
<td>1</td>
<td>$7,650</td>
<td>$7,650</td>
</tr>
<tr>
<td>Med. piece liner</td>
<td>2</td>
<td>$10,700</td>
<td>$21,400</td>
</tr>
<tr>
<td>Bottom piece liner</td>
<td>3</td>
<td>$12,000</td>
<td>$36,000</td>
</tr>
<tr>
<td>Supplies</td>
<td>3</td>
<td>$4,000</td>
<td>$12,000</td>
</tr>
<tr>
<td>Labour</td>
<td>3</td>
<td>$5,500</td>
<td>$16,500</td>
</tr>
<tr>
<td>Installation</td>
<td>3</td>
<td>$1,850</td>
<td>$5,550</td>
</tr>
<tr>
<td>Mantles (total)</td>
<td></td>
<td></td>
<td>$99,100</td>
</tr>
<tr>
<td>Concave</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Concave (total)</td>
<td>1</td>
<td>$102,930</td>
<td>$102,930</td>
</tr>
<tr>
<td>Liners total cost</td>
<td></td>
<td></td>
<td>$202,030</td>
</tr>
<tr>
<td>Total tonnage (24 weeks)</td>
<td></td>
<td></td>
<td>7,754,100</td>
</tr>
<tr>
<td>Cost per ton</td>
<td></td>
<td></td>
<td>$0.0261</td>
</tr>
</tbody>
</table>
Table 6-7 Specific downtime per liner and projected total liner downtime

<table>
<thead>
<tr>
<th>Liner Replaced</th>
<th>Quant.</th>
<th>Unit</th>
<th>Total Downtime (h)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mantle</td>
<td>2</td>
<td>12</td>
<td>24</td>
</tr>
<tr>
<td>Mantle and concave</td>
<td>1</td>
<td>72</td>
<td>72</td>
</tr>
<tr>
<td><strong>Total downtime</strong></td>
<td></td>
<td></td>
<td><strong>96</strong></td>
</tr>
<tr>
<td><strong>Total tonnage</strong></td>
<td></td>
<td></td>
<td><strong>7,754,100</strong></td>
</tr>
<tr>
<td><strong>Average downtime (hour/Mton)</strong></td>
<td></td>
<td></td>
<td><strong>12.38</strong></td>
</tr>
</tbody>
</table>

A comparison between these two calculated costs indicates that the suggested policy enables a reduction of app. 13% (from 0.0301 to 0.0261 Canadian dollars) over the total annual liner cost (parts, rebuilds and installations) and the cost associated to the amount of downtime involved in the replacements is also expected to drop as the average annual downtime would decline by 15% (from 14.57 h/Mton to 12.38 h/Mton).
7. Conclusions

7.1 Achievements

This thesis has presented the development of a novel approach to assessing wear in gyratory crushers. Through the use of this approach greater understanding into the relationship between, crusher wear, crushing chamber geometry and production capacity and quality has been gained. In addition, the research has had tangible results of direct benefit to the supporting organization. More specific outcomes of the research are as follows:

- The use of a new laser profiler device (LPD) to measure the crusher chamber was implemented at HVC and improvements were made to the original measurement procedure. The redesign of LPD’s support structure resulted in a 45% reduction in the necessary time for measurements. The investigation of the initial issues presented in the application of the new device resulted in the correction of several problems and thus, in the improvement of the accuracy of the generated liner profiles.

- Chamber data provided from the measurements and crushing operational data from mine information systems were collected for the two crushers. The data comprised of 40 complete sets of measurements and information of approximately 23,300 hours of records for crushing operational parameters. Calculations and graphical analysis
techniques were developed and integrated in a new software tool to facilitate data analysis.

- The data analyses conducted provided an understanding of crushing chamber characteristics and their impact in crushing performance. In addition, wear rate as a function of production was determined for the concaves, which enabled wear prediction for the bottom part of the concave.

- The knowledge gained by the analyses helped in the evaluation of the current liner management policy. The evaluation resulted in a revised maintenance schedule based on the use of two new mantle profiles designed for this application. The proposed liner management policy is expected to reduce overall liner costs as well as to enhance crushing performance. Thus, a real opportunity to increase the profitability of the operations was gained with this work.
7.2 Future Work Opportunities

Continue with the laser measurement and the improvement of the process. The continued collection of additional measurement data can facilitate the assessment of mantle wear and enables the determination of mantle wear rates as functions of:

- chamber characteristic
- concave life period
- mantle position variation
- mantle material

Expand the work done by combining concave and mantle wear prediction and, therefore develop a CSS prediction tool to be used online with the operations.

Re-evaluate LPD performance and application in order to assess the necessity of building an alternative device based on laser technology or the application of a new technology such as sacrificial sensors as has been the focus of research for cone crushers.
8. **Reference List**


Appendix A  Drawings of the new support structure.
Appendix B  Description of the measurement program
To improve the accuracy of the measurement results obtained using the LPD and to enhance the scope of these results, such as with the addition of CSS and wear rate determination, a new program was developed using Microsoft Excel. The major functions and the calculation procedures applied in this program are described in a sequential format as follows.

**Step 1**

Four files generated from the measurement test are loaded in the program. Each file contains a table of coordinates generated by the LPD (an example is shown in Table 5-1). The files correspond to the following measurements:

1st - shooting the calibration bars with the laser beam perpendicular to the track;  
2nd - shooting the bars after levelling the mirror;  
3rd - shooting the concaves of the crusher; and,  
4th - shooting the mantle after rotating the mirror 90 degrees.

**Step 2**

The program identifies at which actuator position the calibration bar was first targeted during the first and the second calibration measurements. Then, using the trigonometric relations between these two values, the inclination angle of the track (α) is calculated (Figure 5-4 pg. 58 illustrates the process).
Step 3

The program corrects the data tables contained in the other two files (mantle and the concave measurements) utilizing a model developed from results of field-tests which provided evidence of systematic deviations between distances measured and real distances (the graph in Figure 5-19 pg. 84 shows the deviations).

Step 4

Each pair of coordinates, from both liners measurements, is transformed into a pair of coordinates in the Cartesian system with the origin (0,0) being the centre of the shaft located at the top part of the laser track ("track" coordinate system), as shown in Figure B-1. The program performs this calculation using the angle α and the trigonometric relationships between the original coordinates and the ones in the "track" coordinate system, as illustrated in Figure B-2.

![Figure B-1 Location of the origin of the "track" coordinate system](image-url)
Step 5

The two new sets of points, corresponding to the mantle and the concave profiles, are transferred and plotted into an AutoCAD drawing as two polylines. As shown in Figure B-3, the two profiles are plotted together with the drawings of the original liners and the crusher mainshaft. However, the position of the polylines does not match with the crusher drawing. This is due to the different coordinate system of the crusher drawing.
which has its origin coinciding with the pivot point of the crusher mainshaft ("crusher" coordinate system).

![Figure B-3 Snapshot of the AutoCAD drawing with the measured profiles and the original parts](image)

Note that the mainshaft/mantle is drawn in its central position, i.e. with the eccentric in the position where the CSS is equal to OSS.

**Step 6**

The two polylines (measured profiles) are moved together to a new position that better represents the chamber as measured. In order to achieve this objective, the top region of the liners (low wear region) is used as reference in this aligning process. Once the
profiles are relocated, the X and Y offset values from the "crusher" coordinate system's origin are determined. These numbers are input in the Excel program to translate all data to the "crusher" coordinate system. Due to the crusher throw, a rotation of the mantle profile in the X,Y plane may be required and this procedure is discussed in step 8.

**Step 7**

To model the original liner profiles in Excel, equations are fit to the drawing profile data. As illustrated in Figure B-4, five equations are usually necessary to describe an original mantle profile.

![Figure B-4 Group of lines and arcs that represents a mantle original profile](image)
Step 8

In order to determine the required angle of rotation for the mantle measured profile, two methods are used. The first method utilises Excel to calculate the angle that minimizes the distance between the measured and the original profiles at their topmost regions. In some cases this approach is unsuccessful, in which case the angle is determined in AutoCAD by manually rotating the measured profile.

Step 9

Using the information achieved in steps 7 and 8 and trigonometric relationships, the program calculates new values that describe the measured mantle profile rotated by its maximum and minimum angular displacements (+ and − 0.2415 degrees) which correspond to the crusher eccentric throw dimensions. In addition, following a similar calculation process, the program develops two new sets of equations that describe the new mantle profile at these two extreme positions.

Step 10

The program applies the slicing technique for the bottom part of the crushing chamber, as described in section 5.3.2. For each slice, the program determines the coordinates of the original profiles and the measured profiles where they intersect a line through the midpoint of the slice. For the mantle, the process is repeated for the following three cases:

- maximum angular displacement,
- minimum angular displacement,
- zero degrees of displacement.

Using these results, calculations such as what follows are performed:

- the distance between the mantle, in its maximum angular displacement, and the concave for each slice,
- the minimum distance among the 80 slices (CSS),
- concave and mantle radial loss of material (wear) per slice, and,
- the volume of each slice formed by the mantle measured profile and concave measured profile.

**Step 11**

Although until step 10 all the calculations have been described for the mantle located at its bottommost position (0 mm), the program allows the input of different positions to recalculate all the results. In addition to the ability to determine important characteristics of the chamber for the measurement period, such as the approximate CSS dimension, the program is equipped with a macro that calculates a range of CSS corresponding to mantle positions varying from 0 to 254 mm (0 to 10 in.) with 12.7 mm (0.5 in.) increments.

**Step 12**

The program can simulate different configurations of the crushing chamber and calculate results for various operational options at the mine. For example, it is possible to assume the continuation of the concave and the replacement of the mantle with a new mantle with
a different profile by inputting the correspondent mantle information at step 7 and running the program again.
Appendix C  Example of a measurement drawing result
#4 Crusher Liner Profile
September 25/02

Eccentric Throat Smallest Concave Diameter When New:
- Feb 21 - 91.4", (Mar 21 - 92.1")
- Apr 18 - 93.1", (May 16 - 93.6")
- May 30 - 94.0", (June 21 - 94.9")
- July 25 - 95.0", (Aug 21 - 95.9")
- Sept 25 - 97.0"

Largest Mantle Diameter When New - 90"

Mantle: Installed September 4/02

Concaves: Penticton, High-Cr, Installed Feb 7/02

Sept 25/02 - Tonnes Crushed (approx): 0.89

Measure Date
- Mar 21 - 1.07 m³
- Apr 18 - 1.07 m³
- May 16 - 1.39 m³
- May 30 - 1.56 m³
- June 21 - 2.10 m³
- July 11 - 2.35 m³
- July 25 - 2.74 m³
- Aug 21 - 2.74 m³

Sept 25/02 Mantle removed - replaced with Esco 3-piece STD Concaves removed

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Appendix D  New mantle parts dimensions